

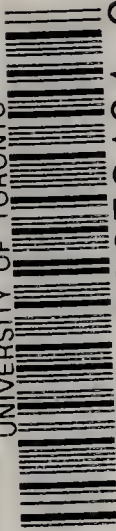
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PRACTICAL COAL-MINING:

*A MANUAL FOR MANAGERS, UNDER-MANAGERS,
COLLIERY ENGINEERS, AND OTHERS.*

BY

GEORGE L. KERR, M.E., M. INST. MIN. E.,
CERTIFICATED COLLIERY MANAGER.

WITH NUMEROUS PROBLEMS ARISING FROM COLLIERY WORK

AND

755 Figures and Diagrams.

SIXTH EDITION.

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NOTE TO FIFTH EDITION.

So great has been the demand for this book that the publishers printed twice the usual number of copies for the fourth edition, and this has now been out of print for some time. The delay in preparing this edition has been caused by pressure of other work, and by the increasing difficulty of compressing into the bounds designed for this book the large amount of new matter accumulated since the last revision. Attention has been specially given to the subjects of Timbering, Mine Gases, Safety Lamps, and Rescue Apparatus, the last having required considerable additions, and new illustrations have been freely introduced. The author has endeavoured to bring the book into line with the most recent legislation affecting coal mining, and in all other respects to fit it for continued usefulness in Schools and Colleges, as well as to the practical miner. Thanks are again given to the publishers for their unstinted efforts to keep the book in the first rank.

G. L. K.

GLASGOW, *February*, 1914.

Reprinted Jan. 1920.

PREFACE TO THE FIRST EDITION.

Not many years ago the works on Coal-mining were few, and in most instances so expensive as to be beyond the reach of the ordinary student or practical miner. This state of affairs has been to a large extent remedied of recent years by the issue of several works of more moderate dimensions and price. Between the small elementary text-

book and the still large and comparatively costly work of reference, however, there yet remains a considerable gap, which it has been the author's endeavour in this volume to fill. The best authorities have been freely consulted and, with due acknowledgment, laid under contribution; while the latest methods of working and the most modern machinery have been described, with the object of presenting an up-to-date account of the important industry under consideration.

Since the publication of the excellent treatise by Jonathan Hyslop, thirty years ago, no text-book dealing to any extent with Scottish practice has, so far as the author is aware, been published. As his experience has been gained largely in Scotland, he has attempted to remedy this omission, and it is to this that the occasional occurrence of a few Scotch words or phrases must be attributed, although these have been avoided, as far as possible, when reference is made to methods prevailing elsewhere. Less attention has been paid to literary style and elegance than to the production of a thoroughly practical and plainly worded text-book, designed not only to aid those endeavouring to qualify themselves for positions as colliery managers and other responsible officials, but also as a daily guide and reference-book for all engaged in and about the Colliery.

The author gratefully expresses his indebtedness to other sources, which, except in cases where a difficulty in tracing the authorship was experienced, he has acknowledged in the text. The works of Hughes on Coal-Mining, and of Foster on Ore and Stone Mining, have been quoted, and a few of the illustrations have been borrowed from those works. He desires to express his thanks to the publishers for the pains they have taken both as regards the text and the illustrations.

G. L. KERR.

BO'NESS, N.B., *September*, 1900.

CONTENTS.

CHAPTER I.

		PAGE
THE SOURCES AND NATURE OF COAL,		1-15
Introductory, ...	1	7
Definition of the term 'rock,'	1	8
Division of rocks, ...	1, 6, 7	8
Stratification, ...	2	8
Cleavage, ...	2	8-11
Inclination of strata, ...	2	13
Dislocations and faults, ...	4, 5	13-15
Division of coal-measures, ...	7	
Coal in other formations, ...		7
Formation of coal-fields, ...		8
Origin of coal, ...		8
Definition of coal, ...		8
Classification of coal, ...		8-11
Selection of coal, ...		13
Calorific power of coal, ...		13-15

CHAPTER II.

THE SEARCH FOR COAL,		16-38
Boring, ...	16	24, 30
Methods of boring, ...	17	25, 31
Chisels or bits, ...	17	25
Rods, ...	17	25
Bracehead, ...	18	26
Sludger, ...	18	28
Béche, ...	18	33
Brakestaff, ...	19	35
Preliminary operations, ...	19	37
Lining the bore-hole, ...	23	
Speed of boring, ...		24, 30
Cost of boring, ...		25, 31
Japanese method of boring, ...		25
Cost of boring plant, ...		25
Mather & Platt's system, ...		26
Diamond boring, ...		28
Davis calyx drill, ...		33
Other objects of boring, ...		35
Surveying bore-holes, ...		37

CHAPTER III.

SINKING,		39-100
Preliminary considerations, ...	39	69
Commencing operations, ...	42	73
Sinking and walling simultane- ously, ...	48	75
Tubbing, ...	53	80
Coffering, ...	55	81
Cost of sinking, ...	58	84
Sinking rectangular shafts, ...	59	85
Special methods, ...	66	86
Pile-driving, ...	66	89
Brick drums, ...	68	92
Iron cylinders, ...		69
Triger's method, ...		73
Kind-Chaudron method, ...		75
Lippman's method, ...		80
Pattberg system, ...		81
Poetsch's method, ...		84
Gobert's method, ...		85
Koch's system, ...		86
Accessories to sinking, ...		89
Ventilation, sinking pits, ...		92
Enlarging shafts, ...		93

CHAPTER IV.

	EXPLOSIVES,	PAGE
		101-119
Definition, ...	101	Detonators, ... 111
Classification, ...	101	Precautions against fire-damp
Gunpowder, ...	102	explosions, ... 113
Chlorate mixtures, ...	103	Safety fuses, ... 113
Nitro - compounds containing		Firing shots by electricity, . . 114
nitro-glycerine, ...	103	Simultaneous firing, ... 116
Permitted explosives, ...	105	Cost of blasting, ... 118
Nitro-compounds not containing		
nitro-glycerine, ...	105	

CHAPTER V.

MECHANICAL WEDGES, ROCK DRILLS, AND COAL-CUTTING MACHINES,		120-133
Mechanical wedges, ...	120	Darlington drill, ... 126
Elliott wedge, ...	120	Adelaide „ 127
Burnett's wedge, ...	120	Ingersoll „ 128
Hydraulic wedges, ...	121	Hammer drills, ... 128
Rock drills, ...	123	Flottmann drill, ... 130
Machine-power drills, ...	124	Hardy-Simplex drill, ... 133

CHAPTER VI.

COAL-CUTTING BY MACHINERY,		134-174
Introductory, ...	134	Ingersoll-Sergeant machine, . . 155
Classification of machines, ...	135	Harrison „ 157
Heading machines, ...	135	Morgan-Gardner „ 157
Disc machines, ...	137	Siskol „ 157
Gillot & Copley machine, ...	138	Choice of machine, ... 162
Rigg & Meiklejohn machine, ...	138	Conditions suitable for coal-
Clarke & Steavenson „	140	cutters, ... 163
Diamond „	142	Timbering at machine face, . . 164
Anderson-Boyes „	143	Labour for machine cutting, . . 164
Jeffrey „	144	Motive power for machines, . . 165
Chain „	145	Cost of machine installations, . . 166
Jeffrey chain „	145	Depth of cut, ... 167
Chain shearing „	149	Laying out face, ... 168
Bar machines, ...	149	Commencing cut, ... 169
“ Pick-Quick ” machine, ...	150	Cost of machine cutting, ... 170
Percussive machines, ...	152	Advantages of coal-cutters, . . 172

CHAPTER VII.

TRANSMISSION OF POWER,		175-208
Location of machinery, ...	175	Electric cables, ... 191
Steam, ...	175	Shaft cables, ... 193
Compressed air, ...	178	Fixing underground cables, . . 194
Valves for compressors, ...	185	Junction boxes, ... 195
Losses in air compressing, ...	185	Generators or dynamos, ... 197
Water pressure, ...	188	Motors, ... 198
Ropes and rods, ...	189	Electrical plant failures, . . . 198
Electricity, ...	190	Test of hauling engines, ... 199

			PAGE
Electric pumping, . . .	200	Cost of various systems, . . .	202
Advantages of electricity, . . .	202	Electrical terms, . . .	202
Disadvantages of electricity, . . .	202		

CHAPTER VIII.

MODES OF WORKING, 209-252

Choice of methods, . . .	209	Special methods, . . .	227
Longwall, . . .	209	Spontaneous combustion, . . .	232
Pillar and stall, . . .	218	Working highly inclined seams, . . .	234
Size of bottom pillar, . . .	218	Double and single stall, . . .	245
Width of stalls, . . .	222	Working thick seams, . . .	248
Extracting pillars, . . .	222	Order of working seams, . . .	251
Panel system, . . .	225	Loss in working seams, . . .	252

CHAPTER IX.

TIMBERING ROADWAYS, . . . 253-294

Necessity for timbering, . . .	253	Iron or steel supports, . . .	276
Timber, . . .	255	„ „ props, . . .	276
Methods of support, . . .	255	Steel or iron sets, . . .	277
Tapered props, . . .	257	Brick walls and girders, . . .	282
Systematic timbering, . . .	259	„ arching, . . .	287
Withdrawing timber, . . .	261	Strength of timber, . . .	287
Timbering drawing roads, . . .	263	Formulae for strength of timber, . . .	290
Driving through loose ground, . . .	269	Preservation of timber, . . .	292

CHAPTER X.

WINDING COAL, . . . 295-369

Preliminary, . . .	295	Cage guides, . . .	321
Pit-head frames, . . .	295	Cages, . . .	328
Iron or steel frames, . . .	300	Cage speeds, . . .	332
Winding engines, . . .	301	Pit-head pulleys, . . .	335
„ ropes, . . .	308	Drums, . . .	337
Iron ropes, . . .	308	Safety hooks, . . .	342
Care of ropes, . . .	310	„ cages, . . .	345
Rope cappings, . . .	312	Cage props or keps, . . .	347
Strength of ropes, . . .	314	Automatic apparatus, . . .	350
Counterbalancing, . . .	316	The visor, . . .	351
Conical drums, . . .	317	Electric winders, . . .	362
Koepé's system, . . .	319	Size of winding engine, . . .	366
Reducing strain on ropes, . . .	319		

CHAPTER XI.

HAULAGE, . . . 370-439

Classification of methods, . . .	370	Feeding of horses, . . .	379
Roads, . . .	370	Cost of horse haulage, . . .	382
Rails, . . .	370	Inclines, . . .	383
Laying roads, . . .	371	Machine haulage, . . .	392
Tubs, . . .	371	Cost of haulage, . . .	418
Arrangements of pit-bottom, . . .	375	Haulage problems, . . .	421
Haulage, . . .	376	Coal conveyors, . . .	424
Horse traction, . . .	377	Types of conveyors, . . .	426

CHAPTER XII.

	PUMPING,	PAGE
		440-513
Preliminary, ...	440	Pumping engines, ... 469
Pumps, ...	444	Pulsometer pumps, ... 489
Pump valves, ...	451	Centrifugal pumps, ... 491
Buckets, ...	453	Sinking pump, ... 501
Plungers, ...	453	Air vessels, ... 505
Pump-rods, ...	454	'Duty' of pumps, ... 506
Size of rods, ...	458	Pumping problems, ... 510
Plant required, ...	468	

CHAPTER XIII.

	VENTILATION,	PAGE
		514-625
Gases present in mines, ...	514	Speed of fans, ... 585
Carbon dioxide, ...	527	Cost of " 586
" monoxide, ...	536	Guiding air-currents, ... 592
Sulphuretted hydrogen, ...	539	Brattice, ... 595
Carburetted "	540	Laws of air-currents, ... 599
Natural ventilation, ...	554	Reducing friction, ... 603
Waterfall "	556	Instruments used, ... 609
Furnace "	556	Coal dust, ... 612
Mechanical "	563	Ventilation problems, ... 623
Fans, ...	563	

CHAPTER XIV.

	SAFETY LAMPS AND RESCUE APPARATUS,	PAGE
		626-698
Definition, ...	626	Electric lamps, ... 640
Davy lamp, ...	627	Construction of lamp, ... 653
Clanny lamp, ...	627	Lighting power, ... 653
Stephenson lamp, ...	628	Cost of upkeep, ... 654
Marsaut "	628	Protector lock, ... 655
Mueseler "	629	Testing lamps, ... 659
Gray "	630	Trimming and cleaning, ... 662
Wolf "	630	Filling and lighting, ... 663
Wolf-Dahlmann lamp, ...	632	Rescue apparatus, ... 666
Evan Evan's "	632	Proto " 670
Evan Thomas' "	633	Meco " 672
Fire-damp indicators, ...	636	Draeger " 676
Hydrogen indicator, ...	637	Weg " 679
Stokes's "	637	Ærophor " 683
Pieler lamp, ...	638	Reviving " 687
Chesneau lamp, ...	639	Instruction of corps, ... 693

CHAPTER XV.

	SURFACE ARRANGEMENTS, COAL CLEANING, ETC.,	PAGE
		699-730
Siding accommodation, ...	699	Coal cleaning, ... 713
Boilers, ...	701	Screening, ... 713
Banking out, ...	706	Jigging screens, ... 714
Tipplers, ...	708	Coal washing, ... 714
Travelling belts, ...	710	Varieties of washers, ... 717
Elevators, ...	712	Cost of washing, ... 730

CHAPTER XVI.

				PAGE
SURVEYING, LEVELLING, AND PLANS, .				731-765
Definition, . . .	731	Stepping the chain, . . .		745
Magnetic variations, . . .	732	Protractors, . . .		746
Dip, . . .	735	Plotting by co-ordinates, . . .		747
Determining the Meridian, . . .	736	„ chords, . . .		750
Setting out Meridian line, . . .	736	Calculation of areas, . . .		751
Colliery plans, . . .	737	Levelling, . . .		754
Underground surveying, . . .	739	Datum line, . . .		757
Vernier compass, . . .	741	Booking the levels, . . .		757
Measurements, . . .	743	Plotting levels, . . .		759
„ on inclines, . . .	744	Problems, . . .		759
INDEX, ...				766

PRACTICAL COAL-MINING.

CHAPTER I.

THE SOURCES AND NATURE OF COAL

Introductory.—The art of mining and science of geology are so closely related that it has become almost impossible to write a treatise on the former without referring to the latter. Text-books on coal-mining are therefore usually prefaced by a short introductory chapter on geology, and the present volume will, in this respect, conform to established custom.

The earth is composed of mineral matter in various combinations which are included under the general term of *rock*. It is an oblate spheroid in shape—that is, a sphere which has been flattened at the poles. It was at one time supposed to consist of a hard, solid, outer crust 10 or 12 miles thick, and an interior of molten material at a very high temperature. This theory was deduced from the increase in temperature observed in subterranean workings, and from the fact that molten lava is thrown out by volcanoes during eruption; but, according to Lord Kelvin, it is much more probable that the earth is a rigid mass from surface to centre with the properties of a solid.

Definition of the term Rock.*—“A rock may be defined as a mass of matter composed of one or more simple minerals, having usually a variable chemical composition with no necessarily symmetrical external form, and ranging in cohesion from mere loose débris to the most compact stone. Granite, sandstone, mud, peat, etc., are all recognised as rocks.”

Division of Rocks.—Rocks are divided into three classes, viz., aqueous, igneous, and metamorphic; or into two, stratified and unstratified.

Aqueous rocks are those which have been deposited where we now find them, by the agency of water. They are generally in layers or beds lying parallel to each other, and are often termed sedimentary

* *Text-book of Geology*, p. 57, by Sir Archibald Geikie.

rocks or deposits. *Igneous rocks* are those which have been subjected to the action of heat and retain no traces of stratification or bedding. *Metamorphic rocks* are rocks in which a crystalline rearrangement of the materials has taken place. They are sometimes called *altered rocks*. Marble is one of the best and most representative specimens of a rock of this class.

Aqueous or sedimentary rocks are deposited in definite layers or beds, this arrangement being termed *stratification*. When the deposits form very thin layers, such as occur in shale, they are said to be *laminated*.

Cleavage.—Cleavage is the term applied to the tendency of rocks and minerals to split along certain planes other than those of stratification, which occurs in stratified rocks and which tends to break the rock up into more or less cubical blocks. Generally when a rock is much intersected by cleavage planes, it loses its property of splitting along the bedding planes. Cleavage planes are said to be due to great pressure.

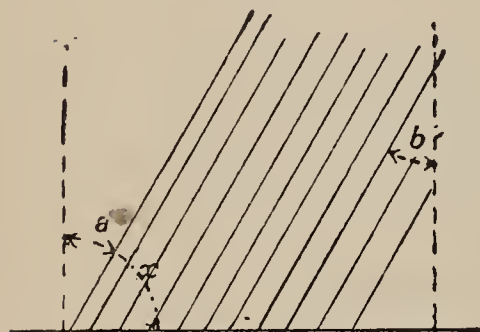


FIG. 1.

a = angle of dip ; b = hade.

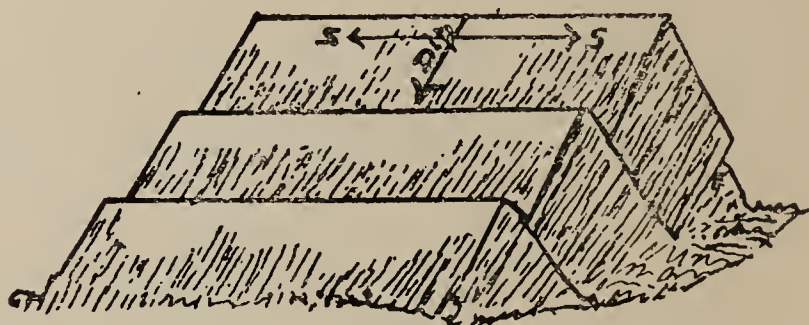


FIG. 1A.

Inclination of Strata.—The strata which compose the crust of the earth were no doubt deposited in horizontal layers ; but only limited areas are now found in that position. In all parts of the world beds of rock are usually inclined at a greater or less ‘angle of dip’ to the horizon, hence they usually come to the surface at some point, and when this happens it is termed the *outcrop* of the bed. The actual edge presented by a stratum at the surface of the ground is called the ‘outcrop.’ In flat, low-lying stretches of country few outcrops may be seen, while in hilly country, especially where the district is intersected by ravines and river-courses, the strata may be seen to outcrop frequently. It is in such positions that rocks can be most easily and advantageously studied.

Dip.—If the beds of rock, instead of being quite level, be inclined to the horizon, they are said to *dip* ; the point of the compass to which they are inclined is called the *direction of dip*, and the angle, or degree of deviation, which the strata make with the horizon is termed the *angle* or *amount of dip*, and may vary from 1° or 2° to 90° or vertical. The angle of dip is expressed in degrees, and is usually measured by an instrument called the clinometer.

Strike.—The prolongation of the strata in a line of bearing at a

right angle to the dip is called the strike (see figs. 1 and 1A). Thus, if the dip be due north and south, the strike will be due east and west. Sometimes the strike and outcrop coincide, as in the case of vertical beds, but more usually it varies with the stratigraphical contour of the beds.

Apparent Dip.—Any inclination of strata which is not at right angles to the strike is called the *apparent dip*; e.g. many strata of quarries, sea-cliffs, etc., have this apparent dip.

Direction of True Dip.—Sometimes the strata or a seam of coal may dip in two different directions, such as when an anticline or

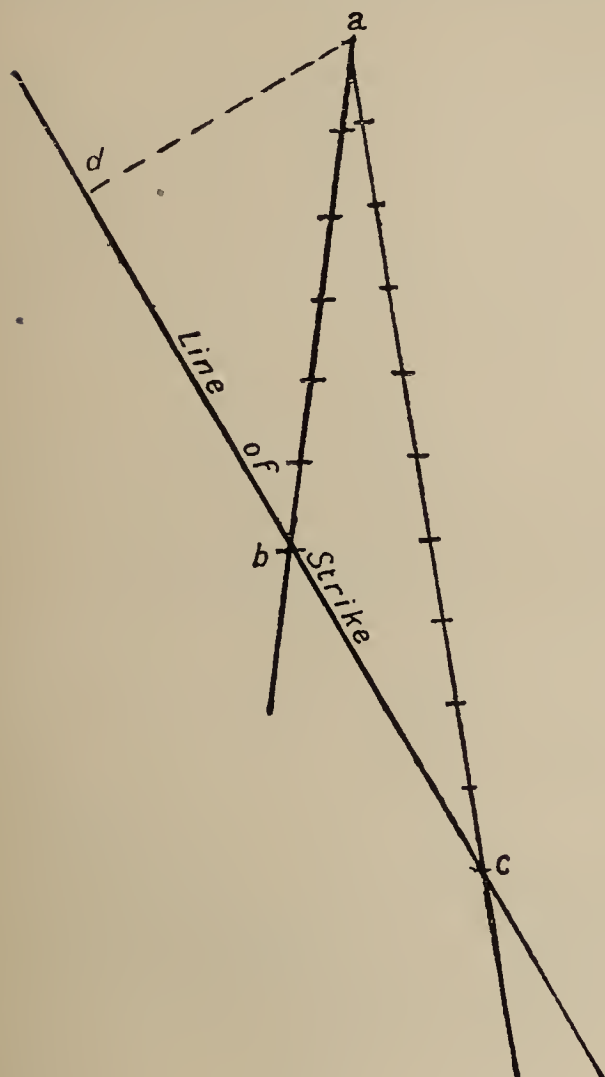


FIG. 2.



FIG. 3.

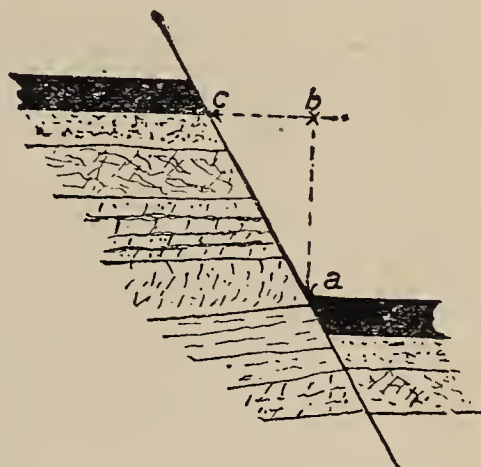


FIG. 4.

ab = vertical displacement.
 bc = lateral displacement.

syncline occurs. The strata will have a different apparent dip from either side of the axis, and in an occurrence of this kind it is sometimes useful to ascertain the true dip. This may be done graphically as shown in fig. 2. Draw to scale the line ab and the line ac with apparent dips of 1 in 6 and 1 in 10 respectively. Measure off along ab six equal units of length and along ac ten equal units. Join bc ; this line will now represent the line of strike. At right angles to bc draw the line ad , which will be the direction of true dip. The amount of true dip is 1 in ad .

Anticline and Syncline.—In many parts of the world the strata, instead of being found in a horizontal or regularly inclined position,

are contorted and bent into folds. The French and Belgian coal-fields furnish notable examples of such distortion. Where the strata dip away from an axis so as to form an arch or saddle, they are termed an *anticline* or anticlinal axis. Where the strata dip towards an axis, forming a trough or basin, it is called a *syncline* or synclinal axis (see fig. 3).

Dislocations, Faults, and Dykes. — When strata have been fractured and displaced or shifted along the line of breakage, it is said to be 'faulted,' and the fissure or break along which the displacement occurs is termed a 'fault,' 'dislocation,' or 'throw.' Faults may occur with or without distortion of the fractured rock masses. In the largest proportion of cases there are both fracture and displacement in the beds, the rents becoming both 'fissures' and 'faults' (see fig. 4). Faults may vary in width from mere sharply defined lines with very little displacement up to gaps of many yards in width and a hundred yards or more of displacement. All the British coal-fields are traversed by many of these faults, the main faults in nearly all cases running almost due east and west. Some-

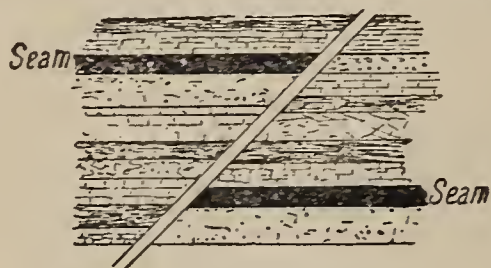


FIG. 5.—Reversed fault.

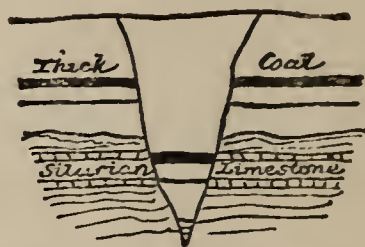


FIG. 6.—Trough fault.

times large faults split up and branch off into a number of smaller faults which gradually thin or die out. Again, faults may only dislocate the strata in one horizon. For instance, a seam of coal may be much faulted, and another, some distance further down, may not be affected at all. This is well illustrated at a colliery near Glasgow, where one seam, 180 yds. from the surface, is much broken up by faults and large areas become very thin, while another seam, 200 yds. further down, is not affected at all by faults and maintains its thickness with great regularity. The angle which a fault makes with the vertical is termed the *hade*, and the line of fracture the *vees* of a fault. The vertical displacement of strata by a fault is termed the *amount of throw* or *throw of a fault* (fig. 4). Faults are sometimes vertical, but are generally inclined. The largest faults, *i.e.* those which have the greatest vertical displacement, commonly slope at high angles. Those of only a few feet displacement may be inclined at angles as small as 18° or 20° from the horizon, but this is exceptional. In addition to vertical displacement faults usually also cause a lateral displacement (*b c*, fig. 4).

A fault is termed a down-thrown fault if the observer is looking from the higher to the lower level of displacement; and an up-throw fault if in the opposite direction. Faults may be either (1) *Normal*;

(2) *Reversed* (fig. 5); (3) *Overlap*; (4) *Trough* (fig. 6); or (5) *Step Faults*. When faults lie at a high angle, 70° or 80° from the horizontal, and have a definite line of displacement, and when the hade is in the direction of the down-throw, it is called a *normal* or *ordinary* fault (fig. 4). When parallel faults run in the direction of the strike, and have their down-throw with the dip, this tends to prevent certain outcrops from appearing. On the other hand, a succession of step faults, with down-throws against the dip, may cause the same beds to crop out again and again, and hence be mistaken for a number of different seams (fig. 7). When a bed has, from some cause, had a portion denuded or worn away, it is known as a *dumb fault*, or *wash-out* (fig. 8). The action of water is the most frequent cause of a wash-out.

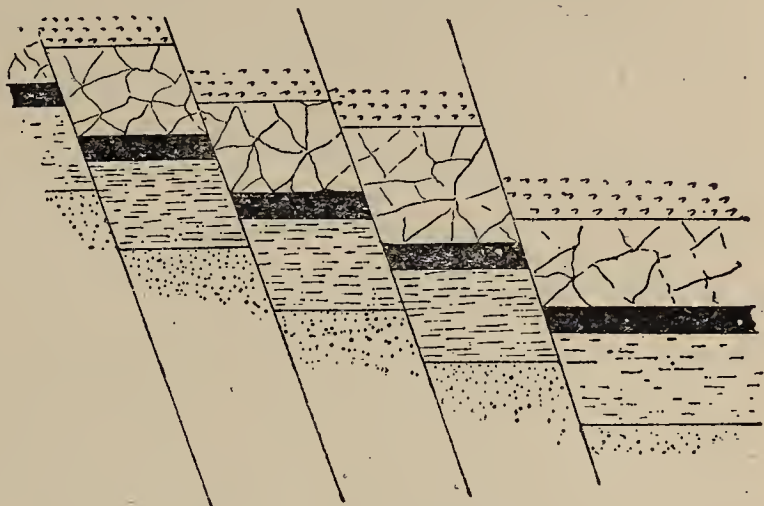


FIG. 7.—Step faults.

The term *dyke* is often confused with the term *fault* by miners, and taken to mean the same thing.

When a fault occurs there is displacement of the strata; but a dyke is usually unaccompanied by any displacement.

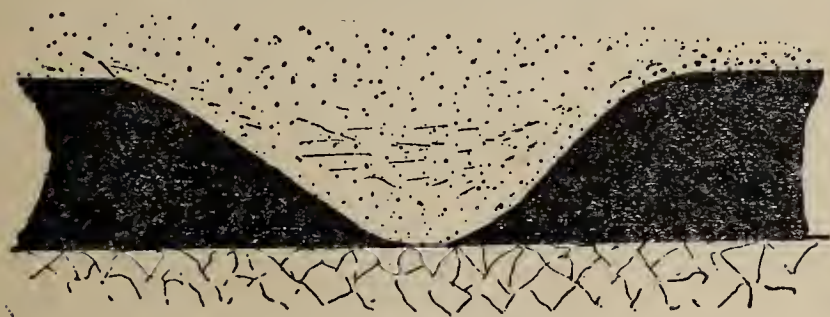


FIG. 8.—Wash-out.

Dykes are wall-like masses of rock which traverse strata in succession from unknown depths and appear in many instances at the surface. They usually consist of basalt or allied rock, and are of volcanic origin, having

apparently been 'intruded' while in a liquid state into fractures. Dykes affect very materially the quality of the coal intersected by them, which is 'burnt' into a soft, cindery, and sooty state, or altered into a hard and incombustible substance. The distance that the coal-seam is affected on either side of a dyke is usually about two-thirds of its width, but in some cases it is greater. For instance, in a colliery in Lanarkshire a whin dyke 100 ft. thick was met with, and it was found that the coal was burnt for a distance of 100 ft. on either side of it. Usually, however, the thickness of the burnt coal is in direct proportion to the thickness of the intrusive dyke. Dykes vary in width from less than a foot to 70 ft. and upwards.

It is not unusual for them to run in nearly straight courses for many miles. Sometimes they occur along the lines of a fault, but very often they are unconnected with faults in any way. Frequently they are found to cross faults without being in the least deflected out of their course thereby.

Calcareous Coal.—Coal which is sometimes designated as ‘*burnt*’ coal may be something different altogether.* In a colliery in Lanarkshire large areas of coal were described on the colliery plans as burnt coal, but the fact that no intrusive whin was found in the district led to more careful investigation. Samples of the coal were analysed, and it was found that its foulness was due not to effects of burning, but to the presence of carbonate of lime (32 per cent.), carbonate of magnesia (15·5 per cent.), ferric oxide (3·28 per cent.), alumina (8·39 per cent.), and smaller proportions of alkalies, silica, and sulphur, with 32·7 per cent. of carbonaceous matter. When a piece was coked it was ascertained that the amount was 42 per cent. In some parts of the area 6 or 8 in. only of the top part of the seam was affected; in other parts a portion at the bottom of the seam was altered; and in some parts the whole thickness of the seam was affected and rendered useless. Mr R. W. Dron, the agent at the colliery, who investigated this phenomena, states that it is difficult to formulate a theory to account for this alteration in the coal. No seams of limestone are found in the strata overlying the seams, but there are several beds of ironstone. He is of opinion that the occurrence of this calcareous coal is due to lime having been deposited in the coal by infiltration from above at a period subsequent to the depositions of the coal-seams. Mr Dron noticed that the alteration in the coal was generally greater in the vicinity of faults.

Division of Rocks into Groups.—The rocks forming the crust of the earth in the British Isles have been divided into five main groups. They are disposed in the following order:—

Quaternary or Post-Tertiary	{ Recent and Pre-historic. Pleistocene.
Tertiary or Cainozoic	{ Pliocene. Miocene.
	{ Oligocene. Eocene.
Secondary or Mesozoic	{ Cretaceous. Jurassic. Triassic.
	{ Permian. Carboniferous.
Primary or Palæozoic	{ Devonian and Old Red Sandstone. Silurian. Cambrian.
	{ Primitive Schists.
Archæan or Azoic	{ Gneiss and other Crystalline Rocks.

* “Calcareous Coal in a Lanarkshire Colliery,” by R. W. Dron, *Trans. Inst. Min. Engs.*, vol. xxvii. p. 92.

Coal-measures.—The formation which has the greatest interest for the coal-miner in this country is the Carboniferous, for it is in this formation that coal is found most abundantly. It consists of three divisions of strata, viz. :—Coal-measures, Millstone Grit, and Carboniferous Limestone.

The upper division or Coal-measures are the strata where coal-seams are most abundantly found. In the English coal-fields, very few workable seams are found below these measures. The millstone grit is usually composed of coarse yellow sandstones, flagstones, shales, and a few thin seams of coal. In the Scotch coal-fields valuable seams of coal and ironstone are found below the millstone grit. In the Carboniferous formation of Scotland, the following four divisions are made:—

Coal-measures	{	Consisting of (a) an upper series of red and purple sandstones and shales enclosing thin coal-seams, and (in Ayrshire and Fife-shire) thin limestone bands ; (b) the productive coal-measures, consisting of white and grey sandstones, shales, coals, fireclays, and ironstones, but no limestone.
Millstone Grit or Moor Rock	{	Coarse thick sandstones with shales, fireclays, thin seams of coal, clayband ironstone, and, occasionally, beds of limestone.
Carboniferous Limestone	{	This series is often subdivided into (a) thick sandstones and beds of shale with three varieties of limestones and some coals ; (b) a group of ordinary coal-measures very similar to the upper coal-measures, and containing valuable seams of coal and ironstone, but no limestone ; (c) sandstones, limestones, shales, with some coals and ironstones.
Calceiferous Sandstone	{	This series may be subdivided into (a) the upper group, consisting of sandstones, shales, oil shales, some very inferior coals, ironstones, and limestones ; (b) the beds lying below, extending to the base of the carboniferous rocks.

It will be seen from the above that the coal-measures in Scotland differ greatly from those of England, inasmuch as a large proportion of the coal in the Scotch coal-fields is found below the millstone grit, while in the English coal-fields very few valuable seams are met with below that formation.

Coal found in other Formations.—It has been shown that coal is most abundantly found in the Carboniferous strata, but it is not entirely confined to that formation, being often found in others—although such coal is seldom of much value compared with that of the coal-measures—both above and below the Carboniferous formation. In New South Wales coal is got from the Devonian series of rocks ; at Bonn, in Germany, and at Bovey Tracey, in Devonshire, the coal-beds are, presumably, of Miocene age ; the brown coals of New Zealand and Australia are believed to be late Tertiary deposits, while at Brora, in the North of Scotland, coal is found in the Lower Oolite formation. Nearly all these are, however, lignites as distinguished from the true coals found in the Carboniferous formation.

Rocks and Minerals associated with Coal.—Coal when found is

generally associated with sandstones, shale or blaes—known amongst miners as bind—limestone, fireclay, ganister, ironstones, iron pyrites, and in Scotland, oil-shales. Most coal-seams rest on a bed of fire-clay; in some districts this under-bed takes the form of ganister.

Formation of Coal-fields.—It is a noticeable characteristic of coal-fields that they take the form of a basin, dipping from all sides towards a central axis. Hence we get the seams cropping out frequently at the surface, which allows of large areas being easily reached and worked. Were it not that coal-fields assume this shape a large part of our coal supply would inevitably be found at too great a depth to be workable.

Origin of Coal.—Regarding the origin of coal numerous theories are held, one being that it was formed in the position in which we now find it, and is the product of vast forests which grew, flourished, and decayed on the site of our present coal-fields. Another is that it was brought into its present position by what is termed the ‘drift’ process, the forests being supposed to have flourished and decayed in one part of the world, while the accumulated ‘humus’ was swept into its present position by the agency of water or ice. While the ‘drift’ theory may explain the origin of a few isolated deposits, there is little doubt that the first-mentioned theory is correct and the one now most generally held by geologists.

Definition of Coal. — Coal is a substance which it is easier to recognise than to define. Nearly everybody is familiar with the appearance and uses of this common mineral, but its definition is attended with several difficulties.

Dr Percy defines it as: “A solid, stratified, mineral, combustible substance, varying from dark brown to black, opaque, except in extremely thin slices, brittle, not fusible without decomposition.”

Sir Archibald Geikie defines coal as: “A compact, brittle, velvet-black to pitch-black, iron-black, or dull, sometimes brownish rock, with a greyish-black or brown streak, and in some varieties a distinctly cubical cleavage, in others a conchoidal fracture. It contains from 75 to 90 per cent. of carbon and a small percentage of sulphur generally combined with iron. It has a specific gravity of 1·2 to 1·35, and burns with comparative readiness, giving a clear flame and a strong aromatic or bituminous smell, some varieties fusing and caking into cinder, others burning away to a mere white or red ash.” Or more shortly: “Coal is composed of compressed and mineralised vegetation.”

Classification of Coal.—The varieties of coal may be classified as—

- (1) Anthracite, or smokeless coal.
- (2) Steam, free burning, or dry coal.
- (3) Bituminous, or caking coal.
- (4) Cannel, parrot, or gas coal, including the Boghead variety sometimes called Torbanite.
- (5) Lignite, or brown coal.

Other classifications are given, the coals being divided according to the percentage of carbon which they contain; the two following of which are examples:—

	Relative Percentage.	
	Fixed Carbon.	Volatile Hydrocarbon.
I. Gaseous, highly bituminous, or fat coals, ...	50 to 60	50 to 60
II. Bituminous { House, manu- facturing, 2nd class gas, } { 2nd class steam coal, }	60 to 82	40 to 18
III. Semi-bituminous, steam, or dry coals, ...	82 to 88	18 to 12
IV. Anthracite, ranging between semi-anthracite and pure anthracite, ...	88 to 96	12 to 4

Professor Frazer's Classification.

	Ratio $\frac{C}{\text{Vol. H.C.}} : 1.$
(a) Bituminous, ...	from 5 : 1 to 10 : 1
(b) Semi-bituminous, ...	„ 9 : 1 „ 5 : 1
(c) Semi-anthracite, ...	„ 12 : 1 „ 8 : 1
(d) Hard dry anthracite, ...	„ 5 : 1 „ 10 : 1

Coals are also sometimes classed as High Quality and Low Quality coal, as shown by the following analyses * :—

	HIGH QUALITY COALS.			LOW QUALITY COALS.		
	Anthra- cite.	Semi- bitumin- ous.	Bitumin- ous.	Anthra- cite.	Semi- bitumin- ous.	Bitumin- ous.
Fixed carbon, ...	88·5	75·5	53·5	75·0	67·0	46·5
Volatile matter, ...	5·0	18·0	40·0	5·0	15·0	34·0
Sulphur, ...	0·5	0·5	0·5	2·0	3·0	3·5
Ash, ...	5·0	5·0	5·0	16·0	12·0	12·0
Water, ...	1·0	1·0	1·0	2·0	3·0	4·0

All these varieties have had a common origin; they are all accumulations of ancient vegetation which has undergone chemical change under certain conditions. In the 'lignite' or 'brown' coal this change has been less complete than in the others.

Anthracite, sometimes also called 'blind-' and 'stone-' coal, has usually a brilliant black lustre, breaks with a conchoidal fracture, and does not soil the fingers when handled. It has been supposed that it has, at some period, undergone a sort of natural coking process, under the influence of subterranean heat, and that this has

* *Practical Engineers' Pocket Book*, 1897, p. 324.

driven off a large proportion of the hydrogen, oxygen, and nitrogen it originally contained. Anthracite gives off little or no smoke and is difficult to ignite, but when burning gives out intense heat. It consists almost entirely of carbon, the best qualities containing 90 to 95 per cent. with 5 to 10 per cent. of hydrogen, oxygen, and nitrogen.

Steam coal closely resembles bituminous coal, from which it differs only in being slightly harder, lighter, and more compact. It does not cake when heated, however, and it is practically smokeless. Its specific gravity varies from 1.27 to 1.30. On analysis the best quality yields approximately 89 per cent. of carbon, 4.5 per cent. of hydrogen, 3 per cent. of oxygen and nitrogen, and 3.5 per cent. of ash.

Average Analysis of a Middle Class Steam Coal.—A sample of steam coal from the Arniston Collieries, Gorebridge, Midlothian, was analysed with the following results :—

Composition per cent.				Composition per cent. exclusive of Sulphur, Ash, and Water.
Carbon,	.	.	73.13	84.66
Hydrogen,	.	.	4.96	5.74
Oxygen,	.	.	6.93	8.02
Nitrogen,	.	.	1.36	1.58
Sulphur,	.	.	1.08	
Ash,	.	.	2.72	
Water,	.	.	9.82	
100.00				100.00

Remarks.—Specific gravity, 1.265; water at 60° Fahr. being 1.000. Colour of ash, pinkish. Nature of coal, caking. Coke, 63.60 per cent.; small bulk, dull in appearance, but fairly hard and coherent. Total heat units (Favre and Silverman), 7269. Calorific power (Thompson), 7029 calories. Evaporative power (determined by Thompson’s calorimeter), 13.09 lbs. of water per lb. of coal.

Bituminous coal, also known as ‘free burning,’ ‘smoking,’ or ‘flaming coal,’ when ignited, burns readily with a yellow flame, giving off smoke freely. On heating it swells into a pasty, bitumen-like mass which ultimately becomes solid. Bituminous coals are misnamed, as they contain no true bitumen. There are several varieties of bituminous coal, which are distinguished according to their mode of burning, which depends chiefly on the relative proportions of carbon, oxygen, and hydrogen they contain. Steam coal approaches anthracite in its properties. Dry or non-caking coal is another variety; it does not possess the property of caking which makes coal so valuable for household purposes. Non-caking coals are generally hard and compact, and when in a fine powdery state do not cohere when heated. The following analyses show the composition of the two varieties of good burning coal :—

Caking Coal.		Non-caking Coal.	
Carbon,	75 per cent.	76	per cent.
Hydrogen,	4 „	4·3	„
Oxygen,	16 „	16	„
Ash,	3 „	3·3	„
Water,	6·3 „	5·4	„

Cannel is generally classed as a variety of bituminous coal, although it is not a true coal, as it contains a certain amount of argillaceous matter, and sometimes even passes into shale or iron-stone. *Cannel* or *gas coal* differs a good deal in appearance from ordinary bituminous coal, being of a dull, lustreless, black colour, not splitting readily into thin layers, and generally devoid of vegetable structure under the microscope. The best qualities of *cannel* are of a tough nature and can be cut readily with a knife; ornaments are frequently made from *cannel* of this kind. An analysis showed—

Volatile matter (containing ·58 of Sulphur),	.	.	40·28 per cent.
Coke, consisting of { Carbon, 49·40	}	.	.
Sulphur, 0·29			
Ash, 6·53			
Water, expelled at 212° Fahr.,	.	.	3·50 „
			100·00

Cannel coal contains a comparatively large percentage of oxygen and hydrogen, and it is therefore valuable for the manufacture of coal gas or paraffin oil, and is only distinguished from the bituminous shales now so extensively used in the manufacture of paraffin by the much smaller proportion of ash which it contains. The yield of gas from *cannel* coal varies from 10,500 to 13,500 cub. ft. per ton, and from bituminous coal, from 9000 to 10,000 cub. ft.

The following is an analysis of a first-class *cannel* coal:—

Mineralogical Characters.—The coal is brownish black, possesses moderate lustre and yellowish-brown streak; fracture irregular, inclining to slaty, with impressions of stigmata; cross-fracture large conchoidal and partly hackly, with slight deposits of calcic carbonate and ferric bisulphide in the natural partings; compact and cohesive; under distillation it partly and slightly intumesces. Colour of ash, brownish white. Thickness of seam, 10 in. Mean specific gravity, 1307 (water 1000). Weight of 1 cub. ft., 81·68 lbs.

CHEMICAL ANALYSIS.

Volatile matters (containing ·44 of Sulphur),	.	.	46·27 per cent.
Coke, consisting of { Carbon, 43·44	}	.	.
Sulphur, ·13			
Ash, 7·82			
Water, expelled at 212° Fahr.,	.	.	2·34 „
			100·00

PRACTICAL RESULTS.

Gaseous Products.

Gas per ton of coal at 60° Fahr., and 30 in. bar., ...	13,640 cub. ft.
Gas from 1 cub. ft. of the coal, ...	497·37 „
Specific gravity of the gas, ...	617 (air 1000).
Hydrocarbons absorbed by bromine, ...	13·25 per cent.
Durability of 1 cub. ft. by 5-in. jet flame, ...	68 min. 18 sec.
Value of 1 cub. ft. of gas in sperm, ...	848·16 grains.
Value of gas from 1 ton of coal in sperm, ...	1681·27 lbs.
Illuminating power of gas in standard candles, ...	35·34 candles.
Sulphuretted hydrogen (H ₂ S) in foul gas, ...	1·25 per cent.
Carbonic acid (CO ₂) in foul gas, ...	2·20 „
Carbonic oxide (CO) in foul gas, ...	11·75 „
Sulphur eliminated with volatile products, ...	9·85 lbs.

Liquid Products.

Tar per ton of coal, ...	26·40 gallons.
Ammoniacal liquor, per ton of coal, ...	9·20 „
Strength of ammoniacal liquor, ...	3·75° Twadd.
Hygrometric water, per ton of coal, ...	5·24 gallons.
Aqueous absorbent capacity of coal (<i>determined by complete saturation</i>), ...	2·76 per cent.

Solid Products.

Coke, per ton of coal, ...	1151·13 lbs.
Carbon in the coke, ...	84·80 per cent.
Ash in the coke, ...	15·20 „
Sulphur in coke, per ton of coal, ...	2·91 lbs.
Heating power of 1 lb. of coke (<i>water from boiling point into steam</i>), ...	11·65 „

Remarks.—This is a very rich cannel coal, and therefore one of great value for ordinary coal-gas enrichment; under distillation it parts with its volatile products very speedily, yielding a large volume of 35·34 candle gas, and at the same time affords 10·27 cwts. per ton of coke of medium quality. The foul gas moreover contains a very small percentage of impurities. Compared with main Lesmahagow cannel coal, represented by 100 (calculated on the basis of a production of 13,000 cub. ft. of gas, and 1535·5 lbs. of sperm value per ton, and having regard also to the value of the secondary products, and the cost of the purification of the gas), this coal is equal to 110·96.

Boghead, or *torbanite*, is a mineral occurring at Boghead, near Bathgate, in Scotland. It has long since been practically exhausted. The mineral was brownish-black, and had a specific gravity of about 1·15. It contained 63 per cent. of carbon, 9 per cent. of hydrogen, 20 per cent. of ash, and 8 per cent. of oxygen and hydrogen. It yielded 15,000 cub. ft. of gas, and about 70 to 80 gallons of oil per ton.

Lignite or Brown Coal.—‘Lignite’ or ‘brown’ coal is the term usually applied to deposits of more recent origin than coals found in the carboniferous formation, to which formation true coal belongs. Lignites vary in colour from a light earthy brown to a deep lustrous black, undistinguishable from ordinary bituminous coal. They contain 50 to 70 per cent. of carbon.

In New Zealand the coal worked is of the lignite variety, and is not of a very high quality.

Brown coals proper usually contain a larger percentage of carbon and a smaller percentage of oxygen than the true lignites.

The general composition of lignites and brown coal may be seen from the following analysis:—

	Carbon.	Hydrogen.	Oxygen and Nitrogen.
Lignite from Bovey Tracey,	67·9	5·8	26·3
„ „ Cologne, .	67·0	5·3	27·7
Brown coal from Hungary, .	72·5	5·4	22·1
„ „ „ Tasmania,	71·9	5·6	22·5
„ „ „ Auckland,	72·2	5·4	22·4

The following table shows the various changes through which coal passes during its transition from wood to anthracite:—

	Weight of 1 cubic foot in lbs.	Carbon per cent.	Hydrogen per cent.	Oxygen and Nitrogen per cent.
Wood, average, ...	30	50·29	6·09	43·62
Peat, „	50	60·83	5·89	33·28
Lignite, „	70	67·43	5·59	26·98
Brown coal, average, . .	75	72·92	5·4	21·58
Bituminous coal, average, .	80	83·48	5·34	11·18
Anthracite, average, . .	90	95·35	2·47	2·18

Selection of Coal.—Dr Percy says that the only sure guide in the selection of coal for any purpose is to make a practical trial on a large scale. A good deal of information may, however, be obtained from reliable chemical analysis, but as a rule the thermal value of a fuel, as determined by a physical test, is never even approximately realised in practice. An anthracite coal with no gas or flame may be suitable for one purpose, while a bituminous coal full of rich smoky gas may be most economical under other conditions. In any fuel large quantities of ash are objectionable, as they reduce the quantity of available combustible material per ton of fuel, and increase labour in handling both fuel and ash, while the fires require more frequent cleaning, which entails a reduction in the efficiency of the boiler by the chilling influence of the cold air admitted during the process.

A large percentage of moisture is also objectionable, as a portion of the calorific power of the coal is unproductively expended in evaporating the combined water.

Calorific Power of Coal.—Modern requirements now demand the

most economic generation of heat for a given expenditure of fuel, no matter to what purpose the coal is put. As already stated, the thermal value of fuel is not easily ascertained with any high degree of accuracy by chemical or physical tests, and only approximate values can be looked for.

The simplest method of finding the calorific power is by using the instrument known as Thompson's calorimeter, which is largely adopted for this purpose. The principles upon which the test is based are :— (1) That the latent heat of steam is equal to 967° Fahr., and (2) that coal or other fuel burned in pure oxygen evolves the same amount of heat as when completely consumed in atmospheric air. The test is carried out as follows: A measured weight of fuel is dried, finely powdered, and intimately mixed with the necessary propor-

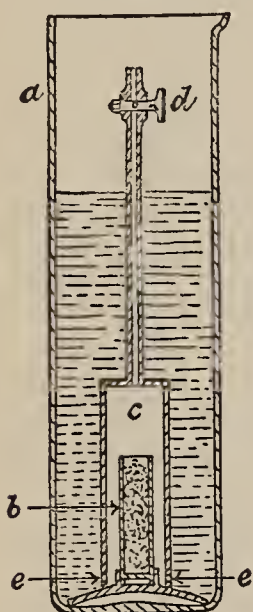


FIG. 9.—Thompson's calorimeter.

tions of a mixture consisting of three parts of potassium chlorate and one part of potassium nitrate. This mixture, which will burn freely without a supply of air, is placed in a copper cylinder *b* (fig. 9), which is primed with a fuse. This cylinder is placed within the copper combustion vessel *c*, and is then immersed in a glass jar *a* containing a known weight of water. The fuse, in the small cylinder *b* containing the chlorate mixture, is lighted, and the appliance is plunged into the glass cylinder containing the water, and is covered by the second copper cylinder *c*, the cock at *d* being shut. After a few seconds the fuse ignites the mixture of coal and potash, and the products of combustion, passing through the water in a finely divided state, communicate the whole of their heat to the water.

The temperature of the latter is carefully noted at the commencement and end of the test, and it is only necessary to multiply the weight of water by the number of degrees of heat communicated to it to find the calorific value of the fuel. The amount of water capable of being converted into steam per lb. of fuel burnt is directly as the elevation of the temperature; thus, if the thermometer showed a rise of 7·5°, then *one pound of fuel would evaporate 7·5 lbs. of water*. Tables to facilitate calculation are supplied with each instrument.

The theoretical evaporative power of fuel may also be calculated from the ultimate analyses by the formula :—

$$P = 15 \left\{ C + 4 \cdot 28 \left(H - \frac{O}{8} \right) \right\}$$

Where C = weight of carbon in 1 lb. of fuel.

H = „ hydrogen „

O = „ oxygen „

P = lbs. of water at 212° F., converted into steam at 212° F. per lb. of fuel.

As an example, in the analyses of the caking coal already given the carbon was 75 per cent., the hydrogen 4 per cent., and the oxygen 16 per cent. These would be in the ratio of .75, .04, and .16; then by applying the above formula we have—

$$P = 15 \left\{ .75 + 4.28 \left(.04 - \frac{.16}{8} \right) \right\} = 15 \{ .75 + (4.28 \times .02) \} = 15 \times .8356 = 12.53 \text{ lbs.}$$

The following table shows the calorific power of coal from widely different sources * :—

Locality.	Nature of Coal.	Calorific Power of Dry Coal free from Ash (in British thermal units).
Toula, Russia, . . .	Lignite,	13,837
Manosque, Basses Alpes, . . .	„	12,584
„ „ „ „ . . .	„	13,253
France and Germany, . . .	Brown Coal,	11,340–14,220
England, ...	Caking Coal,	15,804
„ „ „ „ . . .	„	16,108
Basin of Donetz, Russia, . . .	„	15,651
Creusot, France, ...	„	17,319
„ „ „ „ . . .	Anthracite,	17,021
Basin of Donetz, Russia, . . .	„	14,866

From recent experiments made on Scotch coals the calorific power and specific gravity are shown in the following table :—

	Specific Gravity.	Calorific Value (B.T.U.)
Ell, ...	1.266	13,464
Main, ...	1.261	13,662
Splint, ...	1.292	13,365
Gas, ...	1.290	13,266
Virgin, ...	1.286	13,464
Kilsyth Haughrigg, ...	1.291	13,563
Bannockburn Main, ...	1.306	14,157
Kilsyth Coking, ...	1.275	14,058

The calorific value of a pound of fuel in B.T.U. can be calculated from the formula :

$$x = 145C + 620(H - \frac{1}{8}O),$$

where C, H, and O represent the percentages of carbon, hydrogen, and oxygen present as determined by analysis.

* *Coal, its History and Uses*, p. 250.

CHAPTER II.

THE SEARCH FOR COAL.

Boring.—The search for coal in an unknown district is the application of geology to practical uses. In such a search all available means are taken to obtain information, such as the examination of quarries, beds of rivers, and railway cuttings. Even the ploughing of fields has often led to the discovery of the presence of coal when there were no other indications.

An examination carefully carried out in any district will reveal whether the strata belong to the coal-bearing formation or not. The discovery of a few fossils, such as sigillaria or stigmaria, will at once identify the rocks; or an 'outcrop' of coal may be discovered at the surface, but this is not often the case, particularly if the coal-bearing strata have been deeply overlaid by newer formations. In such circumstances resort must be had to boring to decide whether coal be present or not.

Boring is the means adopted to determine the existence of beds of minerals, such as ironstone, coal, and salt, lying below the surface of the earth, and to obtain information respecting their position, thickness, and quality.

The uses of bore-holes vary considerably, but in coal-mining boring is usually undertaken to supply the following information:—

- (1) To obtain a correct section of the strata passed through.
- (2) To ascertain the exact depth of the seam or seams from the surface.
- (3) To ascertain the thickness, quality, and number of seams.
- (4) To find out the nature of the roof and floor of the seam or seams.
- (5) To determine the inclination of the strata, and, if possible, the number and size of faults or dislocations traversing the area to be worked.

In establishing the existence of dykes or faults underground, bore-holes sometimes save time and money which might otherwise be wasted in exploring by means of shafts, particularly when the 'vees' of the fault is nearly vertical or ill-defined, and it is difficult to determine whether it is an up-throw or a down-throw fault. When conditions like these are met with the usual practice is to drive the roadway a short distance through the faulted metals, and

then to put a bore-hole up from the roof and another down from the floor. The strata ascertained by these bore-holes is then compared with the strata in the roof and floor sections prior to the fault being met with. This will usually lead to a determination as to whether the seam has been displaced by an up-throw or a down-throw fault.

By their aid the gradient of a road that would intersect the dislocated seam can also be determined.

Methods of Boring.—There are two chief methods of boring, viz. percussive and rotary. These may be again subdivided thus :—

- | | | | |
|-----------------------|---|--|---|
| (1) Percussive Boring | { | With Rods. | { Ordinary method of chipping and removing débris.
Japanese method of 'plunging' without removing débris. |
| | | With Ropes. | { Chinese and other methods with a spring pole.
Ordinary method employed in American oil districts.
Special methods, such as Mather & Platt's and others. |
| (2) Rotary Boring | { | Boring with augers in soft material.
Diamond rock-drill boring. | |

Hydraulic methods are applied to both systems of boring.

The percussive method is largely adopted for shallow bore-holes, or in soft, easily worked rock. It is commonly carried out by means of free-falling tools, which chip or cut the rock into angular fragments. The rotary method grinds the rock into powder, or can be made to cut out a solid core. The commonest method of boring, for depths of 5 fms. or more, is carried out by means of a steel chisel screwed into an iron rod, and suspended from a spring-pole. The tools used for a bore-hole in ordinary strata are chisels, rods, bracehead, augers, and sludgers for extracting the loose material; tools for dressing the sides of the bore-hole and for extracting broken rods or chisels: also keys for screwing and holding the rods, and tubes for lining the holes.

Chisels or Bits.—The form, sharpness, and temper of the cutting tool employed vary according to the rock which has to be cut through. Various chisels are in use: flat or straight-edged for ordinary strata; V or diamond-point chisels for hard rock; the T chisel for gravel; while others, with cutting edges shaped like an S or Z, are used for different kinds of work, but these chisels are difficult to sharpen and maintain in good order. For soft ground, such as clay or peat, augers are used. The chisels are 18 in. to 24 in. long, 1 in. to 2 in. diameter, and 2 in. to 3 or 4 in. in breadth of face. They are made of the best steel, and weigh from 3 to 4½ lbs. each. Fig. 10 shows some of the forms used.

Rods.—The rods are made of wood or iron, more commonly the latter, the best material being selected. They are octagonal, round, or square in section. Ordinary rods are $\frac{3}{4}$ in. to 1½ in. square, $\frac{7}{8}$ in. and 1 in.; they are made in lengths of 1½ ft. to 10 or 15 ft., 10-ft. lengths being most commonly used; the bottom rod is always about 3 ft. long. The usual mode of connecting

the rods is by a screw-joint (fig. 11). Iron rods 1 in. square weigh about 10 lbs. per yard. Wooden rods are generally made in 10 to 30 ft. lengths of pitch pine, and not less than $2\frac{1}{2}$ in. square. The sections are joined by ordinary butt or scarf joints and iron strapping plates.

Bracehead.—For shallow holes boring can be accomplished by the single bracehead, actuated by two or more men, for a distance of 10 or 15 yds.; beyond that depth a double bracehead is used until 20 or 30 yds. is reached, when a spring-pole and windlass will be required. The single bracehead is made with a wooden handle about 3 ft. long and 3 in. diameter at the

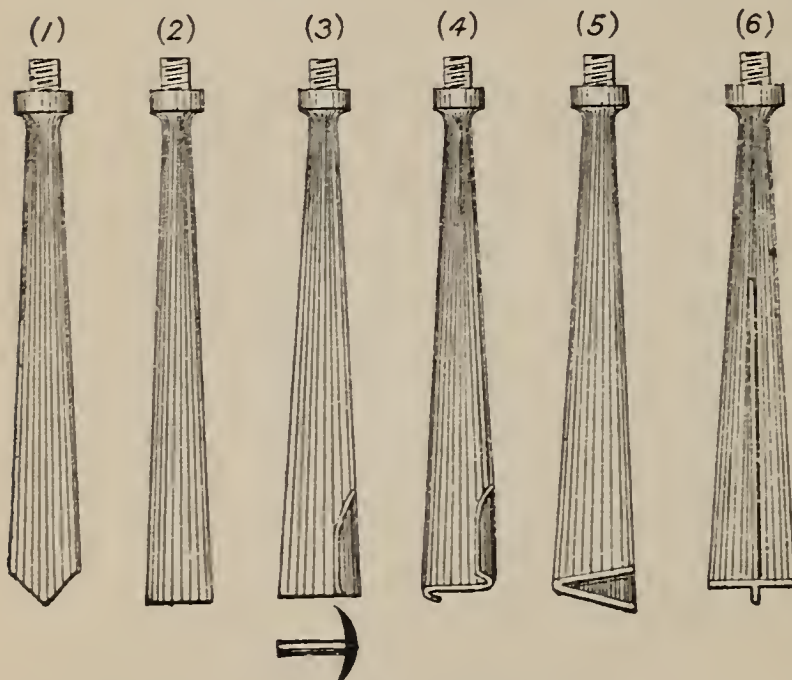


FIG. 10.—Chisels.

- (1) V-chisel ; (2) flat chisel ; (3) T-chisel ; (4) S-chisel ;
(5) flat V-chisel ; (6) cross or four-winged chisel.

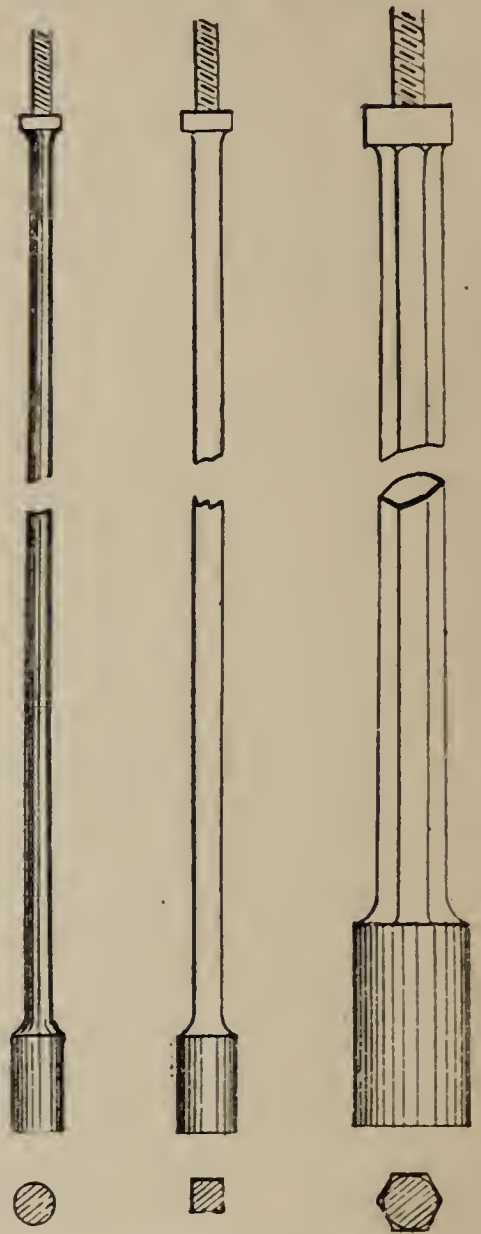


FIG. 11.—Rods.

centre, and tapers at each end. The centre is furnished with an eye made of iron, to which the rods are attached (fig. 12).

Sludger.—The sludger is usually a tube 3 to 10 ft. in length, and of a diameter suitable for the bore-hole. It is provided with an ordinary clack or a ball valve at the bottom (fig. 13). When it is required to clear the bore-hole, the sludger is lowered, and worked up and down a few times at the bottom in order to fill it with the broken material; it is then drawn to the surface, and the contents carefully examined.

The Béche is the tool used for extracting broken rods in cases of fracture. It is about 2 ft. long, and hollow for about 16 in.

at the lower end, the diameter of the opening at the bottom being about $1\frac{3}{4}$ in. and tapering to $\frac{5}{8}$ in. diameter (fig. 14). Other tools used for extracting broken rods are the 'crow's-foot,' spiral worm or wadhook, the bell-box and bell-screw (see fig. 15).

The Brake-staff is a lever of pitch pine, 10 to 14 ft. long, having a fulcrum $1\frac{1}{2}$ ft. to 2 ft. from the end next the rods. At one end is placed an iron hook, a rope being attached to it to enable the men to give it motion (fig. 16).

Preliminary Operations.—When boring is about to be commenced, a platform of wood, 2 or 3 ft. square and 3 in. thick, is laid on the ground, and a hole bored in the centre for the rods to pass through into the bore-hole. Short lengths of rods, 18 in. to 3 ft., are used at the beginning until the hole attains sufficient depth for the ordinary

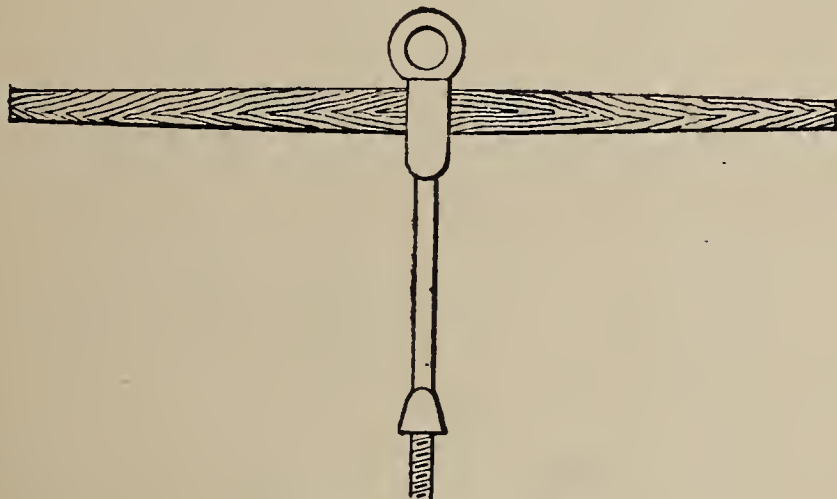


FIG. 12.—Bracehead.



FIG. 13.—Sludger.

lengths to be used. The hole should at starting be larger in diameter than the deeper portions of the boring are intended to be.

If the bore-hole is to be deep, it is a common practice to dig a small pit, 6 ft. square and 12 to 15 ft. deep, before boring is begun. This small pit is of great utility, as it gives additional clearance for the withdrawal of the rods, removes the loose material at the surface, and reduces the ultimate cost of boring. During the actual operation of boring by percussion, the rods are raised about 1 to $1\frac{1}{2}$ ft. and allowed to fall suddenly, driving the chisel against the rock. Every time they are raised the master borer gives them a slight turn with the 'tiller' (fig. 15), causing the chisel to deliver a blow in a fresh direction. When the tool has been at work for some time the bottom of the hole gets filled with débris, which has to be removed by the sludger, which is screwed for this purpose to the end of the rods. When the ground is soft, and the grindings fine and sandy, a sand pump, working on

practically the same principle as an ordinary cylinder pump, is used. If the hole has to be deeper than 20 or 30 yds. a boring trestle and frame are erected; the former as a fulcrum for the brake-staff, and the latter in order to raise the rods easily and speedily when required.

In fig. 15 are shown some of the tools used in ordinary boring, but there are many others. In boring it is usual to erect some sort of head-gear to enable the rods to be raised quickly and easily. This head-gear may consist of a triangular frame of three long wood poles, either circular or square, meeting at the top, where they are fastened together by a bolt. The head-gear may be from 20 ft. to 60 ft. in length, the higher the better; but whatever height is adopted, it ought to be a multiple of the lengths of the boring-rods used, so that when the rods are raised, the joint for unscrewing should be just above the top of the bore-hole. Thus, if the rods are in 12 ft. lengths, the boring frame ought to be 24, 36, or 48 ft. high. If the hole is to be a deep one, a steam winch is used, as it raises the rods more speedily, and is more reliable than a hand windlass.



FIG. 14.—Béche.

When the hole reaches a certain depth the rods require to be balanced in some way, as their whole weight, if allowed to fall on the chisel, would damage or break it. To remedy this, the weight of the rods may be transferred to the end of the brake-staff, or a rope can be used instead, and by a suitable arrangement a weight sufficiently heavy to cut the rock is

allowed to fall on the chisel. This plan has been adopted in Mather & Platt's system of boring. Some boring engineers prefer wooden rods for boring, for when the hole fills with water the rods are buoyed up, and a great deal of their weight is thus taken off the chisel.

LIST OF TOOLS SHOWN IN FIGURE 15.

- | | | |
|--------------------------|------------------------|-----------------------------|
| 1. Spiral worm or miser. | 10. Shoe-nose shell. | 20, 21. Lengthening pieces. |
| 2. Bell screw. | 11. Auger-nose shell. | 22. Lifting dog. |
| 3. Bell box with cleats. | 12, 13. Shell augers. | 23. Nipping fork. |
| 4. Crows-foot. | 14. Bow dog. | 24. Hand dog. |
| 5. Bell-mouthed shell. | 15. Spring dart. | 25. Snatch block. |
| 6. Auger shell. | 16. Tillers or levers. | 26. Auger cleaner. |
| 7. Worm or screw auger. | 17. Gravel chisel. | 27. Holding-up rod. |
| 8. Plug drill. | 18. Clay auger. | 28. Tie screw-driver. |
| 9. Parallel worm auger. | 19. Reamer. | 29. Spring hook. |

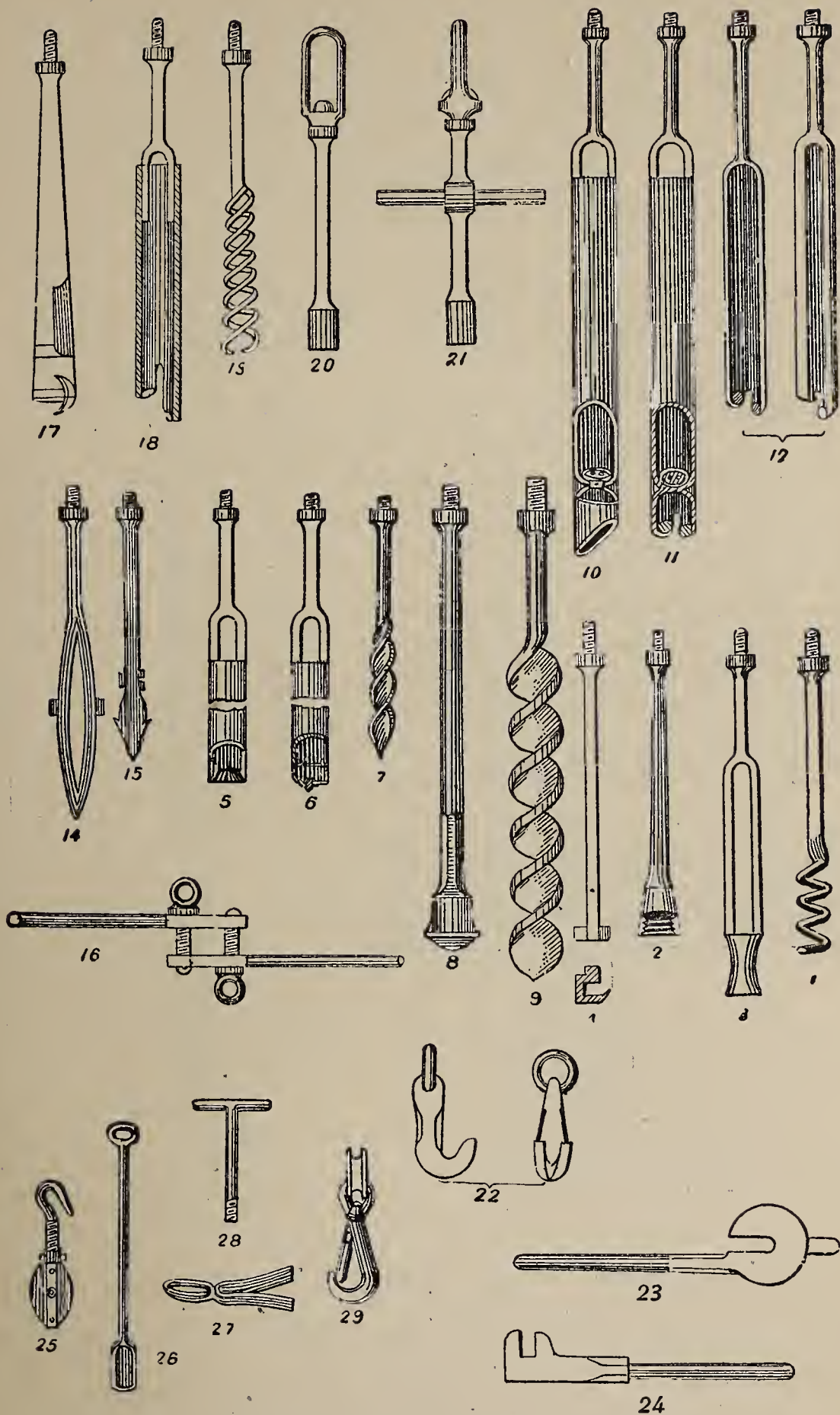


FIG. 15.—Boring Tools. (For list see opposite page.)

Another arrangement to guard against fracturing the rods is the sliding joint, usually fixed 10 to 20 yds. above the chisel, the rods *a a* below it being made extra strong (fig. 17). When the chisel strikes the ground, the upper lengths of rods *b* move over the sliding joint, until the beam to which it is fixed has completed its stroke, when an elastic stop at the upper end helps to deaden the fall, and thus the shock due to the chisel and rods striking the rock simultaneously is avoided.

On the return stroke the collar *c*, suspended from the rods *d* by the fork *f*, catches against the projecting crosshead *e*, and thus lifts the chisel again.

*Oeynhausens's Sliding Joint.**—This contrivance (figs. 18, 19) consists of an upper piece *a*, provided with a slot in which the lower piece *b* can slide; *b* is prevented from falling out by a crosshead, and carries the boring chisel, whilst *a* is attached to the line of rods. When a down-stroke is made and the chisel strikes the bottom, the piece *a* slides over *b*, and is therefore but little affected by any jar produced by the blow of the tool. The length of the stroke is arranged so that the top of the slot will not descend

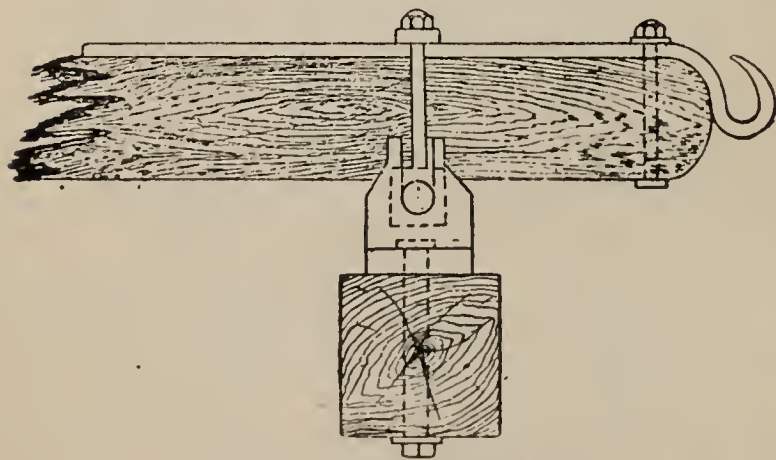


FIG. 16. — Brake-staff.

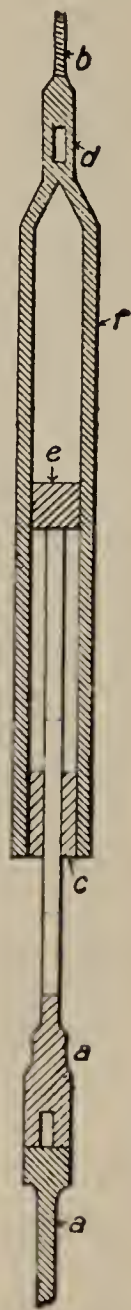


FIG. 17. — Sliding joint.



FIG. 18.

far enough to touch the crosshead; *a* is then raised once more and again catches the crosshead.

Arrault's Free-fall Cutter.—In this apparatus the boring tool is suspended from the catch (*h*, fig. 20). The part *a b* has a pin, *i*, which lies in an oval hole. While the rods are being lifted the boring beam strikes a bumping piece, and their upward movement

* *A Text-book of Ore and Stone Mining*, by Sir C. le Neve Foster, seventh edition, pp. 143, 144.

is suddenly arrested; inertia carries the catch up a little, the end *a* strikes an inclined surface and causes the end *b* to move outwards and detach the tool. When the rods are lowered the part *h* hooks itself on without difficulty, and the chisel is raised and dropped. This tool requires the boring rod to be guided, for otherwise the hole might not be bored straight.

If the rods happen to break, the 'Béche,' 'Crow's-foot,' or some other grapnel is used to grip and raise them. A simple kind of grapnel is a bell-mouthed tube about 5 ft. long (fig. 21). Near the bottom of the

inside of the tube are fixed four steel blades or springs. To extract the broken rods the tube is lowered until it passes over a joint below the fractured rod, the steel blades being pressed outwards when passing this joint, but immediately it is passed they press firmly in on the rod, and the grapnel is then raised, taking the broken rods along with it. A steam or hand windlass to which a rope is attached is usually employed for clearing the hole, and

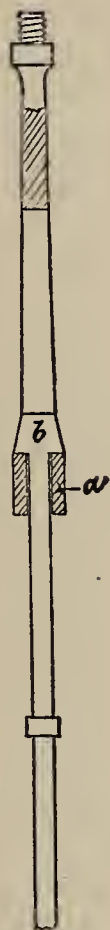


FIG. 19.

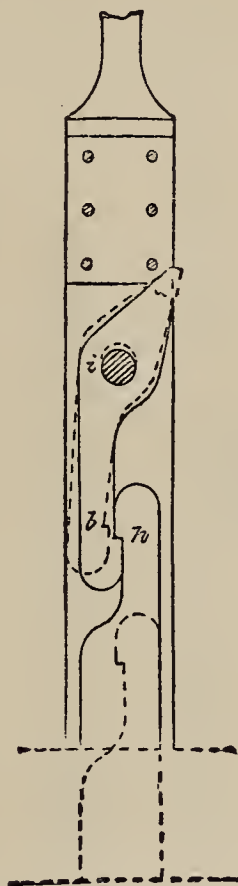


FIG. 20.—Arrault's free-fall cutter.

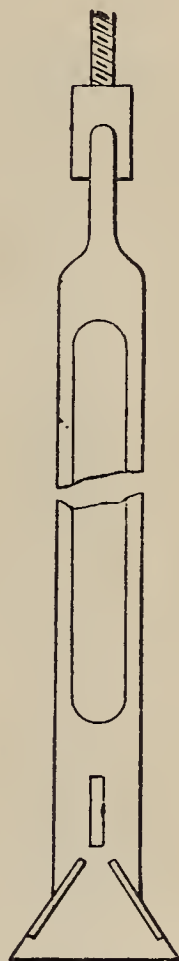


FIG. 21.—Grapnel tube.

also for raising the rods when the chisels are being changed.

Lining the Bore-hole.—If the sides of the hole are of a soft nature and apt to fall in, tubes ought to be inserted to protect the walls and to allow the rods to work freely. Those used for this purpose are generally made of wrought iron $\frac{1}{8}$ in. to $\frac{5}{8}$ in. thick, in 9 to 12 ft. lengths. They are forced down by repeated blows from a heavy hard-wood or iron drive-block or 'monkey.' A drive-head is fitted to the top of the tubes and a cutting shoe to the bottom. The drive-block is raised by a winch or steam, by means of a rope, and dropped from a suitable height, striking twenty-five to thirty blows per minute. By this means the lining tubes are driven into position. The tubes may get bent by such hammering, and it is then better

to use either screw-jacks or hydraulic rams to force them down. The tubes are made with either screw-joints (fig. 22), or lap-joints with rivets, the former being the more frequently used.

Removing the Lining.—If the tubes can be withdrawn, they are often taken out after the bore-hole is finished, but if they resist removal, they are left standing, as the cost of withdrawing them would, in many cases, be more than the original cost of the tubes. When the friction is not great, Kind's plug or shuttle is used. It consists of an oval ball of wood or iron, of slightly smaller diameter than the tubes. It is fastened to the end of the rods and lowered

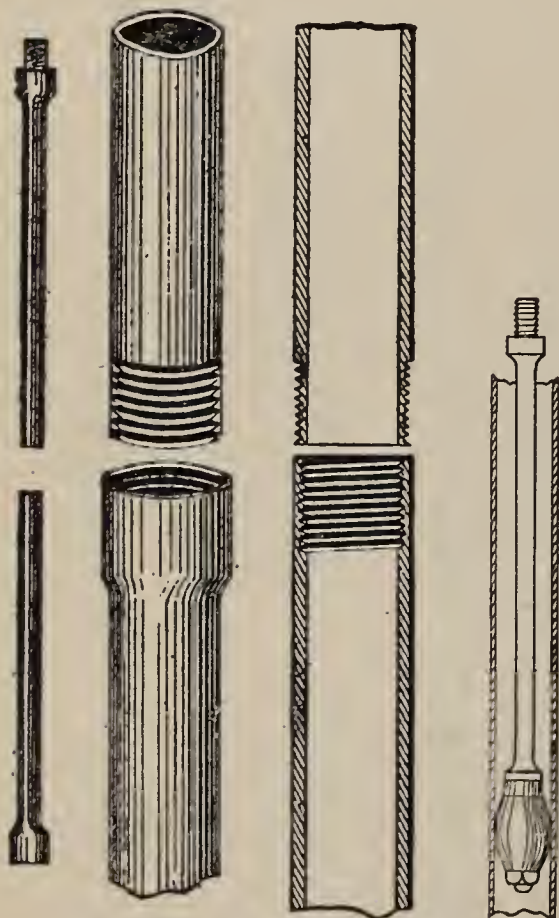


FIG. 22.—Tubes for lining.

FIG. 23.—Plug.

down; a short, open tube resting on its upper surface is filled with coarse sand. When the desired position has been reached, the short tube is raised by a rope, and the sand spreads over the plug, filling the space between it and the tubes, thus causing sufficient friction for the latter to be withdrawn when the plug is drawn up (fig. 23). If it is difficult to withdraw the tubes, a tool provided with two darts or jaws which work on a spring has to be used. When this tool is being lowered down, the jaws are closed, but as soon as they reach the bottom of the tube they fly outwards, and enable them to be withdrawn (see 15, fig. 15). For withdrawing tubes from deep bore-holes, Mather & Platt use a patent hydraulic casing cutter or 'splitter.' An apparatus called a 'harpoon' is also used for the same purpose.

Speed of Boring.—The speed with which a bore-hole is put down varies according to the nature of the strata passed through and the kind of tools and machinery employed. In ordinary coal-measures the rate of progress may, in the early stages, be as much as 1 ft. per hour; but as the hole gets deeper the speed will be very much lower, owing to the greater amount of time taken up in raising and lowering the rods, and the rate of progress may not exceed 1 or 2 ft. in twenty-four hours. In hard rocks, such as whinstone, the distance bored may not be more than 6 in. per diem.

André gives, as an average speed in different measures, 1 ft. 9 in. per eleven hours.

Cost of Boring.—This will vary greatly according to the nature of the ground passed through and the wages ruling in the district.

In ordinary boring by spring-pole in the coal-measures, the prices ruling in the North of England are usually 7s. 6d. per fathom for the first 5 fms., 15s. for the second 5 fms., 22s. 6d. for the third, and so on, increasing 7s. 6d. per fathom for every additional 5 fms. depth. The prices in Scotland are 4s. to 4s. 6d. per fathom for first 5 fms., 8s. per fathom for second 5 fms., and so on, increasing by 4s. per fathom every step of 5 fms. In other districts the prices are sometimes 5s. 6d. per yard for the first 5 yds., 7s. 6d. for second 5 yds., 9s. 6d. for the third 5 yds., and so on, increasing 2s. per yard for every additional 5 yds. in depth. These prices are generally taken on the basis of bore-holes starting 3 in. diameter.

The cost of bore-hole may be found by the formula,

$$x = \left\{ 2a + (n-1)d \right\} \frac{n}{2},$$

where x = total cost in shillings, a = price of first step, d = increase of price for each additional step, and n = number of steps.

Example.—What would be the cost of a bore-hole 150 fms. deep if the price of boring is 7s. 6d. per fathom for first 5 fms., and increases by 7s. 6d. for every subsequent 5 fms.?

The total number of steps will be $\frac{150}{5} = 30$,

$$\begin{aligned} \therefore x &= \left\{ (2 \times 7 \cdot 5 \times 5) + (30 - 1)7 \cdot 5 \right\} \frac{30}{2} \\ &= \{ (75) + (29 \times 7 \cdot 5)15 \} = 4387 \cdot 5s. = \text{£}219 \text{ 7s. 6d.} \end{aligned}$$

Japanese Method of Boring.*—This method is chiefly applicable in soft alluvial deposits, which can be bored through at the rate of 50 to 60 ft. per day. The tools used are solid, round, iron rods, connected by fish-joints, and terminating in a pear-shaped solid plunger for soft ground and an obtuse pyramidal point for hard ground. By a rocking lever terminating with a Jizai Kagi, or *kettle-catch* (peculiar to Japan), the rods are pumped up, 6 in. at a time, to a height of from 2 to 15 ft., when they are allowed to fall. Material is but seldom taken out of the hole. During the boring the hole is kept filled with water charged with clay. Subsequently it may be lined with bamboo pipe. The method is cheap and rapid.

Cost of Boring Plant.—The following are some estimates of boring plant costs in chisel boring in ordinary coal-measures.

For a depth of 33 ft., with 1-in. rods and 2-in. boring tools,		
to include rigger, rope, and shear-legs,	£21	0 0
For boring 3-in. and 4-in. holes, 50 ft. deep, with 1-in. rods,		
2½-in. and 3¾-in. tools, with rigger, rope, and shear-legs,	35	6 0

* *The Miners' Handbook*, Prof. J. Milne, F.R.S., p. 42.

For 3-in. and 4-in. hole, depth 100 ft., with 1-in. rods, $2\frac{3}{4}$ -in. and $3\frac{3}{4}$ -in. tools, with screws for $1\frac{1}{4}$ -in. bottom rods, to include rigger, rope, and shear-legs, ...	£44	15	0
For holes 150 ft. deep, $1\frac{1}{4}$ -in. rods, $3\frac{3}{4}$ -in. and $4\frac{3}{4}$ -in. tools, with rigger, rope, and shear-legs, ...	57	5	0
For holes 200 ft. deep, $1\frac{1}{4}$ -in. rods, $3\frac{3}{4}$ -in., $4\frac{3}{4}$ -in., and $5\frac{3}{4}$ -in. tools for boring 4-in., 5-in., and 6-in. holes, with screws for $1\frac{1}{2}$ -in. bottom rods, and with rigger, etc., ...	70	0	0
For holes 300 ft. deep, with $4\frac{1}{4}$ -in., $5\frac{1}{4}$ -in., and $6\frac{1}{4}$ -in. tools, to include rigger, rope, etc., and geared windlass, ...	120	0	0
For holes 500 ft. deep, same as No. 6, with £35 extra for rods, ...	155	0	0
For holes 800 to 1000 ft. deep, with boring tools, etc., and 800 ft. $1\frac{1}{2}$ -in. rods, ...	195	0	0

Difficulties and Accidents in Boring.—These, which are common to all methods of boring, may be briefly summed up as follows :—

Brittleness induced in rods by repeated blows and vibration.

Rods, chisels, bolts, and other tools, by carelessness, fall into the hole, and their extraction gives great trouble.

If the rods are not kept rotated, projections will be left on the sides of the hole, which will lose its circular form and cause the rods to get fixed or broken.

The hole may deviate from the vertical, and so cause the rods to get jammed.

The sides of the hole may fall in and cause much labour and expense in clearing again.

By fissures or change of strata the hole may become narrower.

Mather & Platt's System.—In this system of boring, the ordinary rods are dispensed with and a rope substituted for them. The rope *a* passes over a pulley (*b*, fig. 24), and is guided by another pulley *c* to a drum, which is worked by a horizontal cylinder. Between the drum and the pulley a clamp *d* is attached to the boring frame. The rope between the top pulley *b* and the drum is fixed by this clamp during the operation of boring. A vertical cylinder *e* connected directly to the pulley actuates the boring rope to which the boring tool is attached. Steam is admitted into this vertical cylinder *e* on the bottom side of the piston, and the wheel *b*, which works in a sliding frame, is raised along with the rope and boring tool. When the piston has completed its upward stroke, a valve *f* at the bottom of the cylinder is opened, allowing the steam to escape, and at the same time the boring tool falls rapidly, giving a smart blow at the bottom of the hole. The length of the stroke can be varied by self-acting tappets at the bottom of the cylinder. As the wheel *b* can be rotated as well as raised vertically, the boring tool will be raised 2 ft. for every foot the piston is raised, and hence on the downward stroke it will fall with twice the speed of the piston. The horizontal engine and drum attached are only for raising and lowering the boring tool and the sand pump or sludger. The rope is generally flat and of hemp, $4\frac{1}{2}$ in. broad by $\frac{1}{2}$ in. thick.

In this system of boring, the great difficulty at first was the

turning of the boring tool at each stroke. For this purpose an ingenious arrangement was adopted.

The sliding collar (*g*, fig. 25) has a series of teeth top and bottom. On raising the rope on the up stroke, this collar *g* fits into another fixed collar *f* at the top, the lower edge of which has a corresponding set of teeth. When the tool falls and strikes the blow at the bottom of the hole, the sliding collar *g* descends and fits into another collar *e*, having teeth which are set half a tooth in advance of those in the sliding collar *g*. Thus, when the tool falls, the inclined surface of the lower teeth in the collar *g* strikes the point of the teeth in *e*, and finally fits into them, thereby giving the flat rope a turn of half a tooth, and on the tool being raised it is twisted another half tooth, when the sliding collar comes again in contact with the teeth in *f*. The rope therefore receives altogether a twist to the

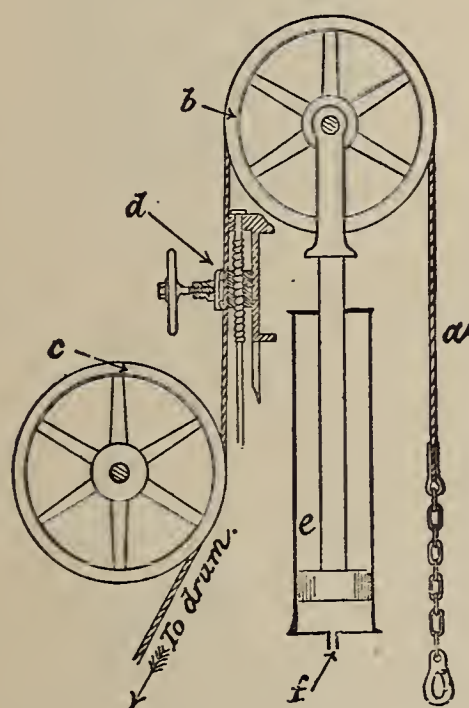


FIG. 24.—Mather and Platt System.

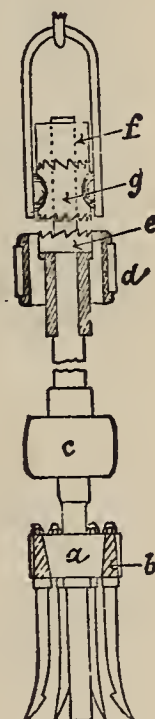


FIG. 25.—Sliding collar.

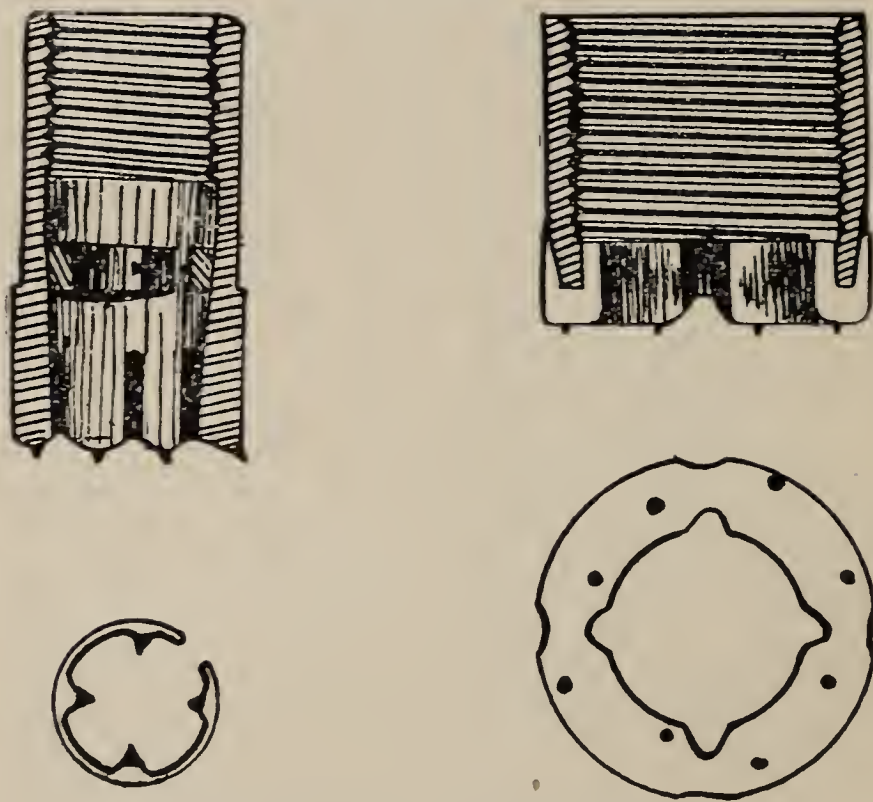
extent of one tooth, and in untwisting it must turn the tool a like amount, causing it to strike in a fresh place. This constant change goes on automatically, and secures the efficient cutting of the rock. The boring tool can be lowered at the rate of 500 ft. per minute, and can deliver about twenty-four blows per minute, the rate of boring being 2 to 4 ft. per diem in deep bore-holes. The cost of boring seems, however, to be more expensive than other systems, a bore-hole at Middlesbrough, 1200 ft. deep, costing no less than £8, 6s. 8d. per foot.*

Messrs Mather & Platt have recently introduced a machine similar to the method employed at Pennsylvania oil wells, *i.e.* by means of a circular rope, attached to the end of a large oscillating

* *Trans. N. Eng. Min. and Mech. Engs.*, vol. xxx. p. 88 ; and Foster's *Ore and Stone Mining*, sixth edition, pp. 156 and 324.

beam, which is actuated by a large belt wheel from an engine, by a connecting rod and crank.*

Diamond or Rotary Boring.—In this system of boring, a solid core of the strata passed through is cut out by means of diamonds set in a cylindrical crown, which is fastened to the foot of the rods, and is furnished with grooves for the circulation of water. At the foot of the boring rods there is a cup 3 ft. long and $4\frac{1}{2}$ in. inside diameter, by which any sediment, too heavy for the water to carry away, is caught. This cup and the rods are attached by two plugs to a core tube 12 ft. long; at its lower end there is a nipple for connecting it with the crown (figs. 26, 27). At the junction of the two plugs there is a loose steel ring, inside which the core tube and



FIGS. 26 and 27.—Diamond drills.

crown revolve, and which serves to prevent the tube from getting worn by friction with the sides of the bore-hole. Inside the crown there is a spring trap for extracting the cores. Its sides are parallel throughout its inside diameter, but bevelled on the outside to allow of its rising and falling. When the crown is revolving the spring trap is 'up,' in order to allow the core tube to rotate freely, but when the core is about to be drawn the trap is jammed down to permit the core to be broken off as required. The boring rods are hollow, about $1\frac{3}{4}$ in. inside diameter, made of mild steel, in lengths of 12 ft., and weigh about 50 lbs. each. A constant stream of water is forced under pressure through these rods at the rate of 12 to 16 gallons per minute. On reaching the bottom this water circulates outside the rods, returning ultimately to the surface, and carrying

* *Trans. N. Eng. Inst. Min. and Mech. Engs.*, vol. xxx. p. 88.

with it any sediment made by the cutting of the core. The rods are rotated by gearing driven by a pair of small vertical engines with cylinders 6 in. diameter and 10 in. stroke, the revolutions of the rods being at the rate of 100 to 200 per minute.

Figs. 28 and 29 show the method of working. At the beginning of the bore the rods are weighted (fig. 28) to give sufficient pressure

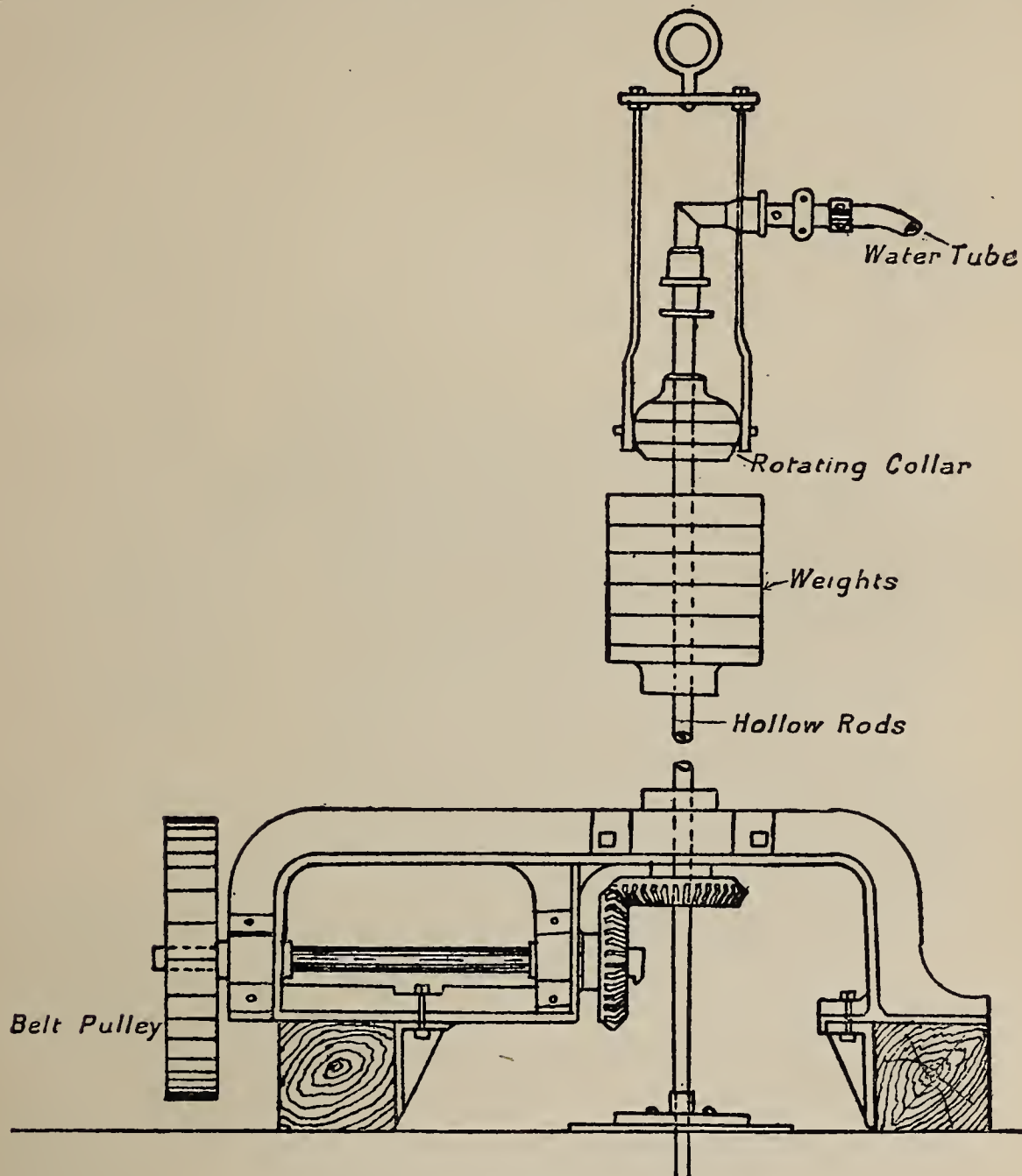


FIG. 28.—Boring frame. *Elevation.*

to the diamonds, the weight being from 15 to 20 cwts. in soft strata and from 20 to 30 cwts. when hard strata are being cut. Fig. 30 shows a Sullivan improved diamond drill, which has been largely used in South Africa, Australia, and America. It is fitted to drill to a depth of 3000 ft., giving a core $1\frac{3}{8}$ in. diameter. The engines, hoisting rig, and feeding mechanism are strongly built; the hoisting gears are proportioned for a heavy line of rods, the drum making 1 revolution to 39 of the engine. This machine is fitted with double

brake bands on the hoisting drum, a friction clutch by means of which the hoisting speed may be altered from slow to fast, or *vice versa*, and double friction roller bearings for sustaining the weight of the drill rods. The total weight of this drill and outfit is 26,000 lbs., and the cost about £1400. As the depth of the hole increases, these weights are gradually taken off, the weight of the rods alone being sufficient to cause the required pressure, while, if the hole be a deep one, part of the weight may require to be counterpoised by others. The diamonds used for boring are what are known as 'Bortz' diamonds, weighing 2 to 6 carats, and costing 15s. to 30s. per carat according to ruling prices. In a crown $5\frac{1}{4}$ in. diameter, sixteen to eighteen stones would be used, costing from £35 to £40. The whole crown, including diamonds, would cost about £80.

Speed of Diamond Boring.—In good firm strata, and with a speed of 100 revolutions per minute, 30 to 40 ft. may be bored through in

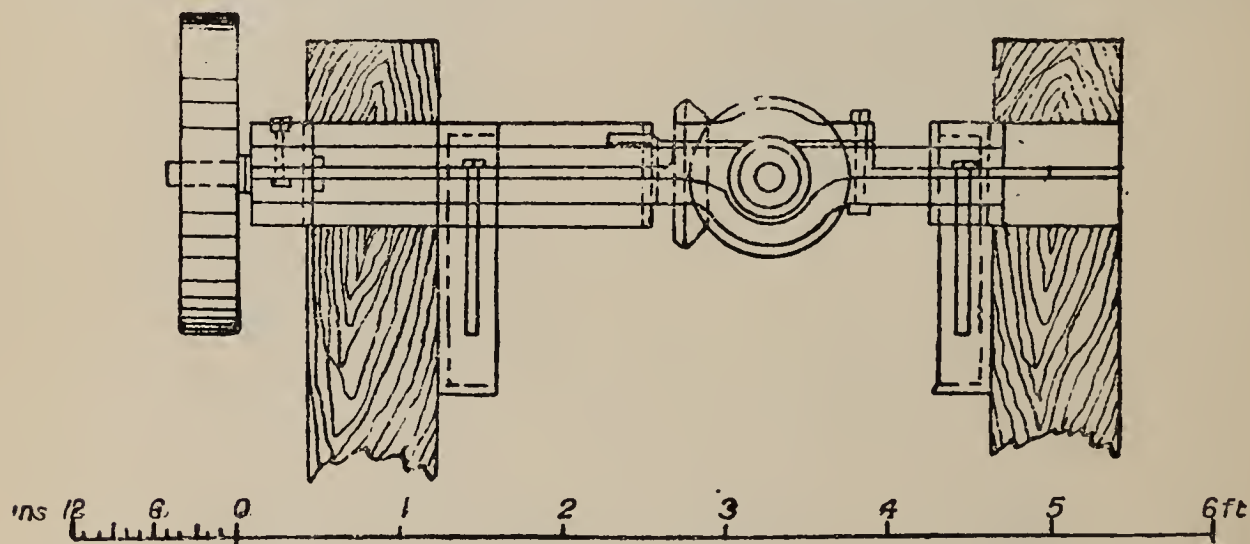


FIG. 29.—Plan.

nine hours; but either in very hard rock or very soft strata the rate would be sensibly reduced, particularly if the hole becomes deep. A bore-hole was put down at Newton in Lanarkshire by Messrs Thomson Brothers, Dunfermline, the depth of which was 149 fms. 3 ft. 10 in. It was commenced on 19th December 1889, and finished on 25th March 1890, the total number of days worked being sixty-seven, giving an average rate of cutting of 13·4 ft. per day of nine hours. To attend to the operations three men were required, a leader and two assistants.* In another bore-hole, 1830 ft. in depth, and varying from 4 to 5 in. diameter, the total time occupied was 280 shifts of nine hours per shift, giving an average cutting rate of 6·5 ft. per shift. For ten bore-holes put down in the Pennsylvania coal-measures, with a total depth of 4576 ft., the average run per shift of ten hours was 18 ft. 2·7 in. A bore-hole put down in Fife, probably the deepest in Britain, to a depth of 4534 ft., took fifty-eight months to complete, and the average drilled per month was 78 ft.,

* *Trans. Min. Inst. Scot.*, 1891, p. 156.

or about 3 ft. per day. The rate of boring by a diamond drill will depend greatly upon the nature of the strata and the depth of the hole.

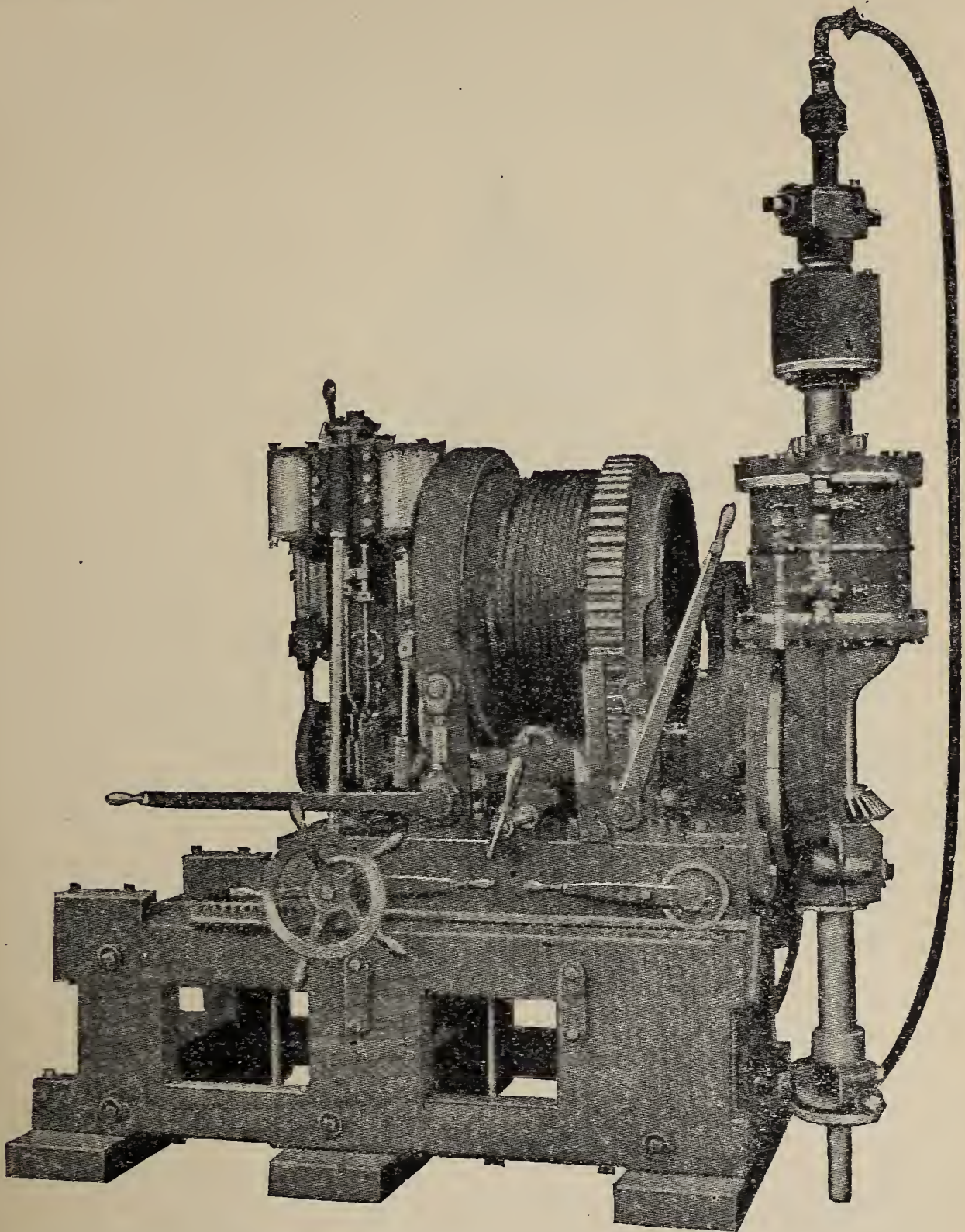


FIG. 30.—Sullivan improved diamond drill.

Cost of Diamond Boring.—In the North of England the cost of diamond boring is 8s. per foot for the first 100 ft., increasing 8s. per foot for every additional 100 ft. In other districts, bore-holes not exceeding 1000 ft. are charged £1 per foot, 1000 to 1500 ft. deep £1, 10s. per foot, 1500 to 2000 ft. deep £2, 10s. per foot. In

Scotland the cost of diamond boring is £3 per fathom for depths up to 150 fms., and from 150 to 250 fms. deep £5 per fathom. Other prices are: holes not exceeding 1000 ft., 20s. per foot; 1000 to 1500 ft., 30s. per foot; 1500 to 2000 ft., 50s. per foot.

For these rates the boring firm usually supply all labour, tools, and machinery required, those for whom the bore is being made paying railway costs and cartage to the site of the bore, and also providing

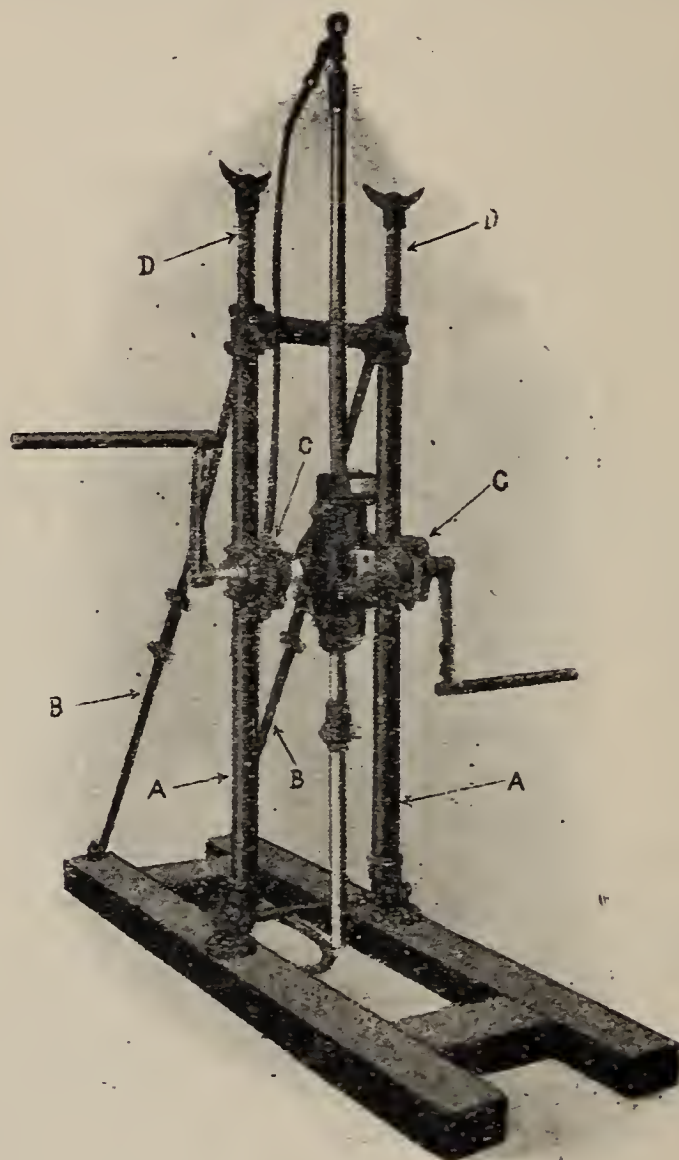


FIG. 31.—Sullivan hand-power diamond drill.

the necessary lining tubes, fuel supply for engine, and water supply required for the boring operations.

Hand-power Boring Machine.—For boring holes 200 to 400 ft., either on the surface or underground, in ordinary coal-measures strata, hand-drilling diamond machines may be successfully employed. Fig. 31 illustrates a machine of this type. The main columns, A A, are attached by hinged feet to the wooden frame, and steadied by back brace-rods, B B, adjustable to any position. The swivel head is secured to the columns by adjusting clamps, C C, to permit the spindle to be turned in any direction, so that holes may be drilled

at any angle from horizontal to perpendicular. The timber foundation prevents any tendency of the columns to spread. The drill spindle is hollow and fitted with a chuck at the lower end to allow of the use of long rods. The thrust of the bit on the rock is taken up by ball bearings on the feed nut, thus reducing friction and the amount of power necessary for operation. A differential screw feed with friction escapement is used, and, in addition, two sets of feed gears, mounted on the machine, permit an instant change of speed while the drill is running. A small hand-power hoist may also be attached when required. When required for underground use the back brace-rods are removed, and the machine is fixed by the extension screws, D D, at the top of the columns, against wood blocks at the roof of the working. The total weight of the machine is 325 lbs., and costs £160 to £200.

Advantages of the Diamond System.—The advantages claimed for this system are that it is more expeditious than methods of boring by a chisel, and that, cores of the strata being obtainable, a correct section of the rocks passed through can be ascertained and the inclination of strata seen, while for deep holes it is cheaper than the chisel method. In very soft strata the cores got are not, however, very satisfactory.

Davis Calyx Drill.—This system of boring, on the rotary principle, closely resembles diamond drilling.* The boring bit, or cutter (fig. 32), consists of a cylindrical metal shell, the lower end of which has been formed, by a process of gulleting, into a series of long sharp teeth. These teeth are set in and out alternately. Those having an outward set are used to drill the hole just large enough to allow the apparatus to descend freely, and the teeth having the inward set dress down the core to such a diameter as to allow the body of the cutter to pass freely over it without binding. The front face of each tooth is perpendicular at the base to the rock to be operated on; while the back of the tooth rises from the same line at an angle of about 60°. Immediately above the cutter is the core-barrel, which is connected to the boring rods by means of a reducing plug, which also serves to close the lower end of the calyx. The calyx is simply a long tube or series of connected tubes, located above the core-barrel, to which it is equal in diameter. The lower drill rods work through the centre of this calyx, there being an annular space between the two, and at the upper end the calyx is kept concentric with the drill rods by means of a centering device. The speed of the cutter is about 10 revolutions per minute. The calyx cutter is only used for the softer rocks, and when hard strata has to be bored through it is replaced by what is known as a shot-bit. This

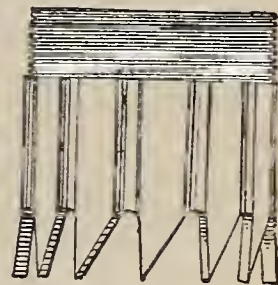


FIG. 32.—Davis calyx cutter.

* *Trans. Inst. Min. Engs.*, vol. xv. pp. 363-366.

consists of a cylindrical steel tube having triangular notches cut at the bottom, with a base of $1\frac{1}{4}$ to 2 in. in vertical height. These notches allow the shot to gather on both sides of the bit, and also provide for the free circulation of the water used to flush the hole. The chilled shot used, No. 6 size, is washed downward through the hollow rods. In the newer type of calyx a hydraulic cylinder is fitted instead of a counterbalanced weight. It consists of a cast-iron cylinder, with a piston and leather bucket inside, and it is to all intents and purposes a hydraulic weighting machine. It is claimed for this apparatus that the pressure exerted on the boring tool can be more accurately ascertained. By means of a gauge connected by a high pressure hose to the water in the cylinder, the foreman driller can, at any time, ascertain the pressure on the tools, as the ratio between the pressure recorded and the actual pressure must always be practically constant. He can also, by opening an exhaust valve at the water end of the cylinder, regulate the rate at which the rods travel downwards. The most useful feature of the hydraulic cylinder is that by its means the foreman can obtain a fair idea of the nature of the strata being passed through.*

Mode of Operation.—When drilling is being carried on, a continuous stream of water is pumped down the drill rods, in the same way as in diamond drilling, the drill at the same time being slowly rotated and forced downward. The rods have to be twisted considerably before they accumulate sufficient energy to overcome the 'bite' of the teeth into the rock, but the moment the surface strain exceeds the resistance below, it begins to grip into the strata by a series of motions similar to that of a stonemason's hammer and chisel action.

The débris which is produced by this action in the formation of the core is carried up by the stream of water to the top of the calyx, where, owing to the reduction in velocity of the water flow, they slowly fall back into the annular space between the drill rods and the inside of the calyx tube. To extract the cores, a quantity of grit is washed down the rods, and it is arrested at the cutter or shot-bit, both of which taper slightly inwards at the bottom, so that when the rods are lifted the particles of grit become tightly wedged between the core and the sides of the tool and the core-barrel, and hold the cores tight. As the rods are raised, the bottom part snaps off, and the core-barrel is then raised to the surface. It is claimed for this system that a longer core can be produced and more accurate results obtained than with the diamond drill, and that the rate of boring is greater. The calyx drill was first used in Australia, but it has recently been used a good deal in this country. A hole 2371 ft. deep was put down at Barlow in Yorkshire, and a good deal of the drilling done in Kent to prove the coal-measures has been carried out by the calyx drill.

* *Trans. Inst. Min. Engs.*, vol. xxxiv. pp. 426-438.

Other Objects of Boring.—Boring has often to be carried out underground when old workings, supposed to contain water or gas, are being approached.* In these circumstances, a drift or pair of drifts 8 ft. wide are driven in advance of the working faces at the nearest point to where the waste is to be tapped. Bore-holes are kept in advance of the face for a distance of not less than 15 ft., and also sufficient flank holes set at an angle of about 45° with the drift. If the coal and roof are 'tender,' and the pressure of the water great, the advance holes will require to extend a good deal further than 15 ft. Whatever length is decided on ought to be adhered to throughout, and the greatest care should be taken, as old plans cannot be implicitly relied on. The holes are usually bored with light iron rods $\frac{3}{8}$ in. to $\frac{1}{2}$ in. square, and in 6-ft. lengths, the drill cutting a hole about $1\frac{1}{2}$ in. diameter, the holes being sloped a little towards the roof.

If the old workings take the form of an irregular boundary of 'stoop and room,' there is always the danger of an open 'drift' being passed by the exploring road at a short distance from it, so that if the flank holes are not sufficiently close to catch the old roads, the water may break through on the side of the exploring drift, possibly at some distance back from the face. After the waste is tapped, it may be necessary to regulate the escape of water to suit the capacity of the pumping plant, otherwise the pit may become flooded. This is often done by the insertion of a tube in the last length of the hole, into which are fitted wooden plugs, 5 to 8 ft. long, tapered 1 in. to $2\frac{1}{2}$ in., having a hole bored through their centre to allow just as much water to escape as the pumping arrangements can adequately deal with. The tube may also have a tap fixed to it, when the water can be drawn off as required. It is also a common and useful practice to attach a pressure gauge to the outer end of the tubes to show the pressure of the water, and thus definitely ascertain the head of water to be dealt with. If fire-damp is suspected, only safety-lamps should be used, and spare lamps kept ready lighted at some distance in the rear. Precautions should also be taken to provide against other gases escaping in dangerous quantities.

In tapping wastes by the ordinary method of drilling holes, over, on an average, 20 ft. in length, there is always much difficulty experienced in getting the tools to clear themselves; indeed, they often get choked up altogether with the loose débris, and occasion much trouble in withdrawal. To remedy this difficulty, boring

* In the Coal Mines Act, 1911 (Rule 67), it states: "Where any working has approached within 40 yds. of a place containing, or likely to contain, an accumulation of water or other liquid matter, or of disused workings (not being workings which have been examined and found to be free from accumulations of water or other liquid matter), the working shall not exceed 8 ft. in width, and there shall be constantly kept extending to a sufficient distance, not being less than 5 yds. in advance, at least one bore-hole near the centre of the working, and sufficient flank bore-holes on each side at intervals of not more than 5 yds."

machines specially adapted for the purpose are sometimes used. Such a machine is shown in fig. 33.

The machine consists of a cylinder *c* $1\frac{1}{2}$ in. diameter, furnished with packing glands through which a spindle *s*, connected to the boring rods, is worked.* Fixed on the frame is a pump-chest *D*, which is connected to the cylinder *c* by means of an india-rubber pipe *E* $\frac{3}{4}$ in. diameter. In this pump-chest are two small plunger pumps, 1 in. diameter and 1 in. stroke. These pumps are worked off two cranks on the spindle *s*, and are supplied with water by a pipe leading to a cistern. On the outer end of the spindle is a handle for working the machine. The whole apparatus is placed on a bogie running on ordinary rails. The machine is kept moving forward, while the drilling proceeds, by means of a chain fixed to a barrel *b*

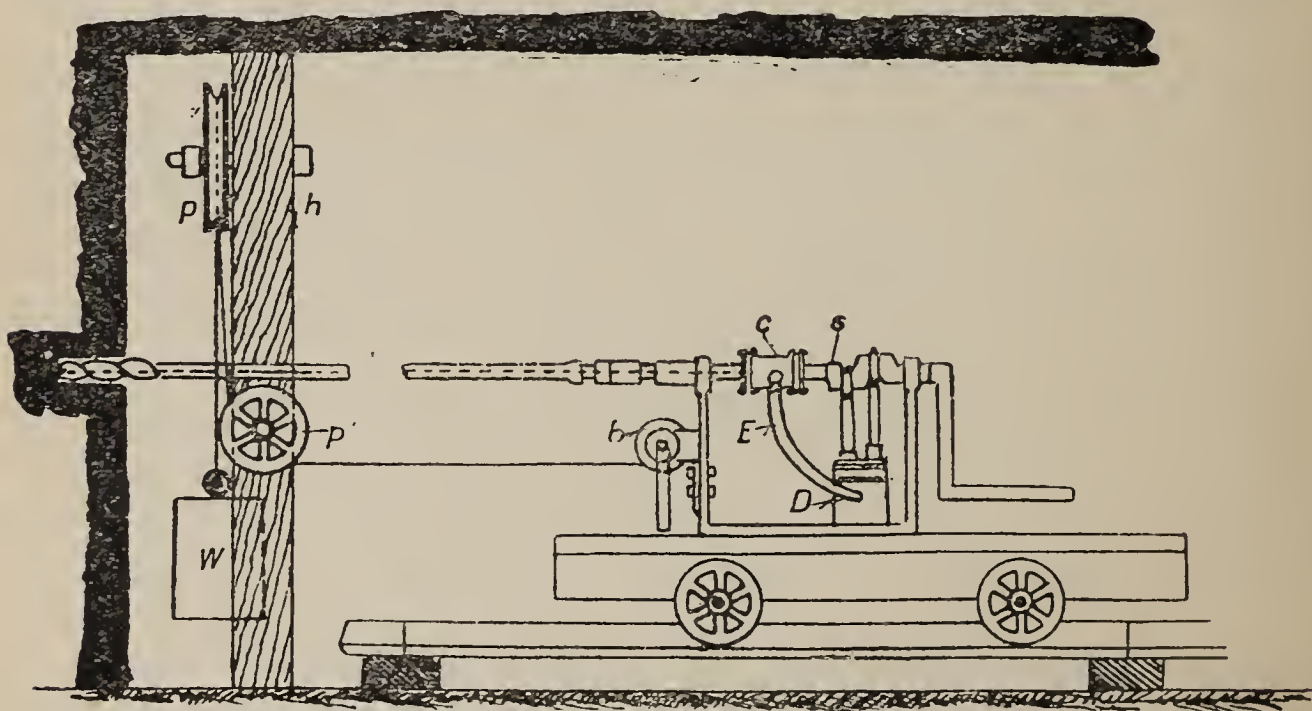


FIG. 33.—Boring machine.

with a ratchet wheel; the chain passes round two other pulleys *p p'* fixed to a prop *h*, and a weight *W* is hung at the other end of the chain. The rods, which are hollow, are $\frac{7}{8}$ in. diameter outside with $\frac{1}{2}$ in. diameter opening, and are made in 6-ft. lengths. They are connected to each other by the ordinary method of screwing. The drill is also hollow and $1\frac{1}{2}$ in. outside diameter, and is of the ordinary description used for drilling holes, except that $1\frac{1}{2}$ in. from the point is an opening to allow the water to escape. In applying the machine, the handle is rotated, and this in turn works the cranks attached to the two force pumps which force the water into the cylinder *c*, and thence into the hollow rods, which carry it forward to the drill point, where it is discharged, and flows back through the hole, carrying the débris along with it. Holes have been bored with this machine for distances of 150 to 170 ft., the rate of cutting being on an average

* *Trans. Min. Inst. Scot.*, vol. xiii. p. 82.

30 ft. per shift of eight hours, employing two men. The machine can be used either in ordinary strata or in coal.

Old wastes may also be tapped by diamond drills, used in much the same way as when employed at the surface. At the Carron Company's Collieries at Bishopbriggs, near Glasgow, bore-holes were drilled in this manner into old wastes containing water, the distance being about 120 ft. The power was derived from a small Priestman oil-engine developing $3\frac{1}{2}$ horse-power, with a consumption of 3 gallons of oil in eight hours. The speed of the engine was 150 to 250 revolutions per minute, while that of the drill was 75 to 125 revolutions per minute. The ignition in the cylinder of the engine was obtained by electric spark from a bichromate battery, which served for about 500 hours.

The speed of boring in the different strata per shift of eight hours with two men was, in sandstone, 30 ft., shale, 18 ft., ironstone, 15 ft., and coal, 47 ft. Better results could have been obtained, but the position where the drilling was being carried on being confined, short lengths of rods, 18 in., 36 in., and 54 in. long, had to be used at first, which necessitated frequent changing.

Other machines used for boring are the Burnside Safety Drilling Machine,* designed for boring long holes, for tapping flooded workings, and a machine made by Mr Robert Martin, which is really an adaptation of an ordinary miner's ratchet boring machine, and which can be worked by hand. It has been found most useful in the highly inclined workings of Niddrie Colliery, holes from 50 to 70 ft. in length being bored with no greater difficulty than attends the boring of shot holes.

Surveying Bore-holes.—Bore-holes are very apt to depart from the vertical, and may thereby give misleading results; for instance, the bore-hole at Barlow in Yorkshire, when it had reached a depth of 2120 ft., was found to have deviated 15° from the vertical, and the bore-hole at Balfour Mains in Fifeshire was found to be 471 ft., or approximately 6° , off the vertical in a total depth of 4534.6 ft. It therefore often becomes necessary to ascertain the amount of deviation. This can be measured by the aid of a clinograph or clinostat, an instrument which was first invented and used by Mr E. F. Macgeorge in the Colony of Victoria, Australia. It consists of two glass bulbs, the upper one carrying a plummet and the lower one a magnetic needle, both bulbs being filled with liquid gelatine. The needle is so arranged that it can swing freely without touching the sides of the glass bulb, and so set itself in the magnetic meridian. The small glass cylinder, terminating in the bulb at the top, is inserted through an air-tight cork and a brass capsule at the upper end. This upper bulb contains a delicate plummet of glass, with a diminutive hollow float at the top and a solid ball at the bottom, which is prevented from falling out by a delicate grating. It is carefully adjusted to

* *Trans. Inst. Min. Engs.*, vol. xxiii. pp. 75–78.

the specific gravity of the solidifying fluid used, and is so arranged that it will assume a vertical position whenever it is free to move.*

To use the instrument, six of the bulbs are placed in a bath of warm water, heated nearly to boiling point, and a brass cylinder is also heated by filling it several times with hot water. The clinostats are then put into this cylinder one after the other, and lowered into the bore-hole, where they are allowed to remain for two or three hours. By this time the gelatine will have 'set,' fixing the needle in the direction it had assumed prior to the solidification. The brass cylinder is then withdrawn, and the clinostats are examined one by one in an instrument specially designed for the purpose of ascertaining the deviation. This system is only suitable for bore-holes under 1000 ft. in depth, as beyond this depth the increased temperature prevents the gelatine from solidifying. There are a number of other methods employed for ascertaining the deviation, such as Nolten's method, in which hydrofluoric acid is used in a glass tube, the acid etching the glass and showing the deviation. At the Barlow bore-hole a clear glass quart bottle was partly filled with liquid cement and placed in a cylindrical steel case, into which it fitted tightly so as to prevent lateral movement. The case had a tightly fitting lid to protect the bottle from the great water pressure at the bottom of the hole. The case and bottle were lowered into the hole, and allowed to stand for forty-eight hours to allow the cement to set at the angle of the hole. Bore-holes may be brought back to the plumb, if deviated, by forcing an india-rubber washer down to a depth of 20 yds. or so beyond the point of deviation, and then running in liquid cement to some feet above where the hole has deflected. The cement is allowed to harden properly, when boring may again be commenced in the right direction. At the deep Fifeshire bore-hole the core-tube stuck fast and could not be withdrawn, which necessitated its abandonment. The hole had then to be deviated past this portion, and this was effected by lowering a mild steel wedge, 12 ft. long, and tapered away to a fine edge at the top so as to deflect the boring tool, into the new path for the continuation of the hole. This had the desired effect, and the boring was continued without mishap.

* For full description of this instrument, see *Mine Surveying*, by Bennett H. Brough, eleventh edition, p. 323.

CHAPTER III.

SINKING.

Preliminary Considerations.—After a coal-field has been sufficiently proved by boring, and the seams have been found to be of good quality and of sufficient thickness to be payable, the sinking of shafts to 'win' the coal has next to be considered. Sinking operations may be divided into two stages: viz., (1) sinking through the surface deposit; (2) sinking through the regular strata.

The surface deposit is often thin and firm and easily sunk through; at other times, however, it is the most difficult part of the work. This is particularly the case when a bed of running sand or mud, a thick bed of mud and boulders, or a bed of peat moss, is met with, such formations presenting considerable difficulties to shaft sinking.

Before starting to sink, several important points have to be carefully considered, among which may be mentioned the following:—

The extent of the field or royalty to be worked.

The number and thicknesses of seams to be worked.

The output to be produced per day to yield a profit on the capital invested.

The quantity of water likely to be met with.

The number of shafts required, and what their size must be.

What their positions should be as regards markets, railway communication, etc.

What faults, dykes, or dislocations exist.

Size and Number of Shafts.—The size and number of shafts required will largely depend on the first three heads: the extent of the field leased, the number and thickness of seams to be worked, and more especially the output required to be raised per day. Two shafts are the minimum required under the Coal Mines Act, 1911, as a single shaft is only allowed under very exceptional circumstances (see Rule 36). The size of the shaft will largely depend on the possible output per day, and on the number of years the coal-field is leased for, or can be exhausted in; points which may be decided approximately as follows:—

$$(1) \text{ Tons required to be raised per year} = \frac{\text{Total tons of royalty}}{\text{Number of years in lease}}.$$

$$(2) \text{ Tons required to be raised per day} = \frac{\text{Tons required to be raised per year}}{\text{Number of working days per year}}.$$

In England leases are generally for from thirty to ninety-nine years. The number of years for which leases are usually granted in Scotland is twenty, twenty-five, or thirty.

The size of the shaft will also largely depend on the depth of the seams from the surface, and the amount of water to be pumped. It will also depend upon the size of tubs or trams which are to be employed for hauling the coal, and how many tubs are to be raised on the cage at each wind. For small outputs, 300 or 400 tons per day and the depth about 200 yds., cages to hold a single tub, holding 10 to 12 cwts., may be employed, and a shaft suitable for such conditions would be 10 or 11 ft. diameter, if circular, and 14 or 15 ft. $\times$ 5½ or 6 ft. inside the lining, if rectangular in shape. These sizes would also allow for the accommodation of pumps 12 in. diameter. If two tubs are to be raised per wind, then the above sizes of shafts would be sufficient if double-decked cages, with a single tub on each deck, are used. In modern shafts such cages are rarely employed, the usual custom being to raise the two tubs on single-decked cages. The shaft accommodation required for cages holding two tubs placed tandem would depend on the size of the tubs. For instance, assume that the tub has to hold 10 cwts. (water level), then the sizes of such a tub would be 4 ft. long $\times$ 3 ft. broad $\times$ 2 ft. deep. A cubic foot of space holds approximately 50 lbs. of loose coal, and 10 cwts. = 10 $\times$ 112 = 1120 lbs.; $\frac{1120}{50} = 22.4$ cub. ft. of space required for 10 cwts. $\therefore$ 3 ft. $\times$ 2 ft. $\times$ 4 ft. = 22.4 cub. ft. A cage to hold two tubs of this size placed tandem would require to be approximately 9 ft. long $\times$ 3 ft. 9 in. broad between guides, and a circular shaft to accommodate two such cages, and allow for clearance space, would require to be 13½ or 14 ft. diameter. If rectangular in shape, it would require to be about 18 ft. $\times$ 10 ft. 6 in. Frequently in rectangular shafts the two tubs are placed abreast in the cages, and when this system is adopted the shaft would be 20 to 22 ft. long $\times$ 6 or 6½ ft. wide inside the lining. Cages with two or three decks, each deck holding two tubs, may also be used in the above sizes of shafts when large outputs are required. When shafts require to be sunk to depths of 600 to 800 yds., it is usual to make the shaft, if circular, 20 to 25 ft. diameter, and if rectangular, 26 to 28 ft. long $\times$ 11 to 12 ft. broad. Shafts of this size will accommodate cages holding four to six tubs on each deck, and also allow space for pumps, etc. For instance, the large rectangular shaft at the Mary Pit, Lochore Colliery, in Fife, is 29 ft. long $\times$ 11 ft. 6 in. broad inside the lining.

Site of Shaft.—In choosing a site for shafts, if the surface conditions are suitable, it is generally best, in the case of a large royalty, to sink as near to the centre of the field as possible, for if this site be chosen, the pits will not be so deep—except where the royalty takes the form of a basin with the deepest part in the centre—as if sunk at the extreme dip; and the length each ton of coal has to be hauled will also be less.

If the amount of water given off in the workings is very large, the shafts should be sunk so as to have a larger area to the rise than to the dip, as pumping large quantities of water from dip workings is very expensive. If, on the other hand, the workings be dry, the opposite plan is adopted, as haulage, etc., is usually cheaper from the dip than from the rise, the latter usually necessitating long inclines, which become, in time, difficult and expensive to work.

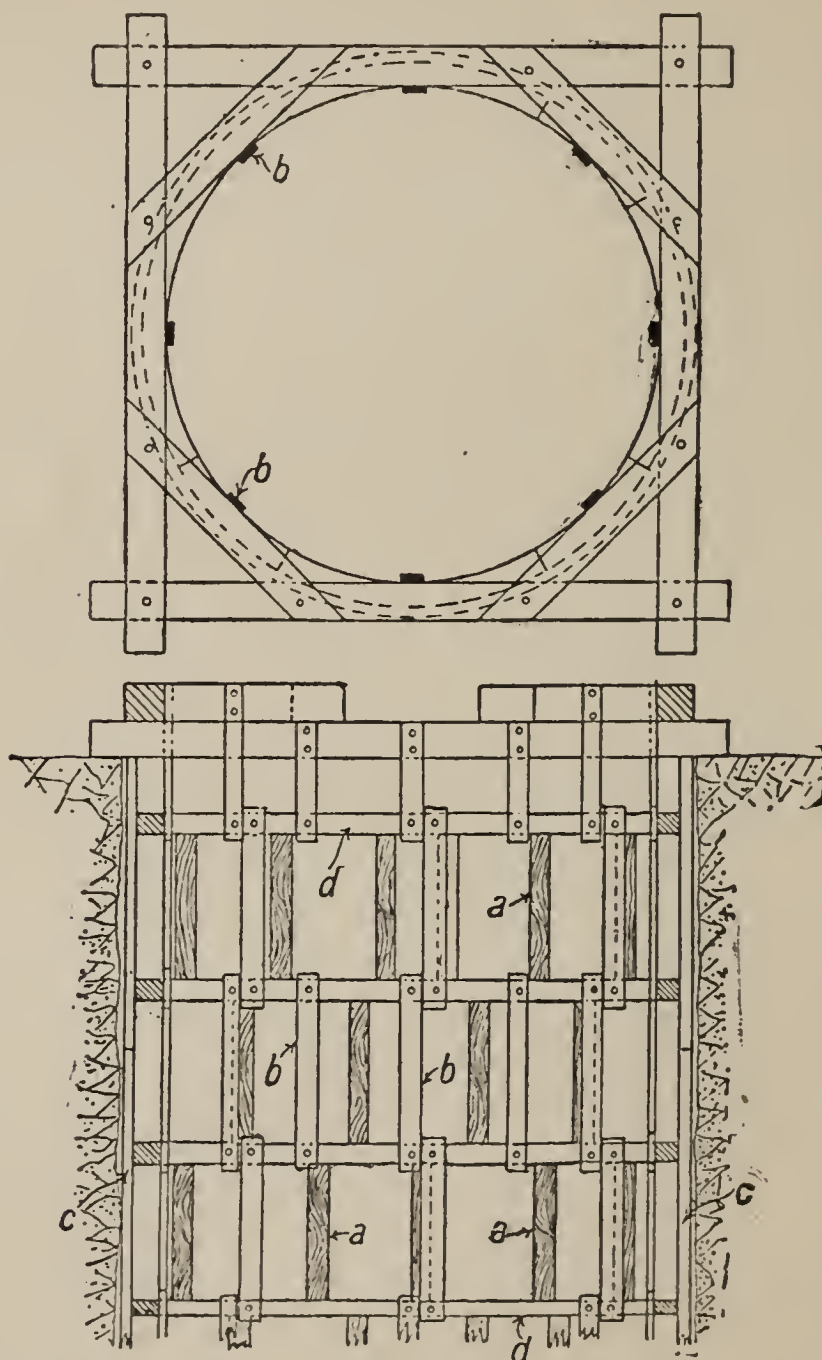
The long side of the shaft is usually sunk in the line of the dip of the seam, so as to get the main roads from the pit bottom set off to level course. If the shafts are not sunk in the line of the dip, then to get the road's level course at the pit bottom, the pavement (floor) would require to be cut on one side, while on the other side it would require to be banked up.

As a rule it is best to sink the shafts to suit the underground workings, and arrange the surface accordingly. The site should also be chosen advantageously in regard to the transit of coal by road, river, or rail, and close to a supply of good water for boilers, etc. It is a very common custom to sink the shafts close together, so as to concentrate the banking and coal-cleaning arrangements. The landowner sometimes stipulates in leases where they are to be sunk.

Form of Shafts.—Shafts may be either rectangular, circular, or elliptical. The rectangular form is almost exclusively used in Scotland and very commonly in America, while in metalliferous mining it is nearly always adopted. Circular shafts are employed in England and Wales for coal-mining, and also on the Continent. In France, and occasionally in Wales and Scotland, elliptical shafts are used, and in some instances square shafts; but this form is not to be recommended, especially in the case of large shafts. Each of these forms has its own advantages, the circular being the best adapted to resist heavy pressure, and therefore suitable for deep shafts. This form is also best suited for ventilating purposes, as there is always a certain amount of space unoccupied by the cages. The rectangular form of shaft is more economical to sink, easier lined and secured, and the space can be better utilised for winding, pumping, and other arrangements, while less material requires to be excavated. The elliptical form combines some of the advantages of both the circular and rectangular, and has been adopted in the sinking of four large and deep shafts recently sunk in Fife.

The author of a paper, read before the Institution of Mining Engineers, in discussing the merits of the different forms of shafts, says, "that in deciding the form of shaft, he fails to understand why in some districts oblong shafts are sunk in preference to circular ones, unless the object is to take out as little ground as possible. It seems desirable, in order to wind a given quantity of coal per day, that there should be the same area (over and above the space occupied by the cage at meetings) in the one shaft as in the other to admit of adequate ventilation. This consideration is in fact so important that

in many instances it is deemed desirable to sink a staple pit for a height of some 120 ft., having a holing into the shaft above and below meeting places, so that the area for ventilation may not be diminished when the cages are passing each other. The timbering in an oblong



FIGS. 34 and 35.—Plan and elevation of temporary timbering with wood curbs.

a a = punch props ; *b b* = stringing deals ; *c c* = lagging deals ;
d d = curbs.

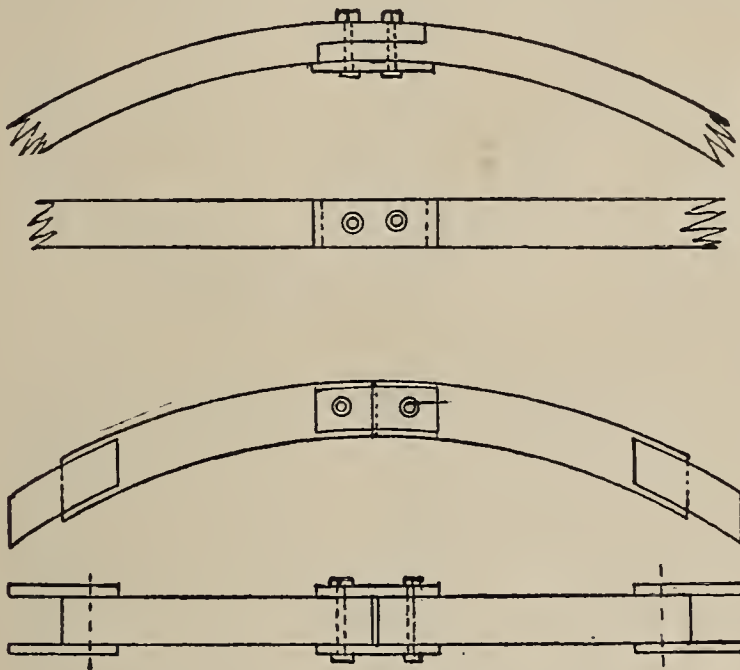
shaft will not last so long as the brickwork in a circular one, and seeing that the shaft is the one entrance into the mine through which all the men must pass in going and coming from their work, it is desirable that it should be made as safe as possible by being securely lined throughout."

Commencing Operations.—The position and size of the shaft having been pegged off, the surface soil is removed, the sides being

supported with a temporary lining, if required, until the 'rock-head' is reached, when a perfectly level bed is prepared on which to lay the first walling crib. If the shaft has to be lined with brick, the walling may be started forthwith, but very often the sinking is carried on to a considerable depth by the aid of temporary lining, and walled up afterwards.

To allow for temporary lining and walling, the diameter of the pit will require to be a good deal larger at starting than the ultimate finished size; thus a pit intended to be 15 ft. diameter inside the walling should be begun with a diameter of 17 or $17\frac{1}{2}$ ft. at least.

Where the shaft has to be supported with temporary lining, a square frame made of strong oak beams, 12 in. to 18 in. square, or 18 in. $\times$ 12 in., and firmly bolted together, is laid across the top of

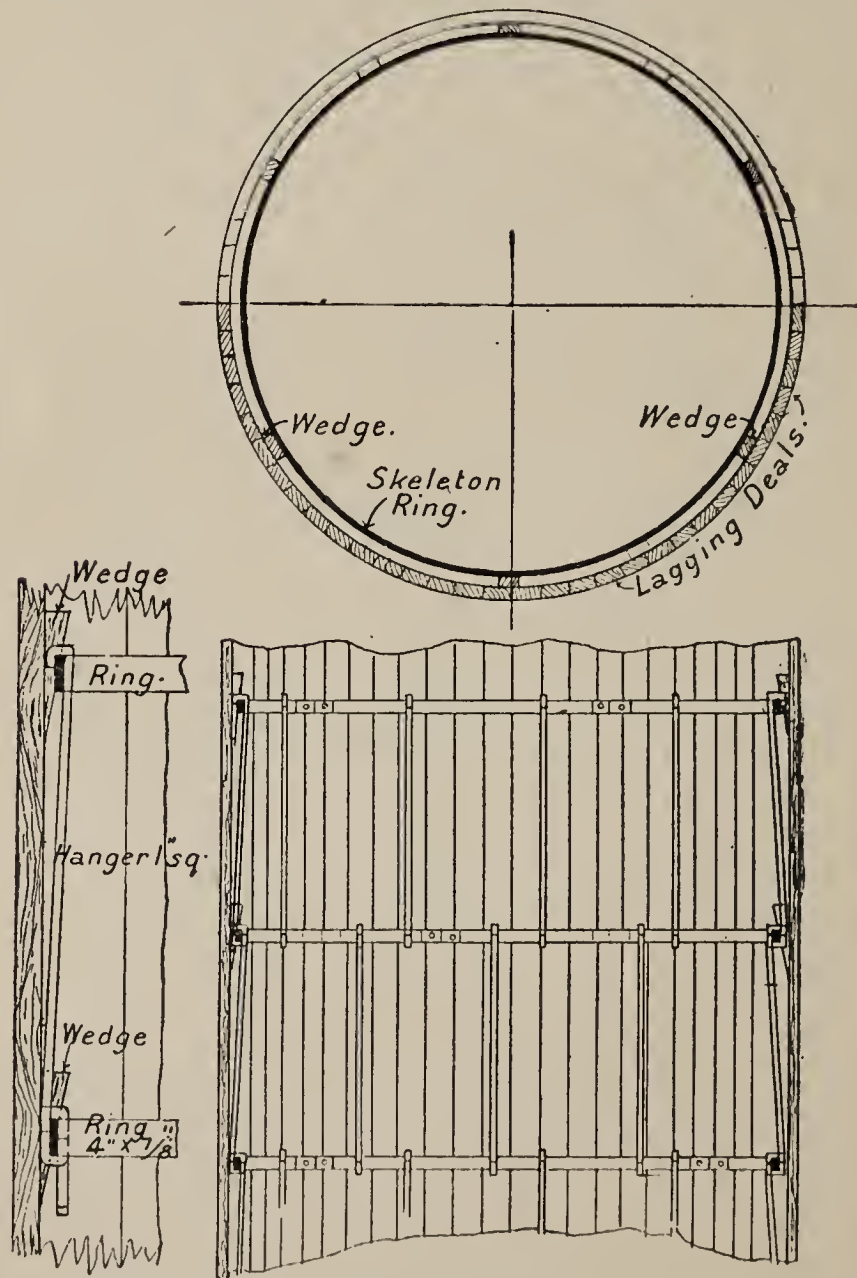


FIGS. 36 and 37.—Wooden curbs.

the pit, and to this the wood lining is secured. Figs. 34 and 35 show how this temporary lining is fixed in the shaft. A curb is placed in position, and 'backing deals' or 'lagging' of white pine boards, 6 to 9 ft. long $\times$ 9 in. to 12 in. broad, and $1\frac{1}{2}$ in. or 2 in. thick, are fitted in all round between the sides of the shaft and the curb. The curbs are of oak or other hard wood, and are made to the circle of the shaft. They are constructed in segments, and each segment is $3\frac{1}{2}$ to 4 ft. long, 6 or 8 in. wide, and 4 to 8 in. deep, or frequently they are 6 in. square. To form a ring or curb the segments are joined together either by wooden or iron cleats and bolts, or simply by bolts and a thin iron plate (figs. 36, 37). The curbs are put in from 4 to 6 ft. apart, according to the nature of the strata. When the first curb has been placed in position the lining or lagging deals are fixed in all round the circumference of the shaft. Other curbs are put in at intervals, and are kept in position by 'punch-props' inserted between them all round the shaft, more lining deals being inserted until

finally the surface is reached, and the curbs are secured to the framing at the top of the shaft as shown in figs. 34 and 35. If this temporary lining continues to any great depth, its weight may be transferred to other beams fixed in the shaft.

It has now become the custom to discard the use of wooden curbs, and to substitute iron skeleton rings in order to keep the temporary



FIGS. 38, 39, and 40.—Plan and elevations of temporary timberings with skeleton rings.

lining in the shaft. The rings are made of flat strips of wrought iron, made to the circle of the shaft, 3 to 4 in. deep and $\frac{3}{4}$ to 1 in. thick. Angle-iron, channel, or T-sections are also frequently used for the construction of such rings. A ring is made up of four to eight segments, according to the size of the shaft, and the segments are joined together by fish-plates and bolts, or in the case of flat strips the joints are sometimes overlapped and fastened with bolts. The rings are hung to one another by means of iron rods with a hook

at each end. Figs. 38, 39, and 40 show the method of using these rings. Behind these rings are placed the lagging, or backing deals, which consist of planking 2 to 3 in. thick. The iron rings are easier to handle and put together than wooden curbs.

When the walling of masonry is about to be commenced, the shaft is 'laid back' for 2 ft. or 3 ft. more than the ordinary diameter, and an even bed prepared for a walling curb; the curb being carefully laid, the walling proceeds, and is carried up for a distance of 10 to 15 yds., when another curb is placed in position. These curbs are now made of cast iron, cast in segments to suit the circumference of the shaft. When the strata are hard and strong, and a section of brickwork is to be put in, no curb need be used; the walling is simply started direct off the rock, and carried up in the usual way. A satisfactory method of proceeding is to carry sinking on for part of the week, and walling operations for the rest of the time. The method of sinking and walling simultaneously is described later. In starting to sink below the walling, the shaft is cut narrow at first, and gradually widened out to its proper width, so as to leave a ledge of rock to support the walling above (see fig. 41). When, at a later

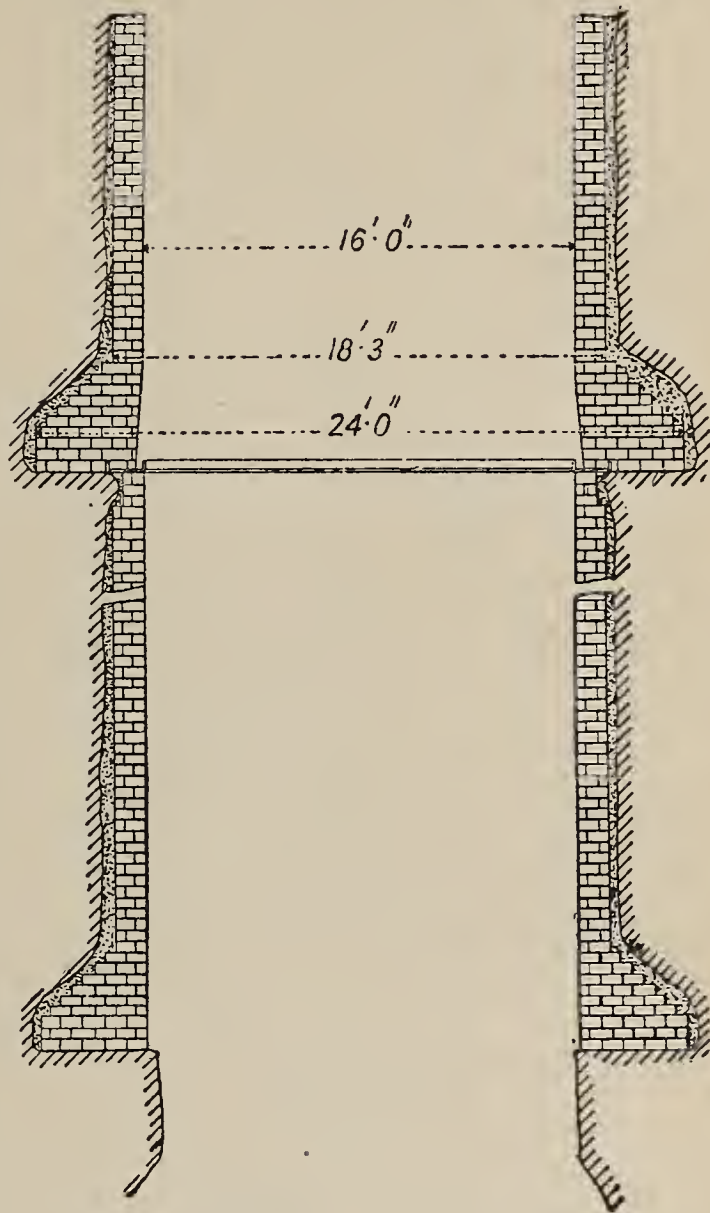


FIG. 41.—Section of walling in shaft.

stage, the walling is being built up from below, this ledge is not removed all at once, but it is taken out in sections around the circumference, these sections or arcs being then built up to the walling above until the junction is made good all round. If the strata is hard rock, only sufficient is taken out to get in a facing of brickwork to preserve the strata from weathering. The masonry is not usually built back close to the strata, but a small space is left, which is filled with fine ashes or other porous material, so that it helps to relieve the pressure arising from the sides on the brickwork. Some mining

engineers prefer to leave no space, but to build the brickwork close back to the strata. The walling is carried on by means of a 'hanging scaffold' or 'walling cradle,' made up of 3-in. or 4-in. planking, bolted together and made to fit the curve of the shaft. It is usually made with 1-in. or 2-in. clearance all round, and guides the workman in the building of the brickwork. The 'cradle' is secured to the

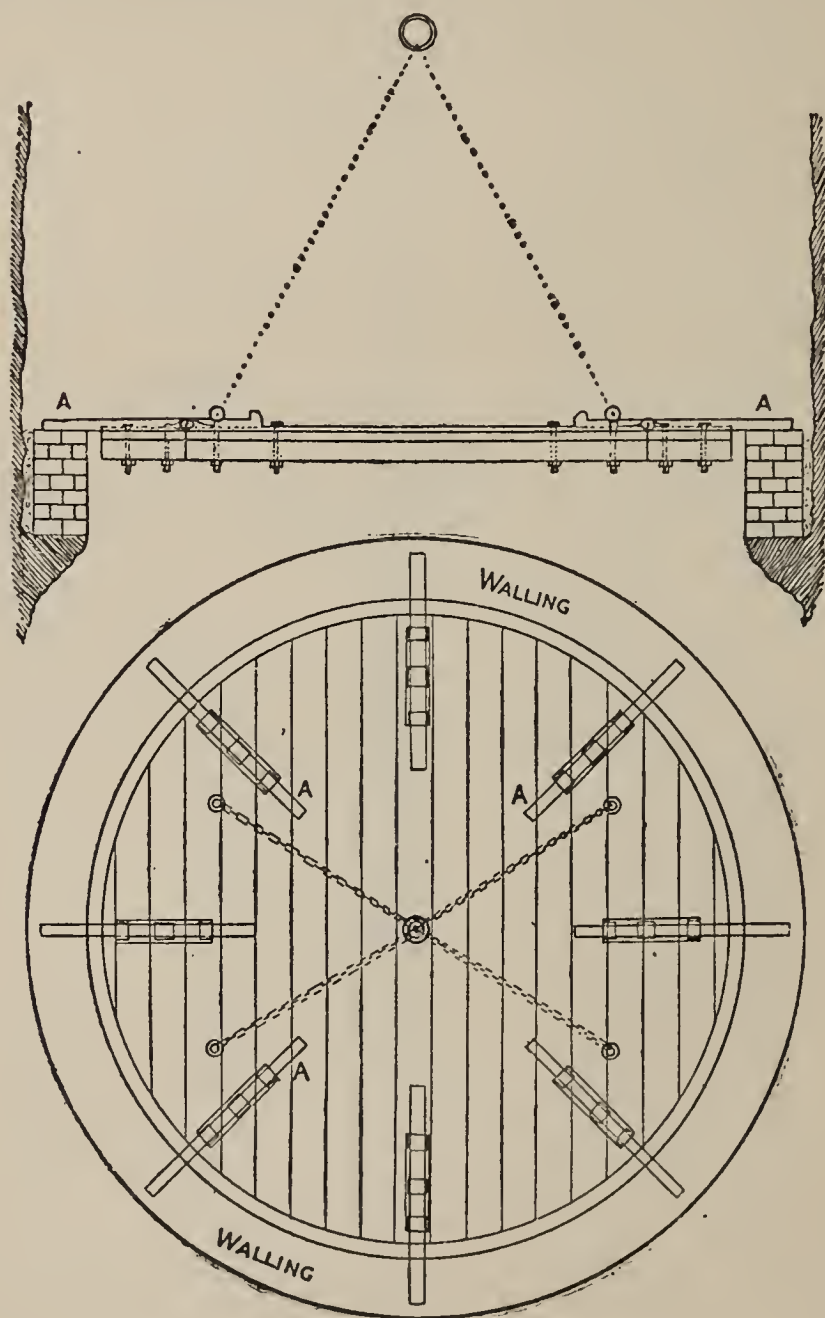


FIG. 42.

A A = resting iron bars.

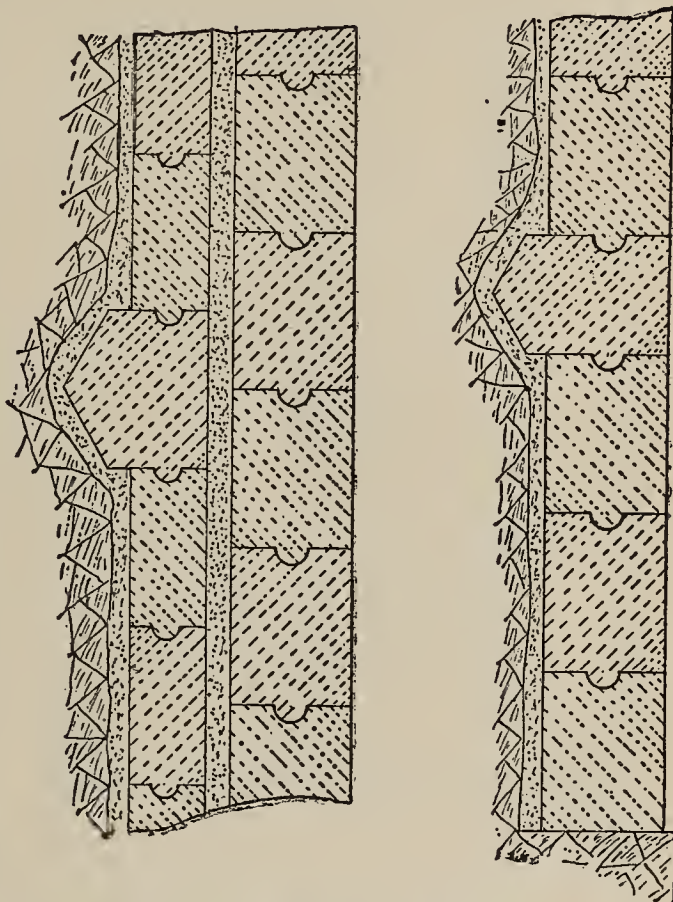
winding rope by four chains, the latter being fastened to the scaffold by eye-bolts (see fig. 42). The scaffold should be made to fold up at the centre, so that when a section of walling is completed it can be folded and slung to the sides of the shaft, instead of being raised and lowered each time.

Sometimes the walling is carried out by the help of iron sections, fixed all round the outside circumference of the walling scaffold.

The height of these sections is from 2 ft. to 3 ft. 3 in. The brickwork is placed around and in contact with it. When a circular tier of walling is thus completed, the iron section is raised and another tier of masonry placed in position. In this way the time usually spent in measuring the diameter and ascertaining whether the masonry is vertical is saved.

When the space between the walling and the strata is to be filled in with concrete or cement, the larger size of iron circle is used, so that more may be left below, as a support, until the cement sets.

Concrete Lining.—Concrete as a material for lining shafts has had a very limited application—unless in conjunction with brickwork—in British mines, but on the Continent of Europe and in America it has been frequently used for securing shafts. A number of rectangular shafts near Scranton, Pennsylvania, were relined with concrete, strengthened by expanded metal struts $\frac{3}{16}$ in. thick. The lining of concrete was put in 12 to 18 in. thick, and it was run in behind a timber box or form made to fit the shape and size of the shaft. The lining was put in in 5 or 6 ft. lengths, and the timber form was left in position until the concrete had time to set, after which it was removed and used again at a higher point in the shaft. In Germany concrete made into blocks has been used to some extent for lining shafts. The blocks are segmental in form, fluted and grooved at the top and sides, and provided with a projection to fit in the top groove of the blocks below (figs. 43, 44). The blocks should be set in good cement. As these blocks are too heavy to manipulate by hand, means are provided for lowering them into position by forming a vertical hole through them, and a horizontal one to intersect the vertical one. A ring bolt is inserted in the vertical hole, and is held in position by a cotter through the horizontal one. When the blocks are placed in the desired position these holes are afterwards filled up with cement.



FIGS. 43 and 44.—Cement blocks for shaft lining.

Armoured or reinforced concrete has been extensively applied in this country in the erection of buildings, foundations for machinery, supporting large sewers, etc., and from the success attained it would

seem to merit the attention of the mining engineer in the securing of shafts and underground roadways.

Sinking and Walling Simultaneously.—The work of sinking has in ordinary cases to be suspended while walling is being performed. In order to obviate this, a scaffold invented by Mr William Galloway is used, consisting of a wooden floor, fixed to an angle iron frame. There are two platforms, and access between them is effected by means of an iron ladder.

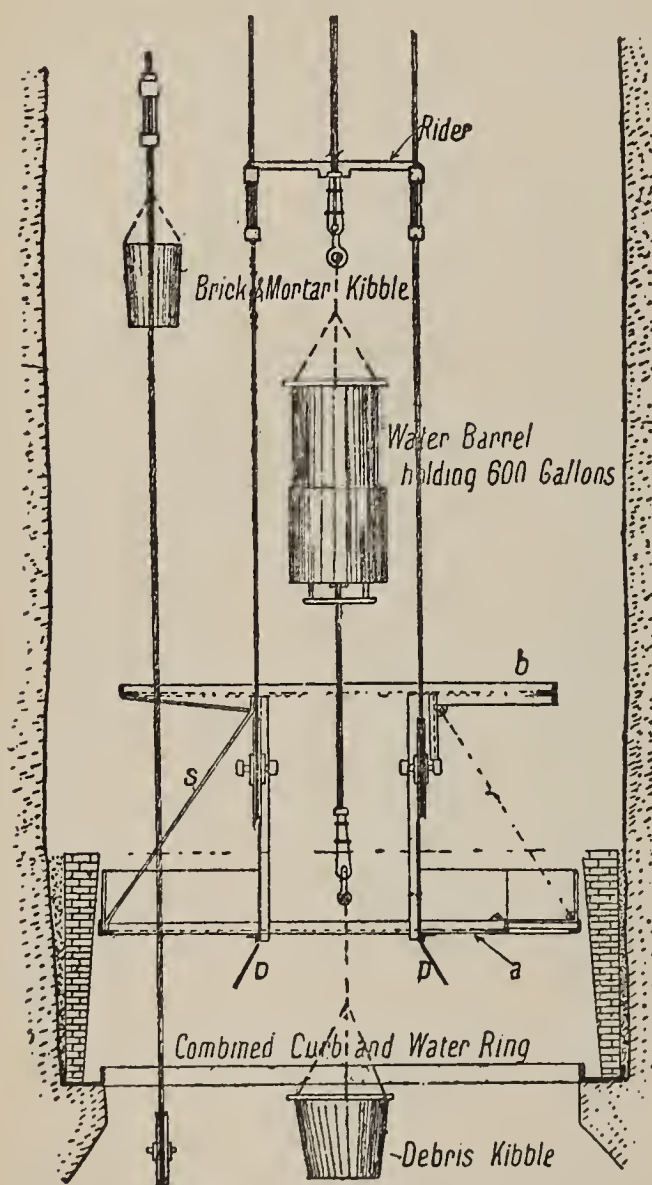


FIG. 45.—Section of scaffold used at Newbattle Colliery.

On one side of the bottom platform is placed a hinged door, which can be raised, to admit of material being raised or lowered. The scaffold is steadied by two guide ropes, and as it occupies nearly the full area of the shaft inside the walling, it saves the time usually taken in measuring the diameter every few yards. At Newbattle Colliery a scaffold, constructed on this principle, was used for putting in the brick lining to a shaft 1650 ft. deep, 20 ft. diameter, walled from top to bottom. The construction of the scaffold will be understood from figs. 45 and 46. The cradle as here used consisted of a working floor *a* and a projecting roof *b*. Between these, ample height is left for men to work. The centre of the cradle contains an opening providing space for two buckets or 'kibbles' to pass each other. This opening is fenced

by a circle of sheet iron $6\frac{1}{2}$ ft. high and $\frac{1}{4}$ in. thick, bolted to the six upright angle bars and hangers.*

The floor stage was formed by a 4 in. $\times$ 4 in. $\times$ $\frac{1}{2}$ in. mild steel angle-bar, bent to the curvature of the shaft, supported on and fastened to the bottom frame or flooring, which was constructed of 5 in. $\times$ 5 in. $\times$ $\frac{5}{8}$ in. steel angle-bars. A door or hatchway was hinged to the floor and raised or lowered by block and tackle fastened by shackles to the door or framing of the scaffold. The

door was recessed to allow of its closing the air-boxes and pipes. Hatchway doors were also left on the other side of the cradle to permit of the brick kibble being lowered through if required. The roof of the scaffold was made of 5 in. $\times$ 5 in. $\times$ $\frac{5}{8}$ in. steel angle-irons covered with $\frac{3}{8}$ in. sheet-iron plates. Where the suspending wheels carried the whole of the cradle, the framing of the roof was constructed of double 5 in. $\times$ 5 in. $\times$ $\frac{5}{8}$ in. steel angle-bars riveted together.

The floor and roof of the scaffold were connected together by four corner steel angle-bars 5 in. $\times$ 5 in. $\times$ $\frac{3}{4}$ in. bolted to the top and bottom framing. Four stays *ss* were placed to the four upright bars for the purpose of supporting and stiffening the roof framing. Two tension-rods $1\frac{1}{2}$ in. diameter were secured to the double angle-bars and to the outer ring of the floor for the purpose of strengthening and carrying the floor.

A sheet-iron ring, $\frac{3}{16}$ in. thick, was bolted to the circular framing, and formed a fence round the scaffold 18 in. high. The difference between the diameter of the pit and the upper side of this ring was about $1\frac{1}{2}$ in., *i.e.* an opening of $\frac{3}{4}$ in. existed between the brickwork and the fence. Fending plates *pp* were bolted to the bottom of the scaffold to guide the bucket into the opening of the floor.

The weight of the scaffold was about $8\frac{1}{2}$ tons, but with a full working load became about 20 tons. The scaffold was suspended in the shaft by four ropes in double purchase, the ropes being 5 in. in circumference and made of Bessemer steel wire. One end of these ropes was made fast at the surface by attachment to a heavy screw which served the purpose of adjustment in the event of the ropes riding unevenly on the drums. The other end passed round the wheel on the cradle and over another wheel on the winding frame, and thence to the crab drum. Four drums were thus required on the crab engine. These ropes between the scaffold and the pulley on the frame also served for guide ropes for a rider to run on, to guide the kettle in the shaft. In addition to these, another guide rope, made fast at the top like the others and looped down the shaft, was provided. Upon this was hung a pulley supporting a water tank and scaffold with two pulsometers, and the suction and delivery pipes. The double rope was brought through the cradle and the

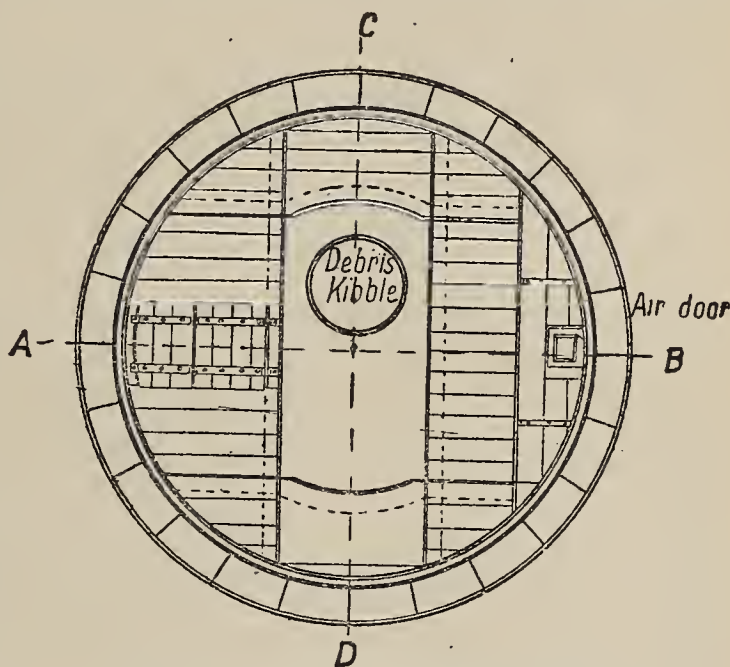


FIG. 46.—Plan of scaffold used at Newbattle Colliery.

loose end led to one of the two drums on the winding engine. The other drum was provided with a winding rope, and the bucket used with this rope ran with a rider, with the pump-suspending rope as a guide at each side, and served to supply the scaffold with bricks, lime, etc. Both drums on the winding engine were worked by clutches, so that when shots were being fired at the bottom, the pulsometer and scaffold, etc., after being disconnected from the steam pipe, could be hauled up out of danger.

The whole of the work and the details of sinking were carried on simultaneously with the walling without the men requiring to be drawn from the pit bottom, the only time the cradle required raising to the surface being when changing shifts.

The advantages claimed for this method are :—

It permits of quick winding with double buckets.

No time lost through sinkers being withdrawn for walling.

The brickwork can be done better by skilled bricklayers, at a less cost.

In forming lodgments or roads off the shaft-side, no delay need be incurred.

The suspension of a substantial structure in the shaft affords protection to the men working at the pit-bottom.

When the strata passed through are soft and unable to bear the weight of the curb and walling, holes are drilled all round the side of

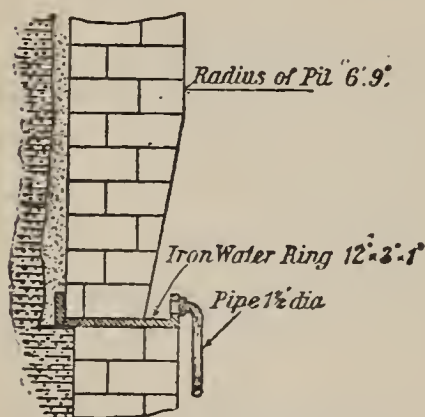


FIG. 47.—Water ring.

the pit, and strong iron rods $1\frac{1}{2}$ in. to 2 in. diameter and $1\frac{1}{2}$ to 3 ft. long, on which the crib is laid, are driven in. The walling then proceeds in the usual way. When these rods require to be used, the distance between the curbs should be very much shorter than in hard strata.

Water Rings or Garlands.—When the strata are giving off water, ring curbs will require to be put in, to protect the brickwork and to convey the water to a lodgment in the shaft, which is done by

having a pipe connection between every two curbs.

The ordinary 'garland' or 'water ring' is usually an iron curb, cast with a groove, and is often a walling and water curb combined; sometimes it is made of wood and an annular space left in it. Above each curb the brickwork is tapered back for a height of 4 or 5 ft., so that a space of 3 or 4 in. is left all round between the outer edge of the curb and the face of the walling to allow the water free access to the ring (see fig. 47). One or more pipes of small diameter are fixed to and connect the various rings, so that the water may be conveyed from the one to the other, and finally to the pump lodgment. The rings are usually put 50 to 60 ft. apart, or at points where necessary, according to the quantity of water to be dealt with. It is most important that these rings should be thoroughly water-tight. Sometimes a layer of well-puddled clay is put in behind, but

a better plan is to lay them on felt or oakum, and grout them with good cement. Instead of having the water ring at the front of the walling as described, some prefer to have them formed in the centre or behind the brickwork. This prevents the rings from getting silted up by material, such as small pieces of coal, etc., falling down the shaft, which can hardly be avoided when winding is going on. Openings or 'pigeon holes' are left in the brickwork all round the circumference of the shaft opposite the rings so as to enable the ring to be cleaned when required.

Bricks and Mortar for Walling.—In walling, a good quality of brick is most necessary, the most satisfactory being good alumina fire-bricks. Except in very small shafts, ordinary shaped bricks can be used; the mortar employed should be made from good hydraulic lime, mixed with 'mine dust' from calcined iron-ore heaps, which makes a better binding material than sand. Sometimes a mixture of cement and lime is used mixed with clinker ashes, ground fine, from the boiler fires. The proportions commonly used to make mortar are one part of good lime to two or three parts of sharp sand, or one part of lime to two parts of sharp sand and one part of ground ashes or coarsely ground coke, or one part lime to two and a half or three parts of 'mine' dust.

The general arrangement of the walling, guides, etc., of a circular shaft is shown in figs. 48 and 49, the shaft being fitted with wood guides.

Cost of Walling.—This will depend on the size of the shaft, the thickness of the brickwork, the quality of the bricks, mortar, etc., used, and the number of walling curbs placed in the shaft. The approximate cost of walling per fathom may be found as follows:—

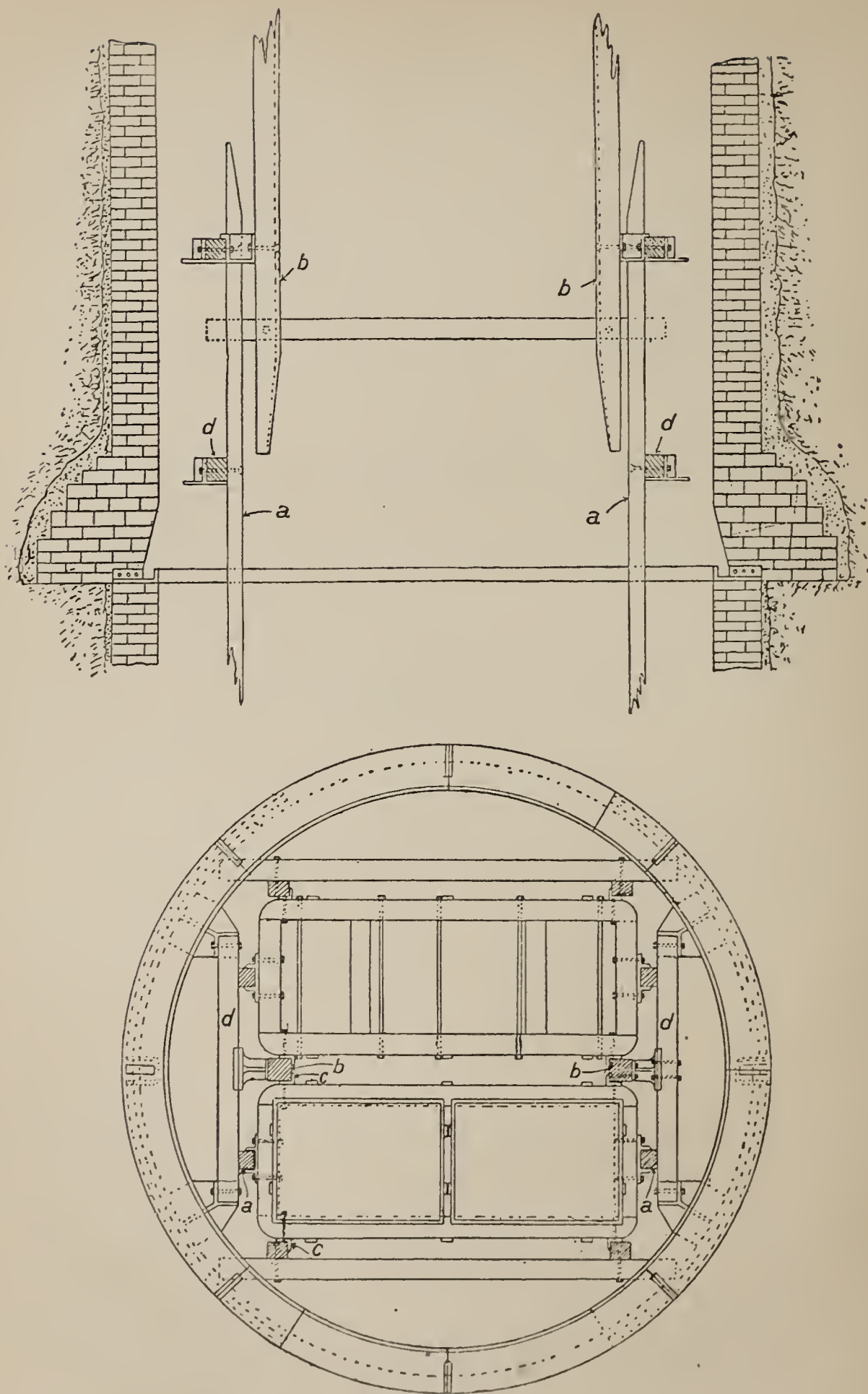
Let C = cubic yards in 1 fathom of walling.
 D = external diameter in feet of walling.
 d = internal " "
 l = number of feet in 1 fathom. "

$$C = \frac{(D^2 - d^2) \times .7854 \times l}{27}$$

Assume the average thickness of brickwork is 14 in., and the internal diameter of the shaft 15 ft., then

$$C = \frac{(17.33^2 - 15^2) \times .7854 \times 6}{27} = 13.10 \text{ cub. yds.}$$

The standard size of brick used for ordinary building is 9 in. $\times$ 4½ in. $\times$ 3 in., and the number of bricks, without mortar, in a cubic yard is 384, or with mortar, approximately 330, or 3 cub. yds. of brickwork is usually taken at 1000 bricks. Good well-burnt bricks will cost 21s. to 25s. per 1000 delivered at the colliery, and mortar will cost, when made, approximately 8s. per ton. A third of a ton of mortar will be required roughly per cubic yard of brickwork. Tak-



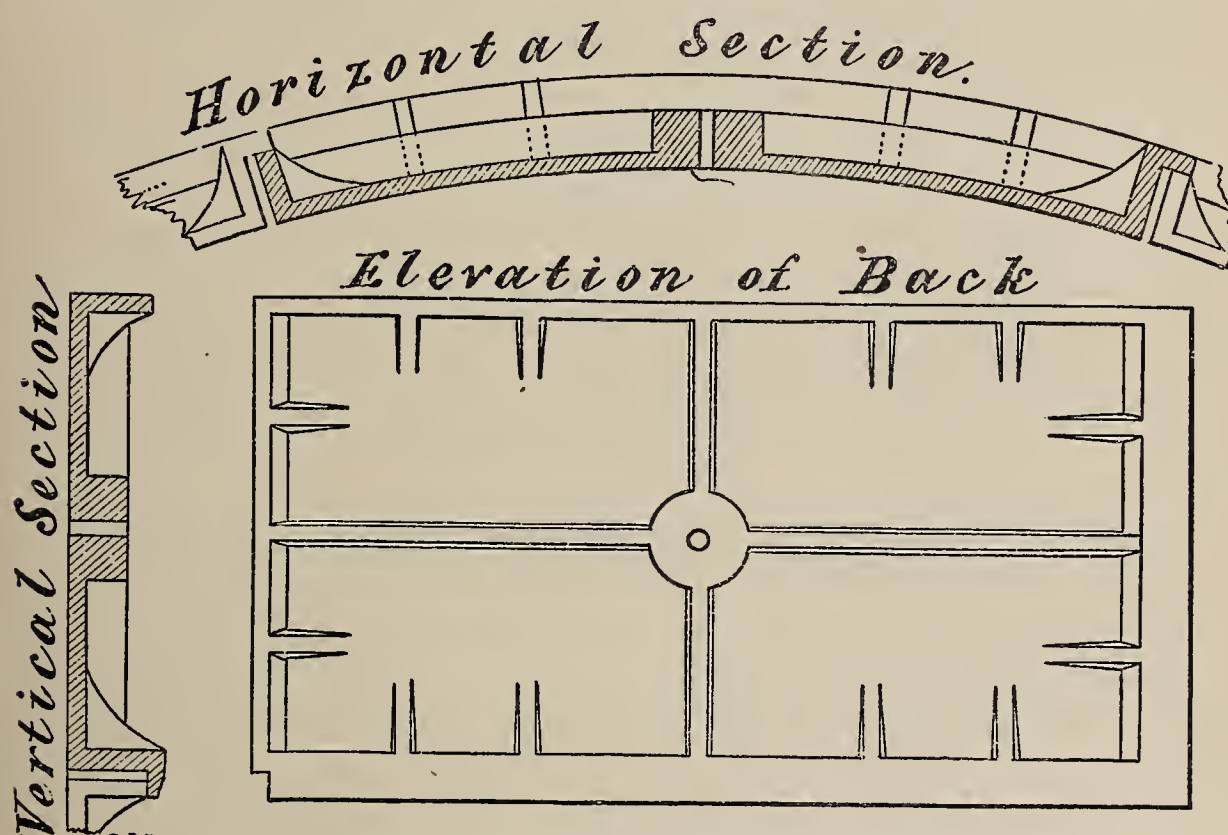
FIGS. 48 and 49.

a a = shaft guides ; *b b* = auxiliary guides at surface ;
c c = angle-iron shoes for auxiliary guides ; *d d* = buntons.

ing these prices as the basis, the approximate cost per fathom would be:—

13.10 cub. yds. of brickwork=4360 bricks at 22s. 6d. per 1000,	£4 18 0
4.33 tons of mortar at 8s. per ton,	1 14 6
Labour of bricklayers at 4s. per cubic yard,	2 12 4
Attendance on bricklayers in shaft and at surface at 5s. per cubic yard,	3 5 5
Total,	<u>£12 10 3</u>

This works out at £2, 1s. 8d. per lineal foot. To this would have to be added the cost of walling curbs, laying of curbs, extra cost of widening out the shaft for each curb or section of brickwork, etc.



FIGS. 50, 51, and 52.—Segment of tubbing.

A near approximation of the cost of walling alone, including labour, will be 15s. per cubic yard of brickwork if set in ordinary lime mortar; 18s. per cubic yard if set in cement mortar; and if concrete lining is put in, 10s. to 12s. per cubic yard. These prices will of course vary as the cost of material varies from time to time.

Tubbing.—When large quantities of water are met with in the strata to be sunk through, cast-iron tubbing is used to keep back the inflow, to save pumping, and keep the shaft dry. The tubbing (figs. 50, 51, 52) is made in segments suitable to the radius of the pit, the depth and thickness of each segment varying according to the pressure to which it is to be subjected. The rings or segments are built up from a 'wedging curb,' carefully laid on a smooth bed cut round the pit. The wedging curb is a box-shaped ring of cast

iron 1 in. to $1\frac{1}{2}$ in. thick, 6 in. to 8 in. deep, and 12 in. to 14 in. wide, and made in convenient sections. At the point where a wedging curb is to be placed, the shaft is 'shorn' back to admit its being placed in position, and a small annular space is left all round. When the curb has been laid in position and securely wedged, the tubbing is built up from it, the joints between every two rings of tubbing being filled with soft fir sheathing or thin sheet lead so as to secure a water-tight joint. The spaces left between the tubbing and the strata are usually filled with good concrete or cement. Fig. 53 shows a section of tubbing with wedging curb in position. The segments of tubbing are generally cast smooth on one side (the side facing the shaft area), and a small hole left in the centre to relieve the pressure of water behind, while it is being built up, these holes being afterwards carefully plugged up. In what is known as the German system of tubbing the segments are cast with flanges and rings on the inside. The flanges are machined, and the segments are secured to one another by bolts and nuts, no wedging being necessary. The advantages claimed for the German tubbing are that it is easier placed in position and adjusted to a circle, does not require to be so heavy, and that the segments can be cast much deeper than in English tubbing. German tubbing has been employed at the new sinkings in Kent, and also at other shafts in Britain, with success. A corrugated form of cast-iron tubbing has been employed on the Continent with excellent results, as it is both strong and light.

Thickness of Tubbing.—The thickness of tubbing required for any given shaft will depend on the diameter of the shaft and the depth or head of water to be resisted. It is not easy to calculate the exact pressure that tubbing may have to resist in any given shaft, as the head of water may not represent the total pressure. If there is any movement of the strata, considerable pressure may result from this cause. Other factors to be taken into consideration are: effects of acid water setting up chemical action and the resulting corrosion of the metal; also the moist air containing gases in up-cast shafts will have a deleterious effect on tubbing; and, further, it is difficult to cast tubbing uniformly throughout. All these points have to be taken into consideration and allowed for in determining the thickness of metal required. A number of different formulæ are given by different authorities for calculating the thickness of tubbing, but none of them are more than approximately correct. The thickness may be found from the following rule:—

$$\text{I. } t = \frac{RpM}{f}, \text{ where } t = \text{thickness of tubbing in inches.}$$

R = radius of pit in inches.

p = pressure per sq. in. in lbs. = head in ft. $\times$.434.

M = factor of safety (6 to 10).

f = resistance of tubbing to crushing.

(For cast iron f = 100,000 lbs. per sq. in.)

Example.—What thickness of the tubing would be required for a shaft 15 ft. diameter, and a head of water 50 fms.?

$$t = \frac{90 \times 50 \times 6 \times .434 \times 10}{100,000} = 1.17 \text{ in.} + \frac{1}{8} \text{ in. (to allow for wear)} = 1.295 \text{ in.}$$

N.B.—This would be the thickness required at bottom of tubing. The well-known formula given by Greenwell * is :—

$$\text{II. } x = .03 + \frac{PD}{50,000}.$$

Where x = thickness of tubing in decimals of a foot ; P = depth in feet ; D = diameter of pit in feet ; $.03$ = a constant.

Taking the same data as above—

$$x = .03 + \frac{300 \times 15}{50,000} = 0.12 \text{ ft.} = 1.44 \text{ in.}$$

The thickness worked by these two formulæ would of course be for the bottom length, say 20 to 30 ft., of tubing ; each section or length would have to be calculated separately.

The following thicknesses of tubing were used in actual practice, which may be useful to compare with the example given : Shaft 20 ft. diameter ; depth 101 to 303 ft. ; tubing varied from $\frac{7}{8}$ in. to $1\frac{7}{8}$ in. thick ; shaft 16 ft. diameter, depth 110 to 225 ft., thickness $1\frac{1}{2}$ in. ; 225 to 340 ft., thickness $1\frac{3}{4}$ in. ; 340 to 450 ft., thickness 2 in.

Corrosion of Tubing.—Certain substances held in solution by water are very injurious to iron surfaces, and to prevent corrosion the tubing often receives a coating of tar or of some hard varnish. In up-cast shafts where furnaces are used for ventilation, the fumes and gases given off by the furnace are also very injurious, and the only remedy is to line the tubing with good fire-brick to protect it.

Coffering.—Another method of shutting off water from the shaft is to use coffering, which is simply a brick wall with a space in the centre, this space being filled in with good cement, which makes a water-tight walling.

A third method of coffering differs somewhat from that usually employed. The wall in this case (fig. 54) is of $4\frac{1}{2}$ -in. brickwork, about 3 ft. at a time being built, while in front were placed sheet-iron plates $a a$ 14 to 16 wire gauge.† In front of this, brickwork 20 in. thick was built up, leaving a space of $1\frac{1}{2}$ in. between the inner and outer walls, to be filled in with cement, in which the iron plates were also embedded. As the pressure of the water became less the thickness of the walling was gradually reduced from 20 in. to 15 in., and finally to 10 in., the space for cement being kept in continuous lengths. The best results are obtained when the water in the pit is allowed to follow the work, so that the cement sets under water, but this is not always practicable. The brickwork in

* *Mine Engineering*, by C. Greenwell.

† *Trans. Fed. Inst. Min. Engs.*, vol. viii. pp. 18, 19.

the walling should also be set in cement. At the bottom of the water-bearing strata a short length of solid walling should be put in wherever possible, and the water conveyed through it by pipes. It is generally found that when water is met with, a considerable quantity percolates through the brickwork for a few days, but as a rule it ultimately becomes quite dry. The cost of coffering of this description was, in a particular instance, £6, 6s. 8d. per vertical foot

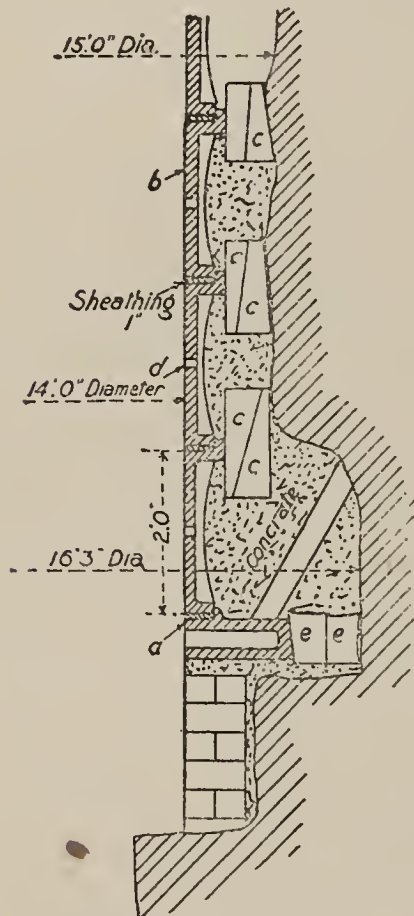


FIG. 53.

a = wedging curb.
b = tubbing.
c c = wedging.
d = plug holes.

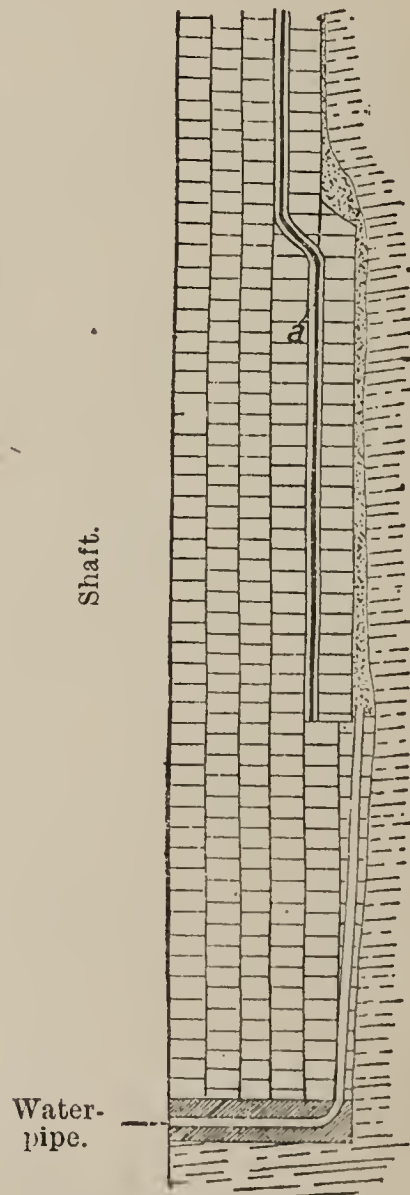


FIG. 54.—Coffering.

over a distance of 240 ft., which included the cost of labour and material, and the exact cost of enlarging the shaft sufficiently to admit of the extra thickness of walling. The cost for tubbing of the same length was estimated at £12 per foot for a shaft 18 ft. diameter. But £12 per foot is a very low estimate compared with some instances where this method has been adopted; as, for instance, at Shireoaks Colliery, where a total depth of 170 yds. of tubbing in a shaft 12 ft. diameter cost no less than £60, 2s. per yard.

During the sinking of the shafts at the Maypole Colliery great difficulties were experienced from extensive water feeders, one of

which delivered 90,000 gallons of water per hour. The coffering arrangements, which were of a complicated and costly nature, have been described in detail in the *Journal of the British Society of Mining Students*.*

Sometimes the coffering is put in simply with a brick wall set in cement and backed with good puddled blue clay. In a shaft at the Plas Power Colliery, South Wales, the following method was adopted.† The shaft was first lined with a 9-in. dry brick wall (*a*, fig. 55), set on wood curbs as the sinking proceeded until a suitable foundation was reached. At this point a cast-iron wedging curb was put in on which to carry up the coffering. This wedging curb consisted of a flat, hollow, cast-iron ring, formed of two plates connected by vertical webs and made in segments of convenient size. Thin sheets

of wood were put in between the segments to make the joints water-tight, and thin wedges driven in between the curb and the sides of the shafts until there was no room for any more. Holes were then made in this wooden packing by means of iron spikes, and as many hard wood wedges as possible driven in. When the wedging curb had been thus fixed, three courses of brick work, set in Roman cement, were built up. Each of these courses (*b b b*, fig. 55) were separated by a space of $5\frac{1}{2}$ in., into which

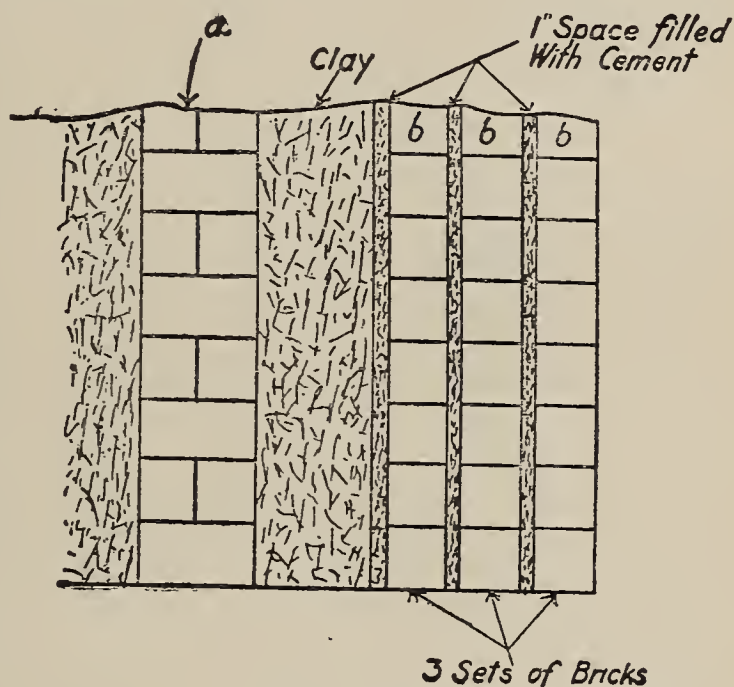


FIG. 55.

cement was run. In the space next the dry brick wall vertical wooden pipes, perforated every 3 in. down their length, were placed to lead the water into plug boxes. When three courses, *i.e.* $13\frac{1}{2}$ in. high, had been laid, bricklaying was stopped, and puddled clay rammed in tight between the inner dry wall and the coffering, the holes in the wooden pipes being plugged as the clay rose. When the coffering rose so high that water ceased to accumulate behind it, the wooden pipes were filled up with stones and hydraulic lime run in. The coffering was then completed, and the holes in the plug blocks filled in with wedges. On the Continent coffering composed of concrete is frequently used for shutting off water in shafts, but this system has so far not found much favour in Britain. The concrete used is made from ground slag and broken bricks, which is cheaper than cement, and found to stand quite well.

* Vol. xxi. p. 65.

† *The Colliery Guardian*, 27th March 1903.

Thickness of Coffering.—The thickness of coffering (*i.e.* if it is composed simply of brickwork set in cement) to resist a given head or pressure of water can be found by the formula, $t = \frac{R p M}{f}$.

The crushing strength of good brickwork set in cement may be taken at 2000 to 2500 lbs. per sq. in., which would be the value of f .

Example.—A shaft is 16 ft. internal diameter (*i.e.* inside coffering), what thickness of coffering would be required to resist a head of water of 120 yds., allowing 6 for a factor of safety?

$$t = \frac{(8 \times 12) \times (120 \times 3) \times .434 \times 6}{2500} = 35.97, \text{ say } 36 \text{ in.}$$

This would be the thickness for the lowest section; the thickness would be decreased, say every 10 yds., as the head of water decreased.

The thickness of coffering required to resist a given pressure of water can be found by means of the formula given in the case of tubbing, except that a different value must be assigned to f , viz., the crushing strength of good brickwork set in cement, which may be taken as about 2500 lbs. per sq. in.

Cost of Sinking.—This will depend on the nature of the strata to be sunk through, the quantity of water to be dealt with during sinking, the size of the shaft, and the price of labour. In the particular districts labour will probably cost from 6s. to 15s. per cubic yard excavated. The cost of sinking and lining the two shafts at Harris's Navigation Colliery, each 17 ft. diameter, was as follows * :—

Average cost per yard for sinking 50 yds. in shale near pit bottom.

	Without Pumps.	With Pumps.
Labour, ...	£9 8 2 per yd.	£10 2 4 per yd.
Material (Stores, etc.), .	2 11 4 ,,	3 0 4 ,,
Total, ...	£11 19 6 ,,	£13 2 8 ,,

In hard rock the cost was £44, 13s. 2d. per yard, using pumps, and the cost for walling in the same shafts, for 18-in. brickwork and two curbs per yard, was £11, 7s. 10d., or an average of £1, 3s. 10d. per cubic yard of brickwork, which seems rather high. To the above cost would require to be added the cost of guides and fixing, which would be about 25s. per yard, if iron or steel guides are used; if wire rope guides, the cost would be 10s. to 12s. 6d. per yard.

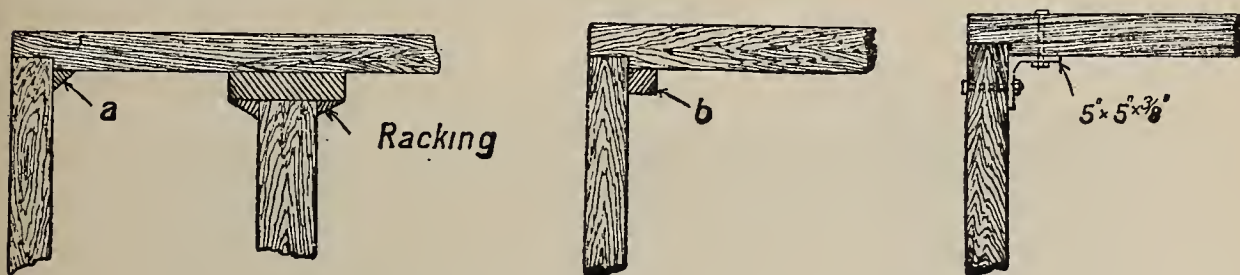
The cost of sinking a shaft 20 ft. in diameter, 600 yds. deep, with 18-in. brickwork, in a Welsh colliery, has been given by Professor Galloway as follows :—

* *Trans. Inst. Civil Engs.*, vol. lxiv. p. 23.

Wages and salaries, £14,400, ...	£14,400
2,500,000 bricks @ 35s. per 1000, ...	4,375
950 tons of lime @ 10s. 6d. per ton, ...	498
3500 tons of sand @ 5s. ,, ,, ...	875
600 tons of coal @ 6s. ,, ,, ...	180
Timber for mid-brattice, etc., ...	1,000
Stores, lighting, etc., ...	3,000
Contingencies, ...	3,000
	£27,328

This gives an average of £46, 10s. 4d. per yard for sinking and walling alone. The rate of sinking averaged 8·3 yds. per week, and 3000 to 4000 gallons of water per hour had to be dealt with.

Sinking Rectangular Shafts.—The procedure is much the same in sinking rectangular shafts as in the sinking of circular ones. The shaft having been properly pegged off, the surface soil is excavated, as already described, and the sides supported with temporary wood, until a convenient depth or the rock head is reached, when the first

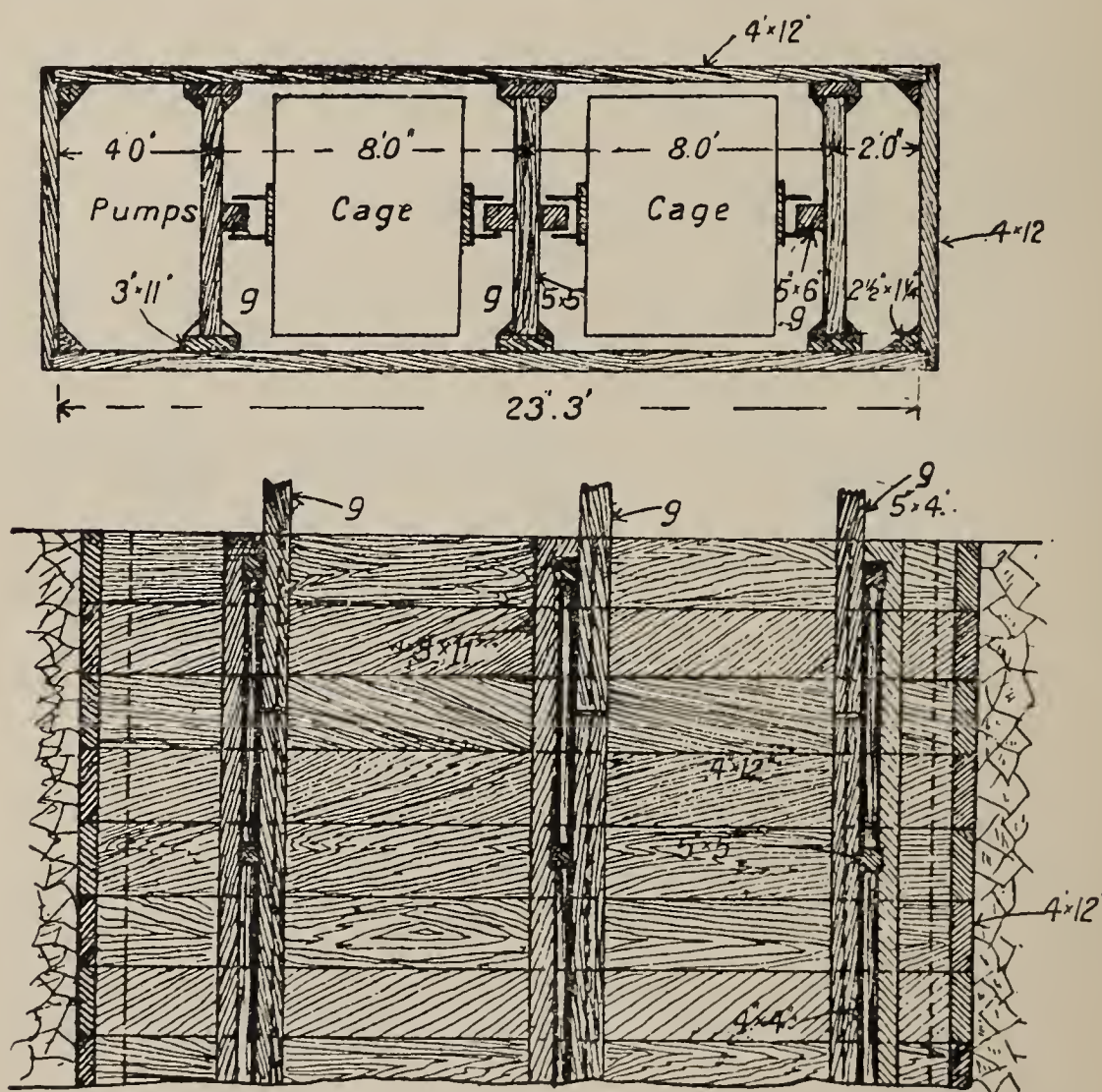


FIGS. 56, 57, and 58.—Fixing timber.

set of 'barring' is usually put in, great care being taken to square the bed for it, and to set it level. The first set having been properly adjusted, others are built up above it, to 3 ft. or thereabouts above the surface, to afford sufficient height for emptying the material excavated, and also to prevent water flowing into the shaft, the back of the barring being well puddled with good blue clay for this purpose.

The sets of barring are fitted into the shaft, either cut square, with an ordinary 'butt' joint, and corner rackings (fig. 56) put in to bind them together; or the more general practice is to notch them into each other (fig. 57), which makes a neater and stronger job. Corner rackings are also used square or triangular as at *a*, *b*, and angle irons are also occasionally employed for this purpose (see fig. 58). They are neat and strong; and they have the further merit of lasting very much longer than wood. The barring when put in position should be well and tightly wedged at the corners, and also opposite each bunton; the spaces behind the barring should be well packed with some light material, branches of fir trees for preference, to ensure efficient drainage. Figs. 59 and 60 show the plan and elevation of a rectangular shaft, and illustrate how the lining,

buntons, etc., are fixed. The sizes of wood used for barring (lining) vary according to the nature of the strata passed through. In ordinary strata not giving off much water, barring of white or red pine, 9 in. $\times$ 4 in., is used at the surface, and 9 in. $\times$ 3 in. in the rest of the shaft, where the strata are good. Where the pressure is great and the shaft large, or where loose material has to be passed through, the barring may be 9 in. $\times$ 5 in. or 12 in. $\times$ 6 in. The corner rackings are usually made from wood 2½ or 3 in., and



FIGS. 59 and 60.—Plan and elevation of rectangular shaft to suit cages with two tubs abreast.

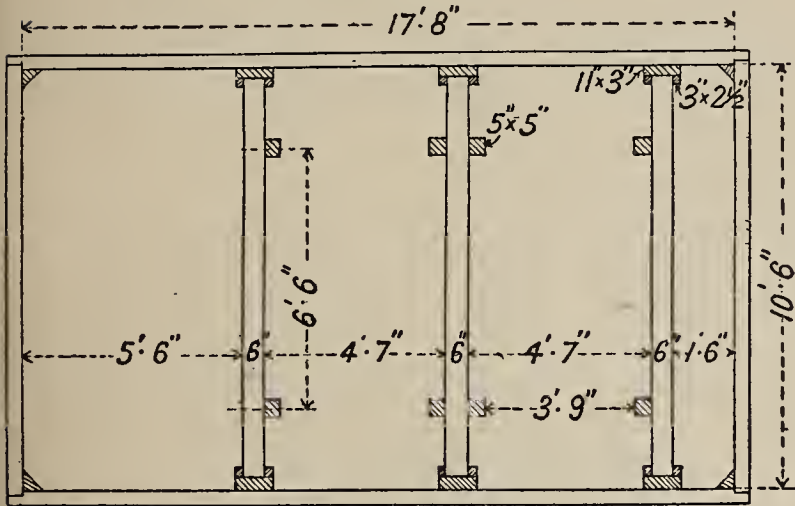
cut diagonally to form two rackings, or 2 in. square ; if angle iron is used it may be 5 in. $\times$ 5 in. $\times$ ¾ in., or 5 in. $\times$ 4 in. $\times$ ½ in.

Wood lining in down-cast shafts lasts, on an average, about fifteen years, but in up-cast shafts the average is much shorter.

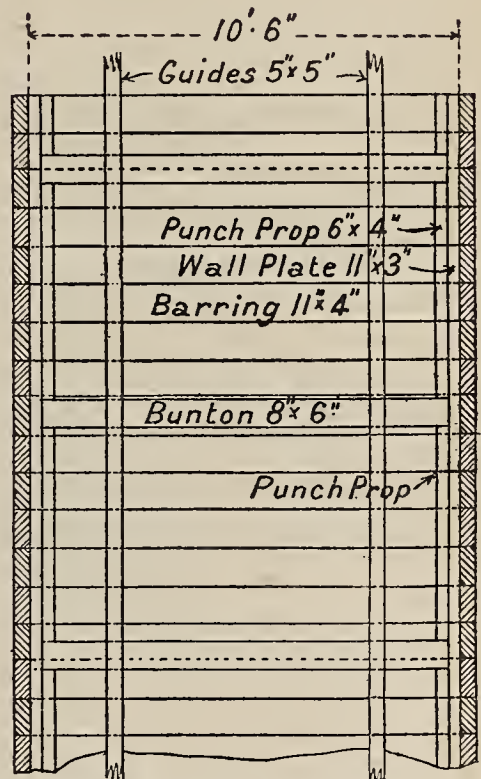
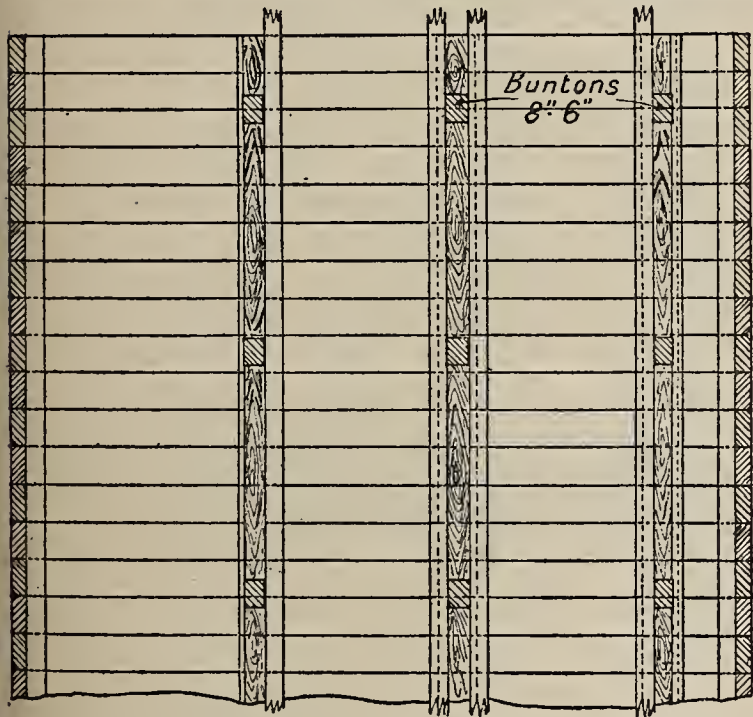
When the barring is fitted in it is further strengthened by wall-plates and buntons, the former being put opposite each bunton, and the latter themselves being put in at right angles to the barring, the perpendicular distance between them varying from 3 ft. to 6 ft., according to the strata passed through, but averaging 4 ft. in modern shafts. The buntons may be of either white or red pine ; the sizes

used are 5 in. $\times$ 5 in., 6 in. $\times$ 6 in., or 8 in. $\times$ 6 in., or for small shafts 8 in. $\times$ 3 in.

It is now the general practice, especially in large shafts, to put 'filling-in pieces' or 'punch props,' the width of the buntion and 2 to 4 in. thick, between the buntions at each end, and also at the centre of each of the latter, these 'punch props' giving greater strength and stability.



PLAN & ELEVATIONS OF
RECTANGULAR SHAFT.



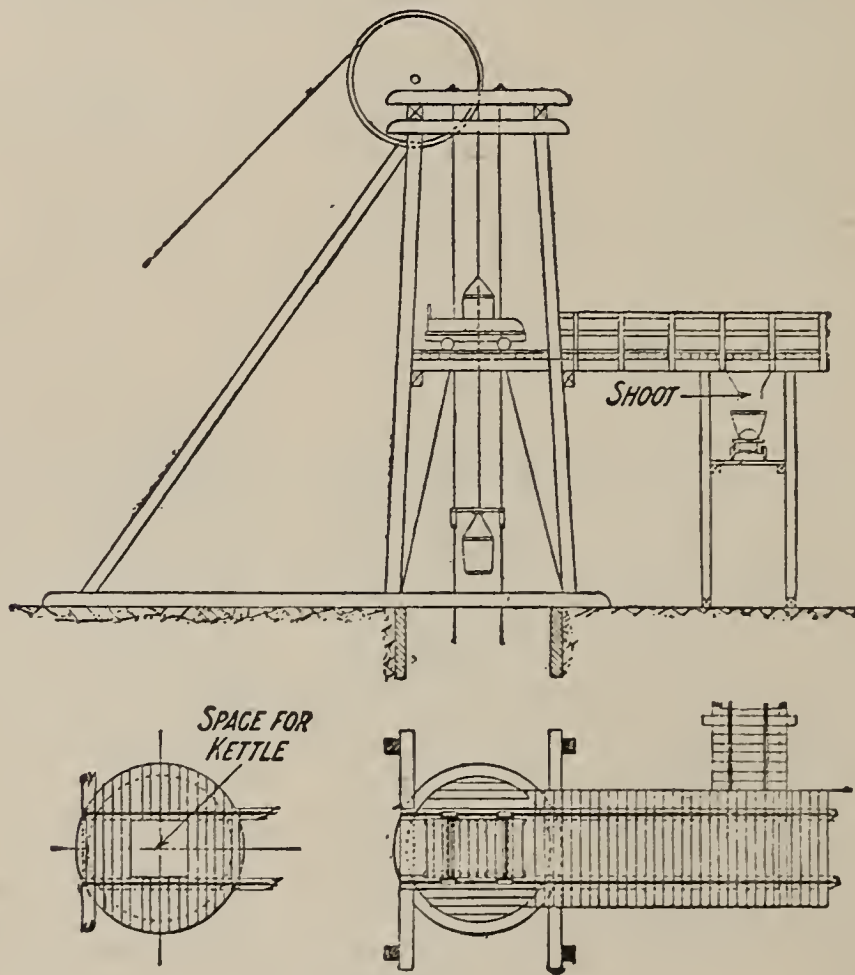
FIGS. 61, 62, and 63.—Plan and elevation of rectangular shaft to suit cages holding two tubs placed tandem.

Rectangular shafts are now generally lined from top to bottom. This gives additional security to the shaft, and facilitates the fixing of the buntions and guides.

After the surface soil has been sunk through it is usual to erect a windlass or steam crane, but a windlass is only suitable for small shafts, and can only be economically used for depths of 15 or 20 yds., beyond which it is better either to employ a temporary sinking engine, or to erect the permanent winding engines at once. For

large shafts steam cranes are much used for the earlier sinking, as they give more power than a windlass, besides being speedier and safer to work with, while the bucket can be swung clear of the shaft, and landed at any desired point for tipping.

If a temporary engine is used for sinking, it should be placed in such a position as not to interfere with the erection of the permanent winding engine, otherwise much delay may be caused. The temporary engine is often erected as close to the shaft as possible, so that the permanent engines may be laid down behind it, and the erection of screens, etc., may be proceeded with while sinking is going on.



FIGS. 64, 65, and 66.

This saves time and enables coal to be dealt with immediately the shafts are sunk. The sinking engine is sometimes placed in such a position that it can be afterwards used for haulage purposes underground.

When sinking, the shaft is usually covered over, only sufficient space being left for the bucket to pass through. When the kibble is tipped at the top of the shaft without the aid of scaffolding, a strong beam is laid across the pit, to which 'sliding' deals are fixed, to prevent the bucket from catching the mouth of the shaft, and also to make it 'strike' easier if no bogie or chain is used. Very often a bogie is used for receiving the kibble when it arrives at the surface, made so that it entirely covers the shaft (figs. 64, 65, 66),

and prevents anything from falling on the men at work at the bottom. Sometimes a chain or rope fixed to a beam on the pithead frame is used to swing the kibble clear of the shaft (see fig. 67). Another method of closing the top of the shaft is by means of folding-doors with rails on their upper sides. The accompanying illustrations (figs. 68 and 69) show the arrangement used by Professor Wm. Galloway while sinking the Llanbradach shafts.* The two

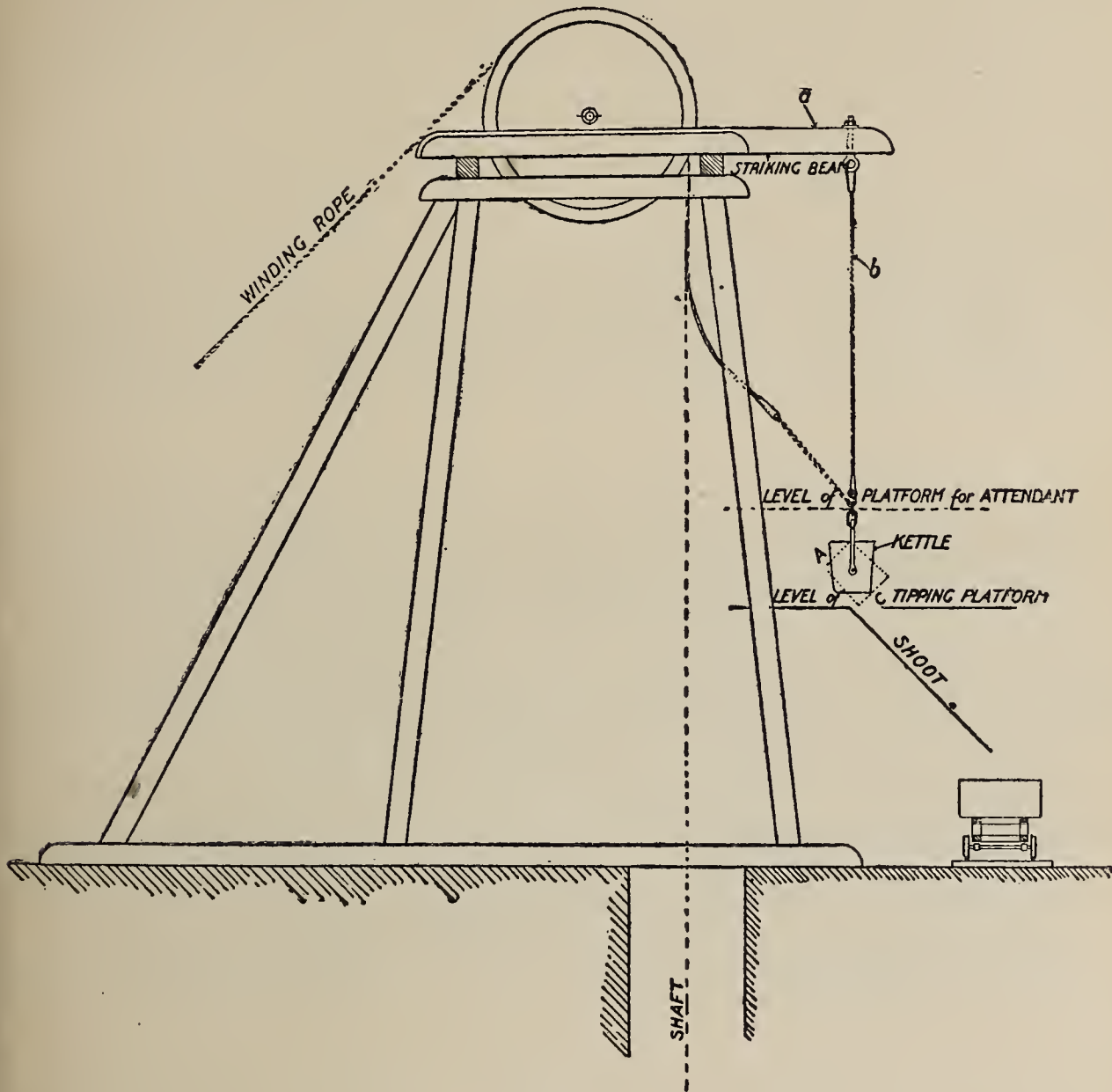
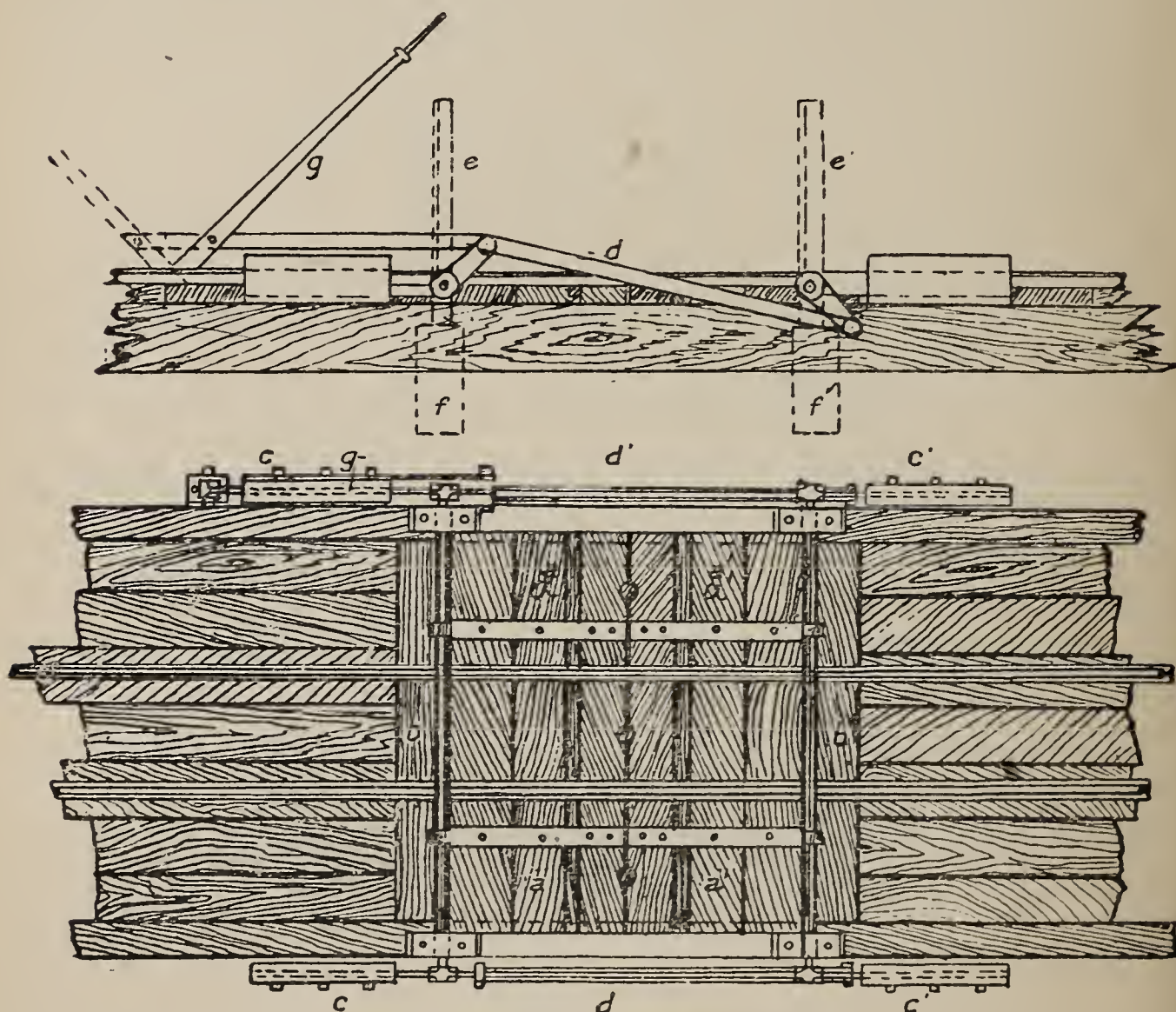


FIG. 67.

folding wooden doors are held together by hinges $a'a'$, which are keyed on to shafts bb' . Balance weights, ccc' , are attached (two to each door), and these are connected by rods dd' , through cranks on the two opposite shafts, so that the doors open and shut simultaneously when the hand lever g is drawn backward or pushed forward respectively; ee' shows the position of doors when open, and the balance weights ff' will then be in the position shown in fig. 68. If the doors are shut when the winding rope is in the shaft, the two

* *Lectures on Shaft Sinking*, p. 7.

guide ropes and the winding rope pass through three holes on the centre line of the door. A beam is put across the shaft directly below the balance weights, which are boxed in to prevent the possibility of any accident. The rods, levers, cranks, and balance weights are also boxed in above, and only the lever *g* projects through longitudinal slots in the cover. In using this apparatus, when the bucket is at the surface, the doors are closed, and a tipping waggon



FIGS. 68 and 69.—Galloway system.

into which the contents of the bucket are emptied is run on, without taking the bucket off the winding rope. The waggon is then withdrawn, the doors opened, and the bucket is ready to descend. In this system the balance weights constitute a danger, as the position they occupy is directly over the open shaft, and should they become detached they would fall down the pit, and probably cause a serious accident. At a sinking near Nottingham this actually occurred, and caused the death of two workmen. To obviate this danger the balance weights may be discarded altogether, and the doors operated by a rope passing over a pulley, with a balance weight attached at

the end, clear of the mouth of the shaft. Another system is to open and close the doors by the aid of a small horizontal steam engine, placed close to the shaft, and worked through crank rods as described for the Galloway doors. Whatever method is adopted, great care must be taken to let no loose material, such as stones, bolts, etc., fall down the shafts.

Preparing the Wood.—All wood, such as barring, buntons, racking, etc., should be prepared at the surface, ready to be sent down the pit as required, as this saves much labour, wood being difficult to cut and dress in a confined shaft. A hand winch, with a thin wire rope and a large muzzle attached, should be kept ready for lowering the wood to the sinkers as required.

Cost of Lining a Rectangular Shaft.—This will vary with the size of shaft, the kind of strata passed through, and the sizes of timber used. It should also be remembered that the price of timber varies from year to year, certain kinds of timber being now much dearer than they were a few years ago. The approximate cost per fathom for lining a shaft 18 ft. $\times$ 10½ ft. would be as follows:—

Side barring, 12 sets, red pine, 18 ft. 0 in. $\times$ 12 in. $\times$ 3 in., at 2s. per cubic foot,	£5	8	0
End barring, 12 sets, red pine, 10 ft. 6 in. $\times$ 12 in. $\times$ 3 in., at 2s. per cubic foot,	3	3	0
Wall plates, 6 sets, red pine, 6 ft. 0 in. $\times$ 12 in. $\times$ 3 in., at 2s. per cubic foot,	0	18	0
3 buntons (taken 6 ft. apart), 10 ft. 0 in. $\times$ 8 in. $\times$ 6 in., at 2s. 6d. per cubic foot,	1	5	3
6 filling-in pieces (punch props), 6 ft. 0 in. $\times$ 6 in. $\times$ 3 in., at 2½d. per lineal foot,	0	7	6
12 buntion rackings, 6 ft. 0 in. $\times$ 3 in. $\times$ 3 in., at 1d. per lineal foot,	0	6	0
4 corner rackings, 6 ft. 0 in. $\times$ 5 in. $\times$ 5 in. $\times$ 3½ in., at 2d. per lineal foot,	0	4	0
Wedges and nails,	0	5	0

Total cost per fathom = £11 16 9
or £1, 19s. 5d. per ft. of shaft.

This estimate does not include the price of cage guides, which would have to be added on. Frequently white pine is used for the barring, and when this is done the cost of these items would be reduced about 20 per cent., but red pine is preferable, as it has greater strength and will last much longer. The above sizes are used for ordinary strata, but if broken metals are encountered heavier timber, especially barring, would require to be used.

Disposition of Labour and Tools Required.—The sinking is generally carried on continuously during the twenty-four hours, with the exception of Sundays, and the number of men employed on each shift varies according to the size of the shaft, etc. For a rectangular shaft 23 ft. $\times$ 7 ft. the number of men employed would be twenty-one, *i.e.* seven men on each shift; for smaller shafts three or four

men on each shift would be sufficient. In a circular shaft 18 ft. to 20 ft. in diameter, sixteen to twenty men should suffice for each shift; an average labour allowance being one man for every 15 to 20 square feet of sinking.

The tools used in sinking are spades, shovels, picks, jumpers, 2 ft., 3 ft., and 4 ft. long, and 1 in. to $2\frac{1}{2}$ in. across the mouth; single and double-headed hammers, stemmers, cleaners, saws, axes, screw-keys, and porting-bolts. Two kibbles will also be required, each to hold 10 to 20 cwts. of material, and also a water-barrel to hold 15 to 30 cwts. of water.

Rate of Sinking.—This will vary according to the nature of the strata sunk through, quantity of water to be dealt with, other difficulties which may arise during sinking, and the machinery and tackle provided for the work. The rate of sinking under ordinary conditions may vary from 6 to 10 or 11 yds. per week, but 8 yds. per week would be counted a fair good average.

At the Lady Windsor Colliery, South Wales, the down-cast shaft, 19 ft. internal diameter, was sunk and walled to a depth of 308 fathoms at the average rate of 28·8 ft. per week; the up-cast shaft, 17 ft. diameter, was sunk and walled at the rate of 26·25 ft. per week.

Sherwood Colliery, Nottingham, was sunk at the rate of 24 ft. per week, shaft 20 ft. diameter. This included eight weeks taken to put in 303 ft. of tubbing.

For rectangular shafts the speed is very similar; two shafts 25 ft. $\times$ 10 ft. inside the lining being sunk at the rate of 8 yds. per week.

Number of Men Employed in Sinking.—This will almost wholly depend on the size of shaft being sunk. For large shafts, 18 to 20 ft. diameter, fifteen to twenty men or more may be employed on each eight-hour shift. In a rectangular shaft, 25 ft. $\times$ 10 ft., fourteen to sixteen men would be employed on each eight-hour shift. A fair average, we think, would be to employ one man to every 15 to 20 sq. ft. of area of the shaft, according to the nature of the strata to be sunk through.

Special Methods of Sinking.—When thick beds of running sand, gravel and water, or peat moss or mud and boulders, are met with, either at the surface or further down, the following special methods of sinking may be employed:—

Sinking by pile driving.

Sinking by brick drum, iron or steel cylinders, rectangular iron cylinders.

Sinking by combination of brick drum and iron or steel cylinder.

Sinking by compressed air; *e.g.* Triger system.

Sinking by boring or drilling out the shaft; *e.g.* Kind-Chaudron system.

Sinking by freezing the strata; *e.g.* Poetsch or Gobert systems.

Pile-driving, or sinking by piles, is one of the commonest and easiest methods of sinking through a moderately deep bed of sand met with at the surface.

The piles used for this purpose are usually of red or white pine 12 to 15 ft. long, 9 in. broad, and 3 in. thick, sharpened and shod with iron at the bottom, to facilitate driving, while at the top a hoop of iron is shrunk on to prevent splitting while the pile is being driven down (see fig. 70). Before starting to sink, a strong framework of timber, of the size required, is fitted together, and laid down on the site where the shaft is to be sunk. The first set of piles are then driven in all round it 'skin for skin,' the commonest method of driving them being by hand, the man using a large mallet. If they cannot be driven easily by the mallet or hammer, a 'monkey' may be used, or the necessary pressure applied by means of a hydraulic ram. As the piles of each succeeding set are driven in they are firmly supported by side and end bars, and buntons placed at convenient distances apart, as shown in fig. 70. When a pit has to be sunk by this method, it must be commenced very much larger than the finished dimensions required, especially if the depth of sand to be sunk through is considerable, as every set of piles put in reduces the size of the shaft by at least 18 in. Sometimes the piles are driven inclined outwards to keep the size of shaft from being reduced too much, but by this method it is more difficult to keep the barring perpendicular. If any space is left between the piles and the walling or barring, it should be filled up with good cement or concrete, and the piles withdrawn if possible. Sinking by piles is an expensive method, and sometimes not a very successful one, if the sand is very quick, or when the strata are watery and mixed with boulders. The limit of depth that can be sunk through by pile-driving is about 60 or 70 ft., but it is more efficient when the depth does not exceed 30 to 40 ft.

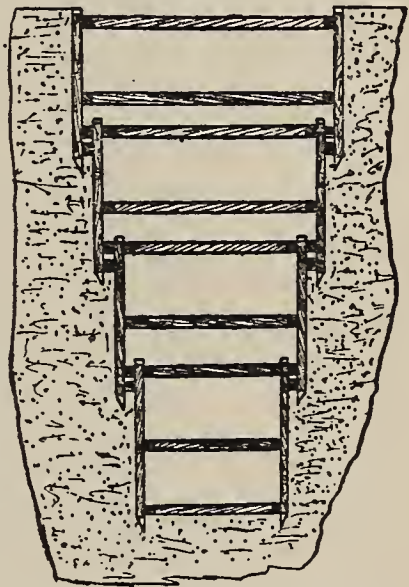


FIG. 70.—Pile-driving.

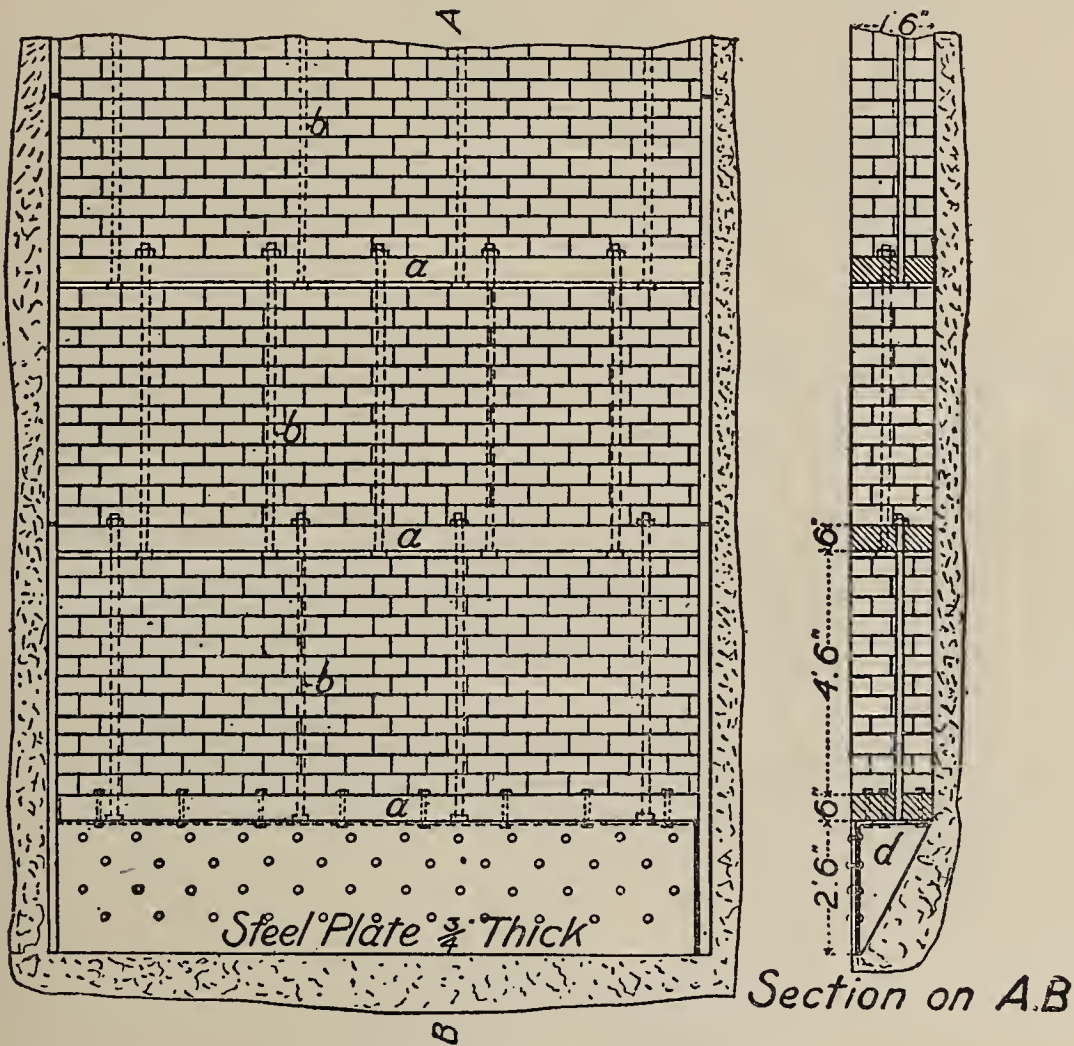
At Bowburn Colliery,* near Durham, a shaft was recently sunk by piling. Glacial drift clay and sand covered the surface to a depth of 156 ft. 9 in., the sand bed commencing at a depth of 67 ft. 6 in. from the surface, and it was determined to sink through this by means of piling. The pit was started 25 ft. diameter to ensure a shaft 13 ft. diameter when finished. The first 89½ ft. were sunk by ordinary methods, and after penetrating 18 ft. into a bed of loamy clay the piling was started. Timber cribs 6 in. square were placed in position, and backing deals 7 in. wide × 1¼ in. thick were inserted all round the shaft and tied with stringing deals 7 in. × 1¼ in., the cribs being spaced 1 ft. 9 in. apart, and held in position by punch props 4 in. diameter. The lowest crib of this timbering was placed at a

* *Trans. Inst. Min. Engs.*, vol. xxxii. p. 385.

depth of 101 ft., the lower portion being lined with grooved and tongued deals 7 in. $\times$ $1\frac{1}{4}$ in. A crib 6 in. square was then suspended by chains, leaving a space of $2\frac{1}{2}$ in. for the passage of the piles; and below this a similar crib was laid, but not hung, the segments being fastened together above and below by iron plates 3 ft. long $\times$ 3 in. wide $\times$ $\frac{1}{2}$ in. thick, bolted with six through bolts, and going down with the piling. The piles were of pitch pine 7 in. wide $\times$ $2\frac{1}{2}$ in. thick, were scarfed for a length of 6 in., and black-lead applied to make them travel easily. The piles were driven downwards by a pitch pine ram worked by three men, the sand being removed as the piles descended. The heads of all the piles were hooped with iron $2\frac{1}{2}$ in. wide $\times$ $\frac{1}{2}$ in. thick. The last crib having reached the sandstone, the usual method of sinking was again resumed till a depth of 6 ft. was reached, when a good crib bed was formed, and the walling started from this. The walling consisted of two rings of fire-brick lumps 12 in. long $\times$ 9 in. wide $\times$ 3 in. thick, with cement grouting 3 in. thick between them. The space behind the walling was rammed with well-puddled clay, and the timber piling was left in behind the walling.

Brick Drums.—Sinking through running sand is often done by what is known as the 'Drum' method. When this system is adopted, a curb of wood, 14 in. to 18 in. broad and 6 in. thick (figs. 71, 72), is laid down on the site to be sunk through; in a rectangular shaft a square frame is used, firmly morticed and bolted together. The curb or frame is carefully adjusted with a straight-edge and spirit-level, to get it perfectly horizontal. Upon this curb a tier of dry brickwork is placed, until a height of 3 or 4 ft. has been reached, when another curb is placed in position and secured to the first by strong tie-bolts of wrought-iron; more brickwork is then placed in position until the drum begins to sink by its own weight. Workmen stand in the centre and excavate the sand as it sinks, taking care that the bottom of the drum is 2 or 3 ft. in advance of the excavation, and at the same time keeping it in a horizontal position, for one of the great difficulties in sinking by this method is keeping the drum truly vertical, so as to prevent it from canting. To reduce the friction of the drum during descent, and also to keep the brickwork intact, a close lining of planking is fixed all round its outer circumference, the joints being made water-tight. As the sinking proceeds more brickwork and curbs are placed in position, and secured by tie-bolts as before, until the solid ground is finally reached, where a perfectly level bed must be made for the reception of the first permanent walling curb. If the ground is not loose enough for the drum to sink easily, a cutting edge is fixed to it, bevelled on the inside and fitted with an iron shoe (fig. 72). For the purpose of keeping the drum plumb and sinking evenly, it is sometimes lowered by strong screws and nuts attached to beams at the surface, or it may be lowered by ropes and winches. This method affords better control over the operations.

Cast or Wrought-iron Cylinder.—Brick drums are liable to stick, and iron cylinders or drums are therefore preferable. These iron drums are made in segments, cast to the curvature of the pit and strengthened by horizontal and vertical ribs, like ordinary tubing, with the exception that the ribs are cast on *the inside*, so that the outside of the metal is left smooth and offers as little resistance as possible when sinking through the sand. The joints are carefully rendered water-tight by putting sheet lead between the



FIGS. 71 and 72.—Hanging brick drum.

a a = wood curbs ; *b b* = hanging rods ; *d* = cutting shoe.

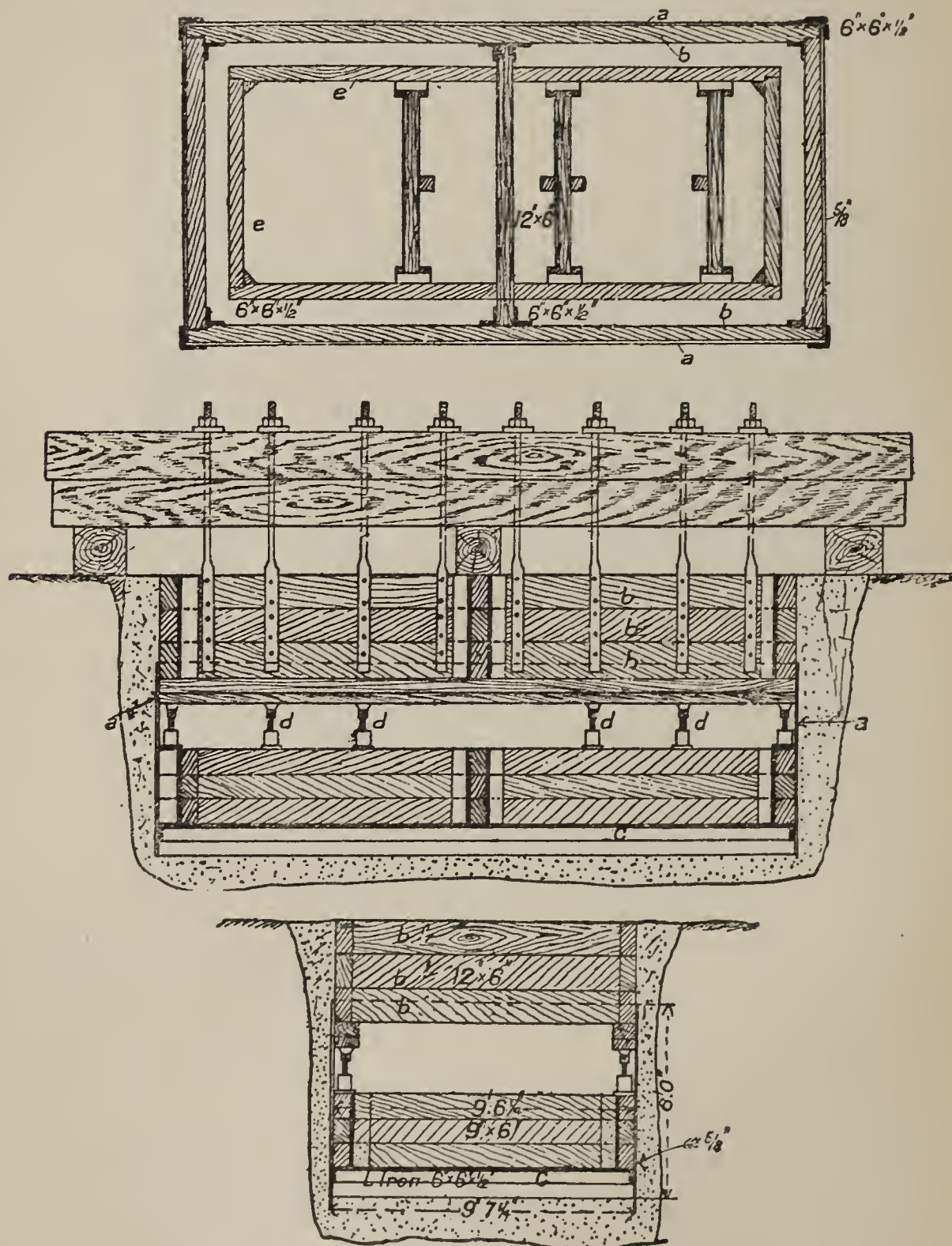
flanges and firmly bolting them together, a cutting edge being attached to the bottom in much the same way as to the brick drum.

In Scotland, a method of sinking rectangular shafts through running sand with iron tanks is adopted which is somewhat similar to the above system. The tanks are usually made of ordinary boiler-plate fastened together by lap joints and rivets.

Figs. 73, 74, 75 show the arrangement of the tank. The tank is made of segments of wrought-iron boiler plate $\frac{1}{2}$ in. or $\frac{5}{8}$ in. thick, and 6 ft. deep. The segments are firmly joined at the corners with overlapping pieces, with which they are riveted 'flush.'

When about to sink, the drum is set in the position of the proposed

shaft, and the sand or mud removed from the inside, until a sufficient depth has been reached to put in two or three sets of barring, *b b*, of 12 in. $\times$ 6 in. pitch pine. The tank is then forced down into the



FIGS. 73, 74, and 75.—Iron tanks.

a=iron cylinder ; *b*=outside barring ; *c*=angle iron ;
d=screw-jacks ; *e*=inner barring for shaft.

sand by means of screw-jacks. Near to its foot is an angle iron to which three sets of oak barring 9 in. $\times$ 6 in. are fixed, for the purpose of giving a seat to a number of screw-jacks which are used to lower

the tank as the sinking proceeds. When the first three sets of barring at the top have been fixed to strong beams placed across the shaft (fig. 74) by means of hangers and nuts, the tank is pressed down a further distance of a foot or so, and the sand dug out until room has been made to add another set of barring below that already fixed, to which the last set is hung by means of wrought-iron straps and corner angle irons. This operation is continued until the tank has been sunk its full depth of 6 ft., when the same procedure is gone through as before, pressing down the cylinders by means of the screw-jacks and adding barring at the bottom as required until the bed of sand has been sunk through.

By this method beds of running sand of almost any depth can be sunk through with safety and rapidity. The tank has, of course, to be made a great deal larger than the finished size of shaft, as will be seen from fig. 73. When the rock head has been reached, the regular barring is built up to form the shaft inside the first temporary barring, and a space, which may be filled in with cement or concrete, left all round. In sinking either with this or any other sort of drum, the greatest difficulty is to keep it vertical, and this can only be done by maintaining a careful watch on the screws when lowering the drum and removing the sand. It is better to make the drum a little wider at the bottom, 1 in. or 2 in., than at the top end, as this will assist it to descend more easily than if it were the same width throughout.

Combination of Brick Drum and Iron or Steel Cylinder.—In sinking through running strata with the brick drum alone it has been found, as already stated, that there is always a considerable tendency for it to go off the perpendicular, to stick fast altogether, and to give an excessive amount of 'skin' friction. To overcome these difficulties, especially in very soft surfaces, a steel cylinder is now frequently used in conjunction with the brickwork. This arrangement has been additional strength to the drum, reducing the skin friction and making it sink more easily. Two shafts were sunk on this principle under very difficult conditions at Olive Bank Colliery, near Edinburgh, and as they form a typical example of this system of sinking, they may here be described more fully.

There are two shafts, 70 ft. apart, each 14 ft. in diameter inside the finished brickwork. The pits have been sunk to a depth of 152 fms., and having the following section at the surface: sand, 5 ft.; gravel, 5 ft.; boulder clay, 22 ft.; silt or mud, 75 ft.; red sandstone (with much water), 40 ft.; total, 147 ft.*

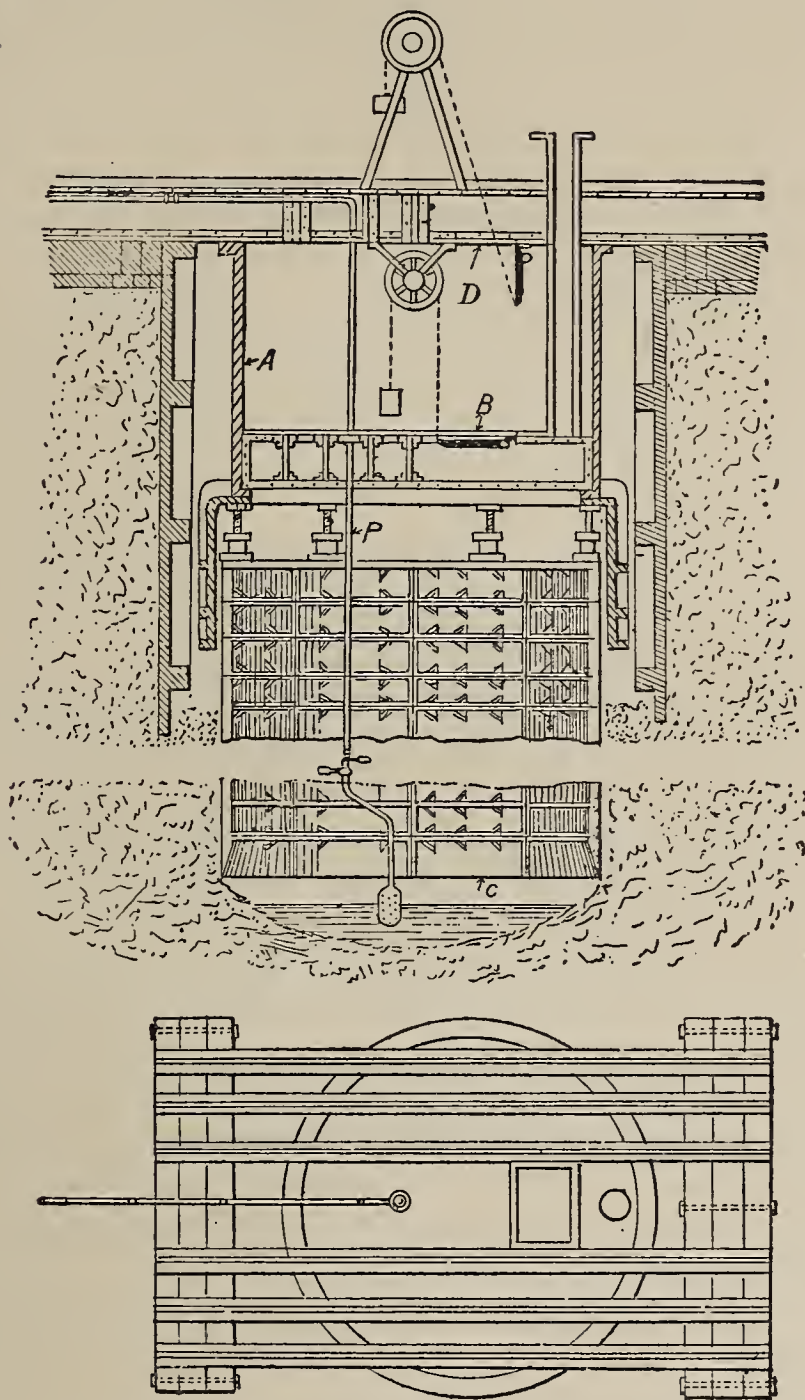
A square pit was first sunk through the boulder clay until the top of the running mud was reached. This pit measured 18 ft. 6 in. inside the wood, and was lined with 9 in. broad $\times$ 4 in. pitch pine. When the pit was secured in this way down to the top of the mud—i.e. to the bottom of the boulder clay, a distance of 32 ft.—the bottom segment of the steel cylinder with the cutting edge was built

* This colliery has now been abandoned owing to difficulties underground.

in and fixed in position. On the top of this segment were built up as many rings of the cylinder as brought it above the surface level, the work of lowering the rings being carried out by means of a steam crane. Before the actual sinking through the mud was entered upon, the cylinder was therefore between 30 and 40 ft. in length. The cylinder was built entirely of steel plates, the bottom segment being 5 ft. deep and $\frac{3}{4}$ in. thick, while the other rings were 4 ft. deep and $\frac{1}{2}$ in. thick. The outside diameter of the cylinder was 18 ft. $2\frac{1}{2}$ in., and the circle made up of twelve segments. To facilitate the sinking of the cylinder the diameter of the bottom ring was 18 ft. 4 in., $1\frac{1}{2}$ in. wider than the rest of the cylinder. The segments were joined together by means of T-pieces, 6 in. $\times$ 3 in. $\times$ $\frac{1}{2}$ in., and at the four joints opposite the centre lines and covering pieces, 6 in. $\times$ $\frac{1}{2}$ in. were also used. At the top of the bottom segment a projecting piece was built all round the inside of the cylinder as a foundation for the brickwork which was to form the shaft lining. This projecting piece was supported by brackets, also made of T-pieces 6 in. $\times$ 3 in. $\times$ $\frac{1}{2}$ in., and fixed at intervals round the circumference of the cylinder. To give further support to this projecting shelf which carries the walling, another projection, consisting of angle irons, was fixed on the bottom of the lower segment, and from this projecting piece a tapered section of brickwork was carried up to the shelf above, on which rests the regular walling. At the outer edge of this upper shelf or support, an angle iron, 3 in. $\times$ 3 in. $\times$ $\frac{1}{2}$ in., was fixed at a distance of 2 ft. from the inside of the cylinder, and this formed the circle for the shaft lining. The lining consisted of an outer ring of 18 in. of brickwork, and the remaining space of 6 in. was filled in with a mixture of lime and pure cement. When the cylinder was placed in position and the sinking through the mud about to start, the procedure was to dig out the silt or mud to a depth of 2 or 3 ft. at a time. If the cylinder would not sink by its own weight, more brickwork was added from the top, the brickwork lining being gradually added to until the surface was reached. When the cylinder would no longer sink by its own weight *plus* the weight of the walling, which was found to occur at a depth of 75 to 80 ft. where the 'skin' friction became excessive, additional weight was added by using pig iron placed on a scaffold resting on 9 in. $\times$ 6 in. pitch pine buntons built into the walling. Before the mud was finally sunk through, the total weight of pig iron resting on the cylinder amounted to 400 tons, and the combined weight of cylinder and pig iron exceeded 800 tons. The digging of mud and the loading of the cylinder were continued till the rock head was reached. The progress made in the sinking varied greatly from $3\frac{1}{2}$ ft. down to $\frac{1}{2}$ in. per twenty-four hours. The total time taken to sink the cylinder through the mud and boulder clay down to the rock was about five months, this time including the fixing of the rings, building the walling, and all the other work connected with the sinking.

Triger's Method.—This system was first applied by a French engineer, M. Triger, about the year 1841, to sink a shaft on an island in the Loire.*

The system essentially consists in forcing iron tubbing down through the ground by pressure applied from above, and in furnish-



FIGS. 76 and 77.—Triger's method of sinking.

ing the tubbing with an air chamber, which forms a double diaphragm interposed between the outside atmosphere and the interior of the pit, and maintains therein a pressure equal to that due to the head of water at the lower end of the tubbing.

Figs. 76 and 77 show the arrangement of tubbing, etc. A is the malleable-iron chamber, with two man-holes D B. When placed in

* *Trans. Min. Inst. Scot.*, vol. vi. p. 27.

position at the mouth of the pit and firmly secured, it admits of further excavation for the introduction of the cutting ring of the tubbing C. Compressed air is forced in through the pipe P into the lower compartment, and the air pressure being greater than that of the water contained in the sand or strata, it holds in check any water tending to flow in, or forces it up through a flexible tube.

The workmen enter the air chamber by a door D at the top, and as soon as they are in, the opening is immediately shut. When the pressure of air in this chamber becomes equal to the pressure in the lower compartment B, the second or lower door is opened, and the workmen proceed into the interior of the pit, and there carry on the operations of sinking, of forcing down the tubbing by means of screw-jacks, and of adding the segments as required.

All the doors and joints should be as nearly air-tight as possible, and the doors leading to the compartments are never both opened at the same time. The maximum depth that can be sunk by this method is about 100 ft., the air pressure at that depth being equal to three atmospheres, or 45 lbs. per sq. in. Even then, it is difficult to get workmen to stand it without injury to health. At Puits Marie, near Aix-la-Chapelle, however, a shaft was sunk to a depth of 111 ft. in water strata by this system.*

In this instance, it was anticipated that the capillarity existing between the particles of running sand would have the effect of lightening the pressure due to this head of water, and the facts justified the assumption. A good many shafts have been sunk by this system, as also foundations for piers of bridges, etc.

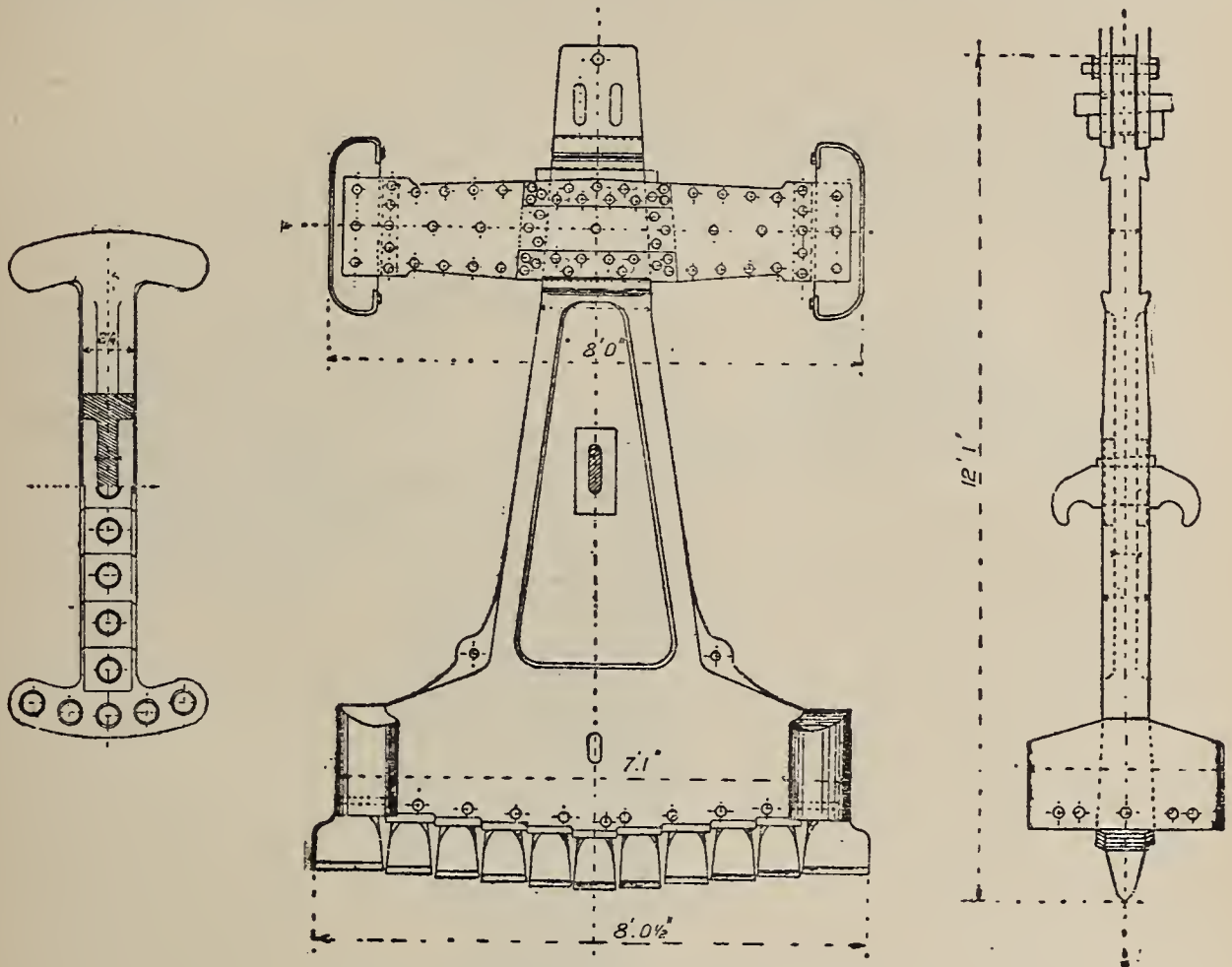
A few years ago two shafts were sunk by this process at the Ardeer Colliery on the Ayrshire coast off the Firth of Clyde. A double difficulty presented itself in this case in that a considerable depth of running strata had to be sunk through, and the shafts being situated close to the seashore the water was quite salt, hence the freezing system could not be applied. The cylinder used was of steel $\frac{1}{2}$ in. thick and 17 ft. 7 in. external diameter, closed at the top. The total depth sunk by the compressed air cylinder was 85 ft., and to accomplish this it required a total weight of 784 tons to exert the necessary downward pressure. The cost of sinking and lining these two shafts to a depth of 83 ft. and a finished internal diameter of 14 ft. was £18, 19s. 4d. per vertical foot.†

Drilling or Boring out Shafts.—In some districts in England, the coal-measures are overlaid by strata containing very large quantities of water, which would entail great expense in pumping during the sinking. In some parts of the French coal-fields, the coal-bearing measures are overlaid with chalk, which likewise contains large volumes of water, the coal strata below being comparatively dry. To sink shafts is, in either of these cases, very expensive, and heavy

* *Trans. Min. Inst. Scot.*, vol. vi. p. 28.

† *Trans. Inst. Min. Engs.*, vol. xxx. p. 205.

pumping machinery would be required while sinking through the water-bearing strata in the ordinary method, while after the coal-beds were reached and tubbing put into the shaft, no pumping apparatus would perhaps be required, and much valuable machinery would be left on hand that would be of little use, and could only be sold at considerable loss. It was to successfully meet and overcome such difficulties that the Kind-Chaudron system of sinking was introduced and adopted. Since its introduction over eighty shafts have been sunk



FIGS. 78, 79, and 80.—Small trepan.

on this system, in various parts of Europe, including six in England. The sinkings at Dover are being carried out on this system.

Briefly, this method consists of boring out the shaft, and then lowering into it a water-tight lining of cast-iron tubbing.

This system of sinking may be divided into the following stages * :—

Alternately boring a small pit in advance and then enlarging it by a larger tool to the full size of the shaft.

Preparing a seat for the 'moss-box.'

Lowering the water-tight lining or tubbing with the moss-box at the bottom.

Putting in the outside lining of concrete.

Pumping out the water.

In the preliminary operations, a small pit 4 ft. to $8\frac{1}{2}$ ft. in diameter is bored out by a tool known as the small trepan, and

* *Ore and Stone Mining*, Sir C. Le Neve Foster, sixth edition, p. 288.

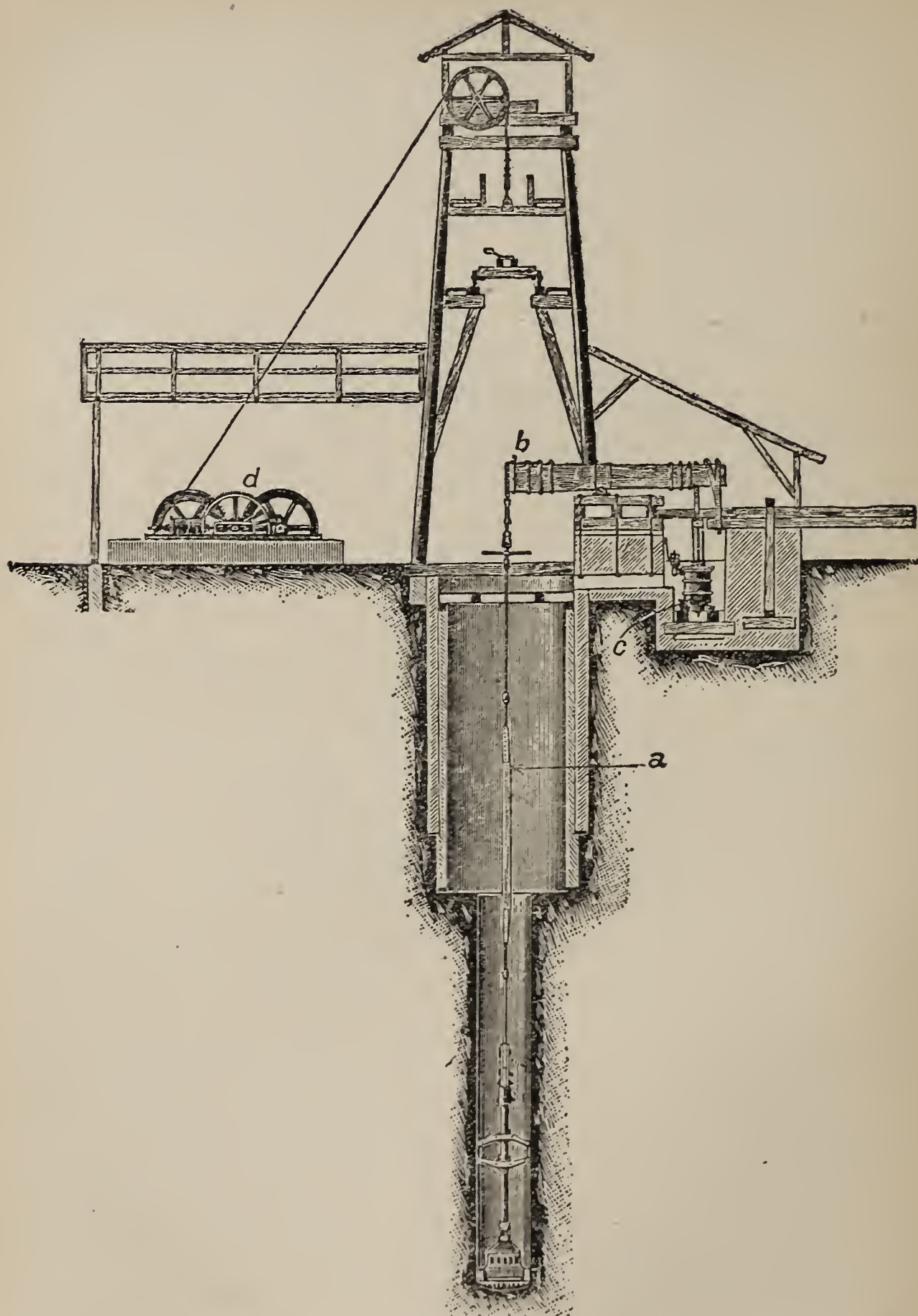


FIG. 81.—Section of shaft showing surface arrangement of Kind-Chaudron system.

a = boring rods ; *b* = boring beam ; *c* = vertical steam-engine ;
d = engine for raising rods, trepan, and débris.

weighing about 8 to 12 tons. It is supplied with 14 cutting teeth or chisels of chilled steel, securely fastened into the jaw of the trepan (figs. 78, 79, and 80 show details of small trepan). The trepan is suspended by pitch pine rods 7 in. to 8 in. square, and in long lengths of 50 or 60 ft. The rods or spears are actuated by a steam-engine with a vertical cylinder of 30 to 40 in. diameter and a stroke of about 4 ft. (see fig. 81). A large strong beam, with the fulcrum nearer the pit than to the engine, is attached at one end to the piston in the cylinder, and at the other to the rods to which the trepan is fixed. The steam-engine actuates the trepan through this beam in much the same way that bore rods are worked by a brake-staff, raising it from 1 to 2 ft. at every stroke and then allowing it to fall sharply by its own weight, the rods being turned in the usual way after each stroke by means of a cross-piece or 'tiller.' The débris in the small pit is removed by a sludger, which can be either attached to the rods or let down by a wire rope wound on a drum worked by a small horizontal engine for the purpose.

When the pit has been bored out for a depth of 20 or 30 yds., the small trepan is withdrawn and the larger tool is set to work.

This large trepan weighs 16 to 25 tons, according to the diameter of shaft, and is fitted with a strong iron bow in the centre which is a little smaller than the diameter of the small pit, into which it fits, acting like a guide. The teeth, about nine in number on each side, are fixed at each end of this bow, those near the centre being longer than those at the extremities (fig. 82), the object being to make the upper edges of the larger pit slope toward the small pit, which has already been bored, and so facilitate the free passage of débris into the receiving bucket or pan. This bucket or pan is inserted at the bottom of the small pit, and the refuse falls into it, as it is cut by the large trepan.

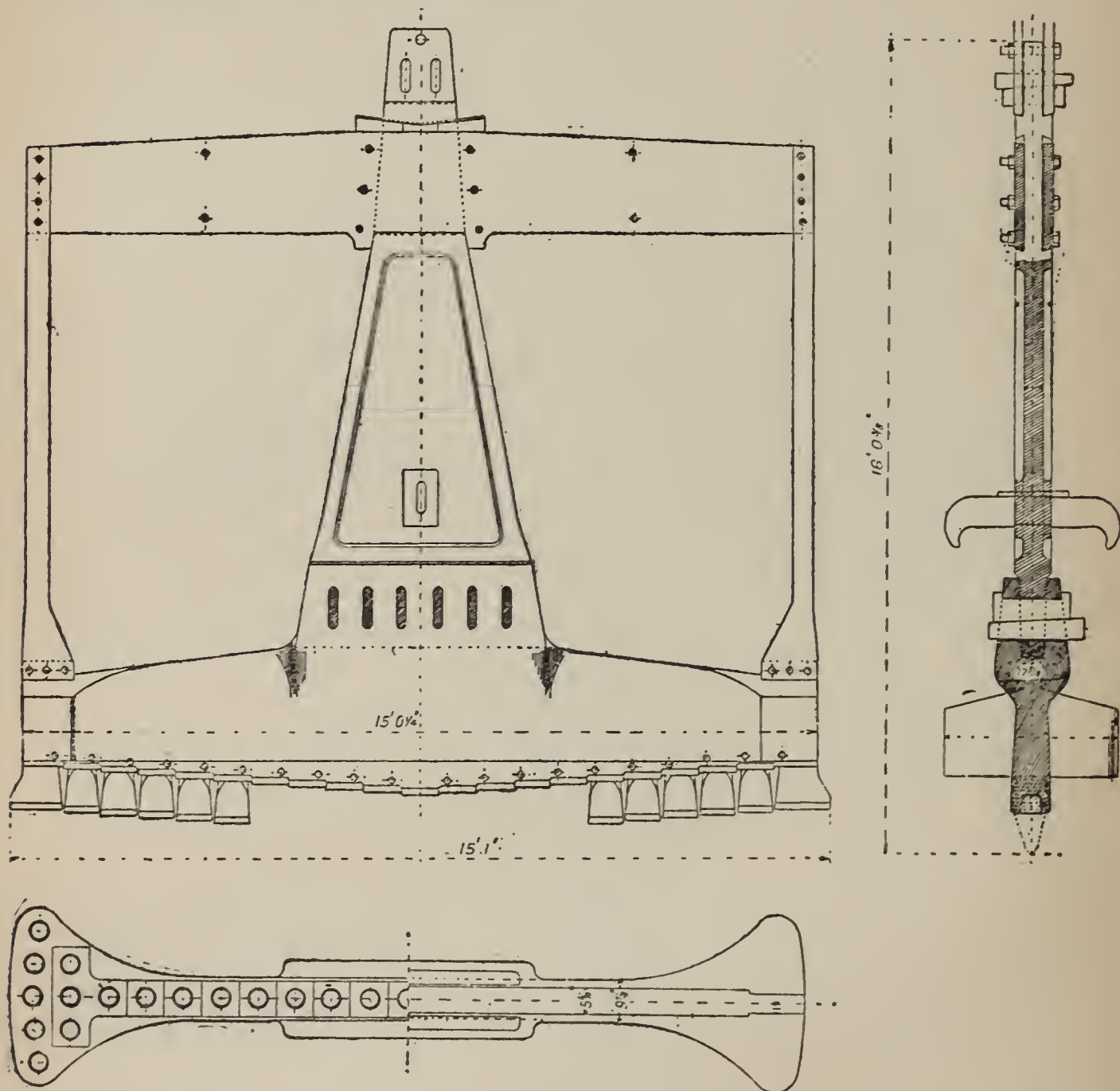
An iron bow is provided on this bucket which can be caught by a grapnel attached to the rope, and thus raised to the surface. Figs. 82, 83, and 84 show detailed drawings of the large trepan. When the sinking has reached the firm rock a smooth bed is carefully prepared for the tubbing and the moss-box to rest on, for upon the tightness of the tubbing depends the whole success of the process.

This bed is made level by a special tool somewhat resembling a large pair of 'lazy-tongs.'

When the shaft has been sunk to the required depth, the most difficult part of the work, viz. the lowering and fixing of the metal tubbing, is then proceeded with. This consists of cast-iron rings *a a* (fig. 85), the full diameter of the finished pit, each ring being about 5 ft. deep and varying in thickness according to depth. They are cast with internal flanges, and the rings are joined to one another by bolts *b b*, the joints being made water-tight by the insertion between them of thin lead sheeting. At the bottom

of the tubbing are two rings with flanges turned outwards and so arranged that they can slide over each other (fig. 86).

The space between these two flanges is filled with moss, which, when compressed by the weight of the tubbing as it is lowered into position, makes a water-tight joint.

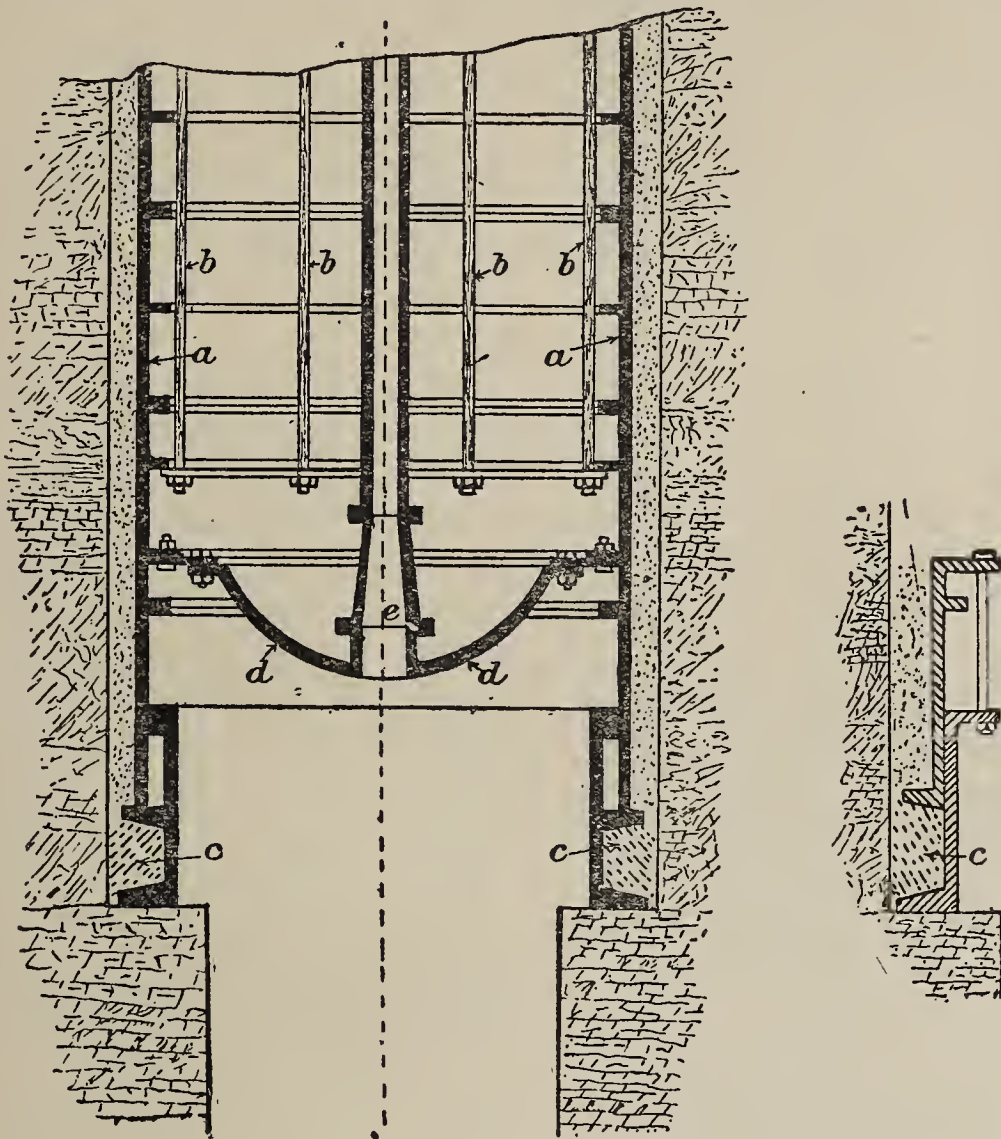


FIGS. 82, 83, and 84.—Large trepan.

Immediately above the moss-box *c* a false curved bottom *d* is bolted on, with a tube *e* in the centre, which allows of rods being worked through it and the pressure of water to be lessened as the tubbing descends. The whole column of iron tubbing, with moss-box attached, is lowered by strong iron screws and rods attached to heavy beams, placed over the shaft at the surface or to beams fixed in the shaft above the water-bearing strata. When the column is lowered, the space left between the outside of the

tubbing and the side of the shaft is carefully filled in with good cement or concrete, lowered in boxes so constructed that their contents can be discharged at any definite point. After ample time has been allowed for the cement to 'set' and harden, the water is drawn out of the shaft by means of a water-barrel or pump, and the rest of the sinking is then proceeded with in the ordinary way.

By means of the false curved bottom or diaphragm mentioned



FIGS. 85 and 86. —Kind-Chaudron method.

above, the pressure can be so adjusted that the tubbing can be practically floated into position, thus reducing the strain on the lowering rods to a minimum. Fig. 87 shows a general view of the tubbing, moss-box, etc., in position in the shaft. Of course, at great depths where, owing to the pressure, the metal tubbing requires to be very heavy, there must be, of necessity, a great strain on the lowering rods. It is evident, therefore, that the method of installing the tubbing will depend largely on the depth, and will have to be adjusted to meet special cases.

Within the last few years several modifications of this system

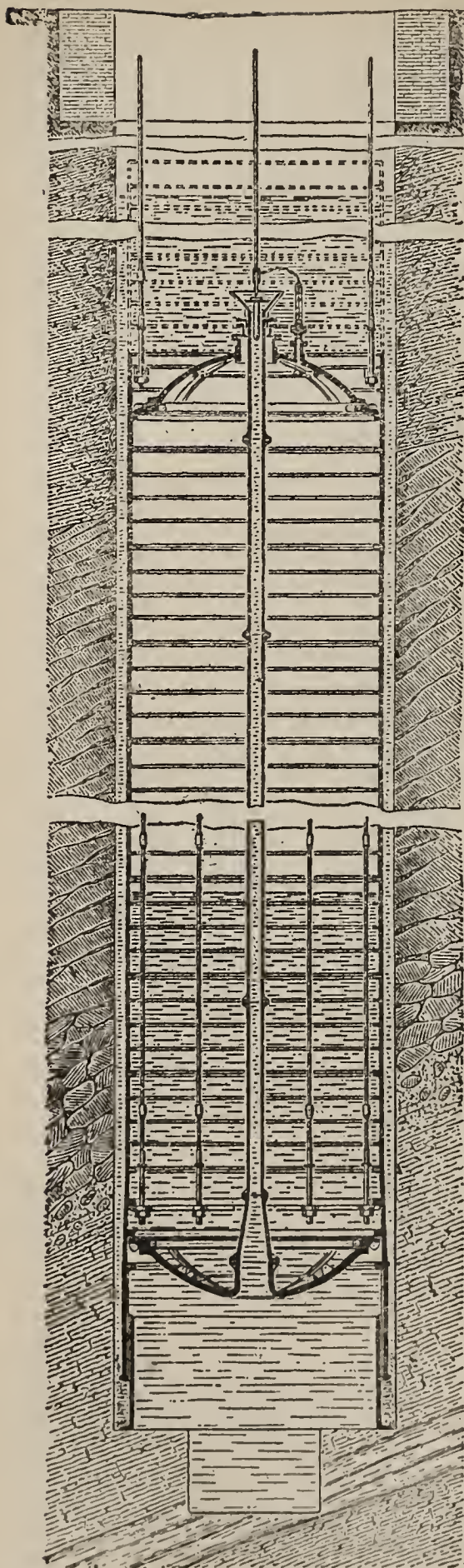


FIG. 87. — General arrangement showing tubbing with moss-box and cover with valve for equalising pressure.

of sinking have been introduced and carried out with success. The moss-box is no longer considered necessary in the fixing of the tubbing, reliance for a water-tight joint being placed in making a carefully prepared bed and good cementing.

At a shaft sunk by the Liévin Company in the north of France, the shaft was bored out in two operations; a first pit 6 ft. 6 in. wide being sunk to a depth of 10 or 12 yds. beyond the watery strata, which was subsequently enlarged by a second boring to the full size.*

On reaching the required depth the teeth of the trepan are set so as to cut a horizontal and level bed for the tubbing to rest on, and the use of moss-box, equilibrium tube, and false bottom can be entirely discarded. Work can, by this method, be carried on with great rapidity; in one instance the small shaft was bored out to a depth of 366 ft. in seventy-five days, the larger one being bored out to 327 ft. in four months twenty-one days, and fixing the tubbing occupied two months longer. The ordinary rate of boring by this system is 9 to 12 in. per day, according to the depth and diameter of the shaft.

Lippman's Method. — This method of sinking is practically the same as the Kind-Chaudron, but instead of the shaft being bored out in two or three operations, it is completed in one, *i.e.* the shaft is bored out from the commencement with a large trepan specially made for the purpose.

* *Ore and Stone Mining*, sixth edition, p. 293.

Advantages of these Systems.—For this and the Kind-Chaudron method of sinking the advantages are * :—

The use of pumps is avoided, unless when the shaft requires to be cleared after the tubbing has been lowered.

The risk of accidents to workmen, which are common in the ordinary mode of sinking, are reduced.

The inconvenience of draining the surrounding springs, which in a populous district depending on these for a water supply would be great, is avoided.

Shafts may be sunk to coal seams through ground which it would be impossible to deal with by the ordinary methods of sinking.

Against these advantages there must, however, be set the fact that these methods of sinking are very costly.

The cost of such sinkings may vary from £50 to £150 per yard depth, according to the strata and the difficulties encountered. This price does not, of course, include the cost of the tubbing and other accessories.

Pattberg System.—This system, like the Kind-Chaudron, is applied for drilling out shafts in water-bearing strata containing large quantities of water. It somewhat resembles the Kind-Chaudron method, but has several distinctive features which are quite new. Two shafts have recently been successfully sunk by this method at the Rheinpreussen Colliery, near Homberg, Germany.

The principal appliances used in this method of sinking are the percussive boring tool mounted on a strong wooden or wrought-iron frame, and supported by a tubular boring rod (see figs. 88, 89), and two mammoth pumps, the whole being slung from a scaffolding over the shaft, and an oscillating drum, driven by a steam engine, for giving reciprocating motion to the cutter. The borer B (fig. 89) hangs on a wrought-iron tubular boring rod, having an inside diameter of 150 mm. (6 in.) and 15 mm. ($\frac{3}{5}$ in.) thickness of metal. The chisel-carrying part *r* is also of wrought iron. It slopes upwards from the centre to both sides, so as to cut a surface inclining towards the centre of the shafts, and has on either side a tube-like piece *a*, from which the small channels *b b* branch off at right angles and lead into the corresponding channels in the steel chisel teeth *z z*. The tubular boring rod, from which the cutting tool is suspended, is in communication with the hollow pieces (*a a*), and supplies the water which flows out at the edge of the chisel teeth. The vertical and horizontal guiding arms *u* and *v*, as well as the other supporting pieces, are made of wood. The apparatus which was first employed had a cutting edge of 6·4 metres (20·99 ft.) broad and 8·3 metres (26·89 ft.) high in the centre, the total weight of the boring piece being 19,800 lbs.

Instead of the screws hitherto used for holding the individual parts together, wedges were used for this new borer, as it was

* *Trans. Min. Inst. Scot.*, vol. vi. p. 28.

thought that owing to the large number of times it had to be raised and lowered, the screws would very soon become loose. Further, instead of constructing this borer with teeth, a straight cutting edge was put on.

At either side of the tubular boring rods is one mammoth pump. These mammoth pumps consist of two pipes (R R, fig. 89) of 3 mm. ($\cdot 117$ in.) and 140 mm. (6.5 in.) inside diameter. The pipes reach down almost to the point of the borer, and enclose a second pipe of the same thickness and 100 mm. (3.9 in.) diameter. In the annular space between the two pipes compressed air is brought from the surface, which is allowed to escape a little above the lower end into the inner tube. This causes a pressure on the

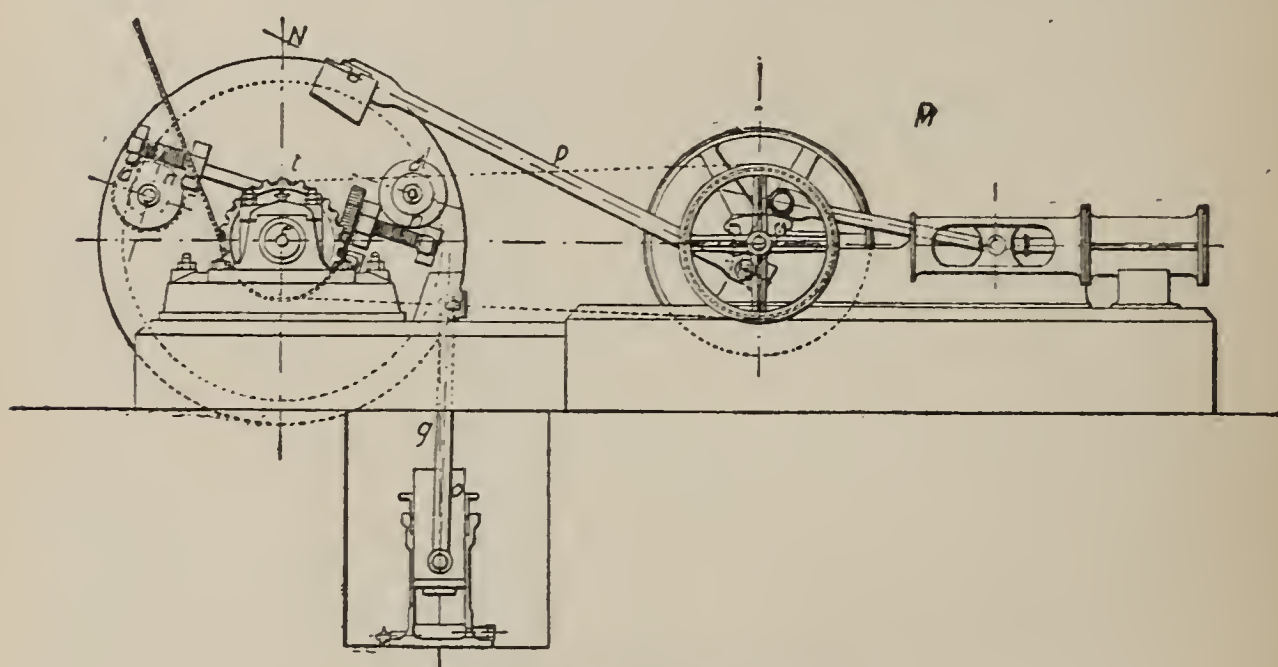


FIG. 88.—Pattberg system.

surface of the detrital sludge, causing it to be sucked off the centre of the shaft and brought to the surface.

The tubular boring rods are led into the shaft scaffolding through a hollow guide, and have on the top a revolving piece to which a rope is fastened. This rope is wound round an oscillating drum (*t*, fig. 88), which is operated by a steam engine through a crank shaft *p* connected to a large drum or disc *N*. In order to release the boring arrangement the weight hanging on the rope is partly adjusted or counterbalanced by steam pressure, by means of a plunger, connected by a rod *g* to the drum *N*.

About 40 horse-power are required for starting the boring arrangement. While boring is proceeding, the alternate slackening and tightening of the rope, to give percussive action to the cutting head, is effected in the following manner:—The drum (fig. 88) revolves on the axle *e* of the boring apparatus, drum *t* and another disc behind are fixed to the axles by wedges. The disc behind holds a circular rack (*i.e.* a rod with teeth on it) into which the

spur wheels $d d$ catch. From a pulley also revolving on the axle e these spur-wheels are driven by smaller spur-wheels $n n$. The belt connecting the pulley with another one fixed on the axle of the disc e is, as a rule, slack. When it is made tight the teeth wheels $d d$ are caused to revolve, and set the boring apparatus in motion. For the purpose of changing the position of the cutting tool at the bottom of the shaft, a 'Krückel' (tiller) is fitted on to the boring rods at the surface, and is operated in exactly the same way as when small bore-holes are being put down by the ordinary percussive method. For the lowering and taking out of the borer and pumps a steam crab is used, which is set up opposite the boring appliance, on the other side of the shaft scaffolding. The total weight of the tubular rods and the necessary pipes amounts to 297 lbs. per current metre (3.28 ft.). The cutting head is worked at the rate of fifty to sixty strokes per minute while boring goes on, the height to which the cutter is lifted being 18 to 20 cms. (7 to 8 in.). At every round the borer is set in afresh twenty to sixty times per minute according to the resistance offered by the ground. About fifty men are required in connection with this work. They are spread over three shifts of eight hours each. In every shift four men are employed on the boring stand.

In the deepening of Shaft IV. at the Rheinpreussen Colliery a sinking wall of 8.90 metres (29.19 ft.) clear diameter was built. The boring of the loose ground was then effected by means of a breaking appliance driven by hand labour. When the wall had been sunk through a layer of gravel of a depth of about 17 metres (56 ft.), the sinking working was temporarily suspended and the bottom of the shaft was filled up with concrete for a depth of 10 ft. After giving this concrete three months' time to become hard, a new sinking cylinder of 6.5 metres (21.32 ft.) diameter was built in and the sinking resumed, the percussive drill being used to penetrate through the concrete, which was cut at the rate of 4 ft.

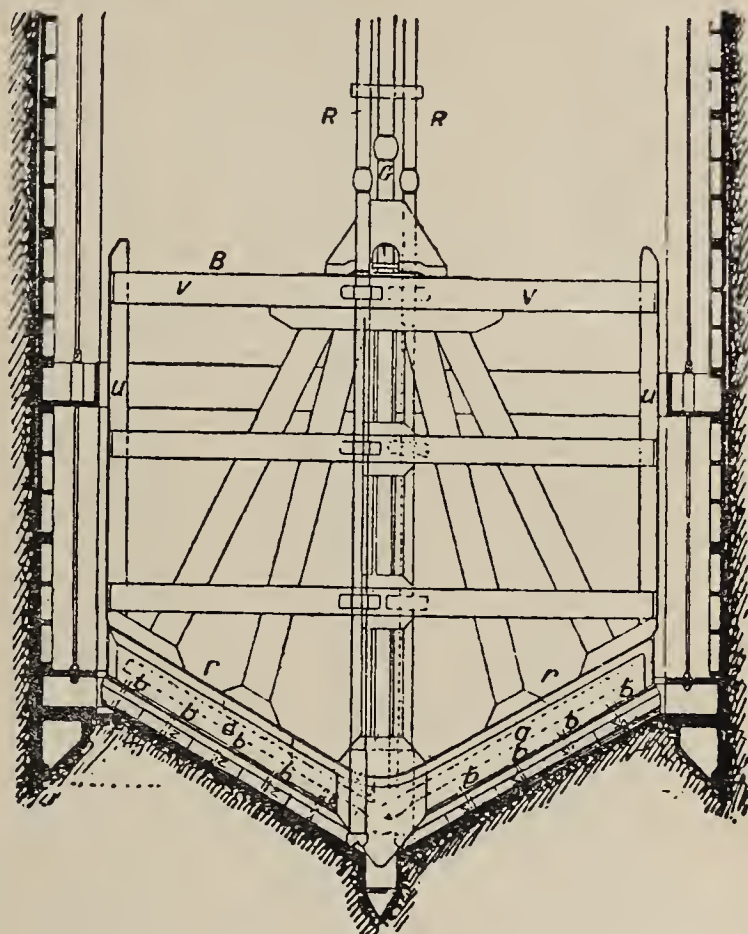
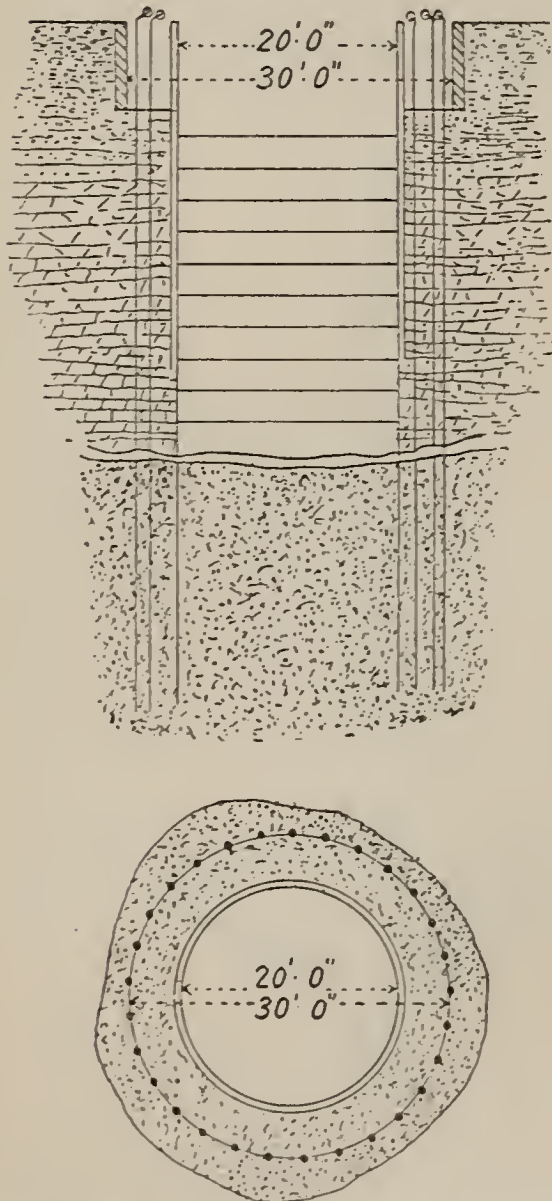


FIG. 89.—Pattberg cutter.

per day. This method of sinking has up till the present only been applied to loose water-bearing strata, and it has yet to be demonstrated that it would be equally successful in hard ground; but in view of the satisfactory results obtained in boring through the concrete layer, this does not appear to be out of the question.

Sinking by Freezing the Strata.—In this system of sinking, watery strata are artificially solidified by freezing. Herr Poetsch was



FIGS. 90 and 91.—Plan and section of shaft showing position of freezing tubes.

the first to invent and use this system in 1888 in the sinking of two shafts near Schneidlingen in Germany. Since then a large number of shafts have been sunk by Poetsch's method, or a modification of it, in France, Germany, and Belgium, and also several in England. When the system was first employed, a series of three or four concentric rings of tubes were placed all round the proposed circumference of the shaft, and also within the area to be sunk through. That practice is now abandoned, and it is now the invariable custom to have only one or at most two rings of tubes, and placed entirely outside the area of the shaft, so that in the sinking the excavation can be carried on more freely, and without any hindrance from the tubes (see figs. 90, 91). The freezing process may be divided into four stages: (a) boring holes for the reception of the freezing tubes; (b) freezing the strata or making and maintaining an ice-wall; (c) sinking through the ice-wall and inserting the necessary tubing; (d) thawing the ice-wall and extracting the freezing tubes. A series of bore-holes are

first put down surrounding the area where the shaft is to be sunk, and these are then lined with tubing through which a freezing solution of chloride of calcium is made to circulate by means of pumps. The freezing mixture, which is at a very low temperature, absorbs heat from the surrounding watery strata, which freeze into a solid mass, when the excavation of the shaft can be carried on in the ordinary way. Fig. 92 gives a sketch of the freezing pipes which are inserted into the bore-holes. They consist of an outer and inner tube, the

freezing liquid being forced down the smaller inner tube circulating round the outer one, and escaping at the top, where it is led back to the refrigerating machine and used over again.

The large tubes, which are 6 in. to 8 in. diameter, are plugged up at the bottom with lead, cement, or any other substance that will render them water-tight; great care being taken to make this stopping secure, as the success of the operation practically depends on this precaution.

The number of tubes required will depend on the strata and the difficulty or otherwise of solidification. At a pit sunk at Lens in the north of France by this method, the area frozen was about 40 ft. diameter and $137\frac{3}{4}$ ft. deep, the number of tubes used was 28, and the freezing of the strata took about 120 days.

In this system there is considerable risk of failure; for should there be any leakage or improper plugging of the tubes, the freezing mixture, which is itself uncongealable, may escape, and by permeating the strata render attempts to freeze the water futile. The freezing mixture or brine is usually a 20 per cent. solution of calcium chloride in water. It is cooled by means of ammonia, circulating in coils, at a pressure of 9 atmospheres (135 lbs. per sq. in.), in a liquid state. The temperature of the coils is 20 to 22° C. below zero, and the brine leaves the cistern at a temperature of about -12° C. and returns to it at -9° C.

Gobert's Method.—In the Poetsch freezing system, when any great depth is reached, the pressure of the liquid within the tubes becomes very high, and frequently brings about leakage of the liquid into the surrounding strata, which renders it impossible to freeze them effectually.

In order to obviate this difficulty Gobert uses a cold transmitter, the pressure of which is lower than that of the water outside the tubes, while anhydrous ammonia vapour is used instead of the freezing liquid in Poetsch's system. With ammonia vapour very low pressures can be maintained, even at great depths, and if the tubes are not water-tight, instead of ammonia leaking out, the water from the surrounding strata would force its way in, and a coating of ice would be formed on the inside of the tubes, which would check the further inflow of water.

In order to vaporise the liquid ammonia in the tubes, these have to be connected with a suction and force pump. This pump sucks in the gas and compresses it into a liquid, with the help of a condenser, and then forces it into the freezing tubes. In order to avoid

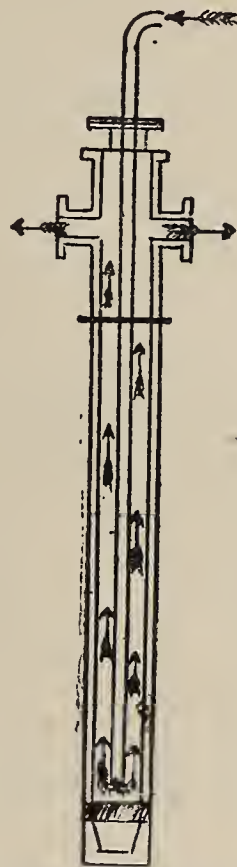


FIG. 92.—Freezing pipe.

the fall of the liquid to the bottom of the freezing tube, and to vaporise as much of it as possible in a given unit of time, the injector is made of a form spiral in one plane (see figs. 94, 95). The liquid, the entrance of which into the injector is carefully regulated, falls slowly in a thin stream within this spiral tube, and meets on its way a series of small orifices placed at various intervals in the tube. By these orifices the liquid escapes into the freezing tube, and vaporises. In the Poetsch system the watery strata must be all frozen from the bottom upwards before the sinking can be proceeded with, but by Gobert's method the strata are frozen from the top downwards, thus allowing sinking operations to be started much sooner. Simultaneously with the sinking operations, fresh strata can be successively frozen, and so allow of continuous sinking. Freezing of the strata can be carried to great depths by this system; Gobert states that

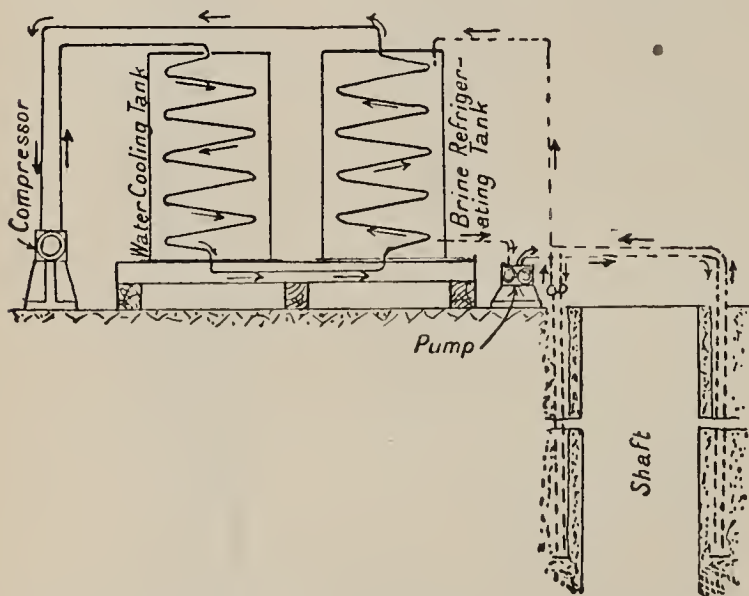


FIG. 93.—Diagrammatic section of freezing process.

strata at a depth of 3000 ft. from the surface can be dealt with. For a recent sinking the cost of this system was £40 per foot.

Koch's System.—

This system resembles that of Gobert's, but gaseous carbonic acid, ammonia, or a mixture of sulphur dioxide, is used as the refrigerating agent. Anhydrous ammonia, which has a density of 0.59, taking air as 1, and boils at

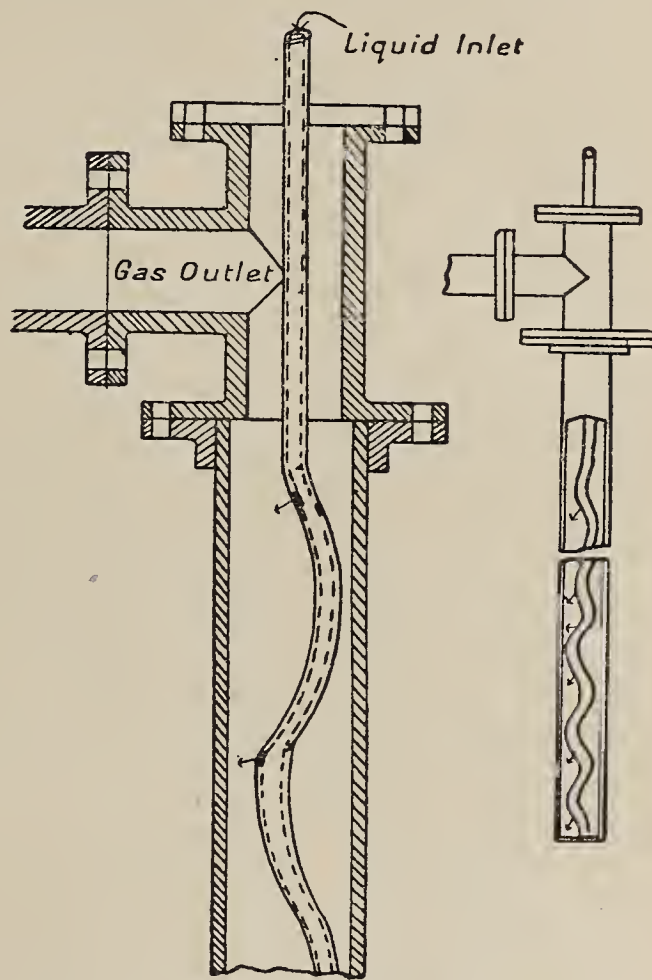
−40° C. at atmospheric pressure, is generally used. At the Washington Colliery, Durham, where the first two shafts were sunk by this system in Britain, the evolving brine used was a solution of 26 per cent. of magnesium chloride dissolved in hot water, which freezes at a temperature of −34° C. The refrigerating agent is first subjected to a pressure of 150 lbs. per sq. in. by two compressors, and then delivered into a small receiver, from which it passes to the condensers, through a pipe 3 in. diameter, and thence into four tubes, each 1 in. diameter. These condensers are vertical iron cylinders, 10 ft. high and 5½ ft. in diameter, and contain, in tiers of four rings, 1600 ft. of tubing, 1 in. diameter, through which the ammonia circulates. About 4000 gallons of water per hour circulate through the condensers, the water being kept in constant motion by means of paddles. This cools the ammonia, reducing it to a liquid. The condensers are connected to the refrigerators by piping 1 in. diameter, the refrigerators, like the condensers, being vertical iron

cylinders, 10 ft. high and 7 ft. in diameter. These refrigerators, of which there are three, are jacketed first with 3 in. of peat-moss and then encased with wood. They are filled with the brine, and contain about 2000 ft. of tubing 1 in. diameter, through which the ammonia circulates after passing through reducing valves, which has the effect of reducing the pressure from 150 lbs. to about 15 lbs. per sq. in. At this point the ammonia is immediately changed from the liquid to the gaseous state, and as this can only be done by absorption of heat corresponding to the latent heat of vaporisation, this heat is taken from the surrounding bath of brine, which is thereby greatly reduced in temperature. Fig. 93 shows diagrammatically the arrangement of the refrigerating plant.

At the Washington Colliery, before commencing the freezing process, the top of the shaft was enclosed, and the exposed pipes covered with straw. A hole was bored, and a pipe 18 ft. long inserted in the middle of the shaft, and the height and temperature of the water in the hole was noted as the gradual increase of ice-wall slowly caused the water to rise.

Five or six sinkings have been successfully carried out by the freezing system in England in strata where very large quantities of water had to be dealt with, and where it would have been practically impossible to have sunk the

shafts by the ordinary open method with pumps. One of these sinkings was at Dawdon Colliery,* near Seaham Harbour, in the County of Durham, where the shafts had to be sunk through the magnesium limestone and sand containing large quantities of water, and a brief description of these sinkings may be of interest. The coal-measures are here overlain by the following strata: soil, 1 ft.; boulder clay, $5\frac{1}{2}$ ft.; old beach-gravel, $4\frac{1}{2}$ ft.; magnesium limestone, 356 ft. 10 $\frac{1}{2}$ in.; marl slates, 3 ft. 1 $\frac{1}{2}$ in.; and yellow sand, 92 ft. 4 in. Each of the shafts, the Theresa and the Castlereagh, were 20 ft. diameter when finished. The Theresa shaft was sunk to a depth of 350 ft. by means of pumps, and lined with 225 ft.



FIGS. 94 and 95.—Gobert's freezing tube.

* *Trans. Inst. Min. Engs.*, vol. xxxii. p. 551.

of cast-iron tubbing and 96 ft. of brick walling. Various sets of pumps jointly capable of dealing with 7000 gallons of water per minute were employed, the largest quantity dealt with being 6075 gallons per minute, which was pumped to the level of a drift 90 ft. below the level of the surface, through which it ran to the sea-beach. At a depth of $349\frac{1}{4}$ ft. a drift was driven from this shaft to a position underneath the Castlereagh shaft, and a bore-hole put down to connect the bottom of the Castlereagh pit with the drift, so as to take the water from this shaft. The length of the bore-hole was 146 ft., and it was lined with steel tubes $9\frac{1}{2}$ in. diameter. Two Evan's pumps were used to pump any excess water which did not drain through the bore-hole. Sinking in the Theresa shaft was then stopped to allow both shafts to be sunk to the same depth, so that both shafts might be sunk simultaneously through the sand. In the Castlereagh shaft at a depth of 200 ft. the water given off was 5750 gallons per minute, and this, with 1300 gallons per minute coming in at the bottom of the Theresa shaft, made a total of 7050 gallons per minute to be dealt with. The pumping of this quantity, with the probability that the feeders would be greatly augmented in sinking through the sand bed, made it desirable that sinking by pumps should cease and the strata be frozen before further progress was made. Prior to the bore-holes for receiving the freezing tubes being drilled, a foreshaft 36 ft. diameter was formed round each shaft, sunk to a depth of 8 ft. and brick lined to the surface. The drilling of the holes was then started, and placed equidistant round a circle of 36 ft. The object of the foreshaft was: (1) to fix securely a tube, perfectly vertical, to act as a guide pipe, so as to keep the bore-holes as nearly perpendicular as possible; (2) to have a convenient chamber in which all connections between the collectors and freezing pipes can be made, and from which each individual freezing tube can be controlled; (3) to take observations of the temperature and watch the circulation of brine in the freezing tubes. Boring was commenced at the Castlereagh shaft on 20th May 1903, and completed on 7th April 1904. Twenty-eight bore-holes were put down on the circle of 30 ft. to a depth of 484 ft., and the average time taken to drill each hole was twenty-four days. At the Theresa shaft boring was commenced on 23rd June 1903, and finished on 22nd April 1904, twenty-eight holes here also being put down. The holes were started at $9\frac{7}{8}$ in. diameter and finished $6\frac{1}{2}$ in., lining tubes $6\frac{1}{4}$ in. diameter being inserted the whole length of each hole. The freezing tubes were then inserted, and consisted of: (a) an outer tube 5 in. diameter, in 16 ft. lengths, put down the full length of the hole and the bottom end closed; (b) an inner tube $2\frac{1}{2}$ in. diameter, reaching to within 33 ft. of the bottom of the tubbing in each shaft; (c) a central tube $1\frac{1}{4}$ in. diameter, open at the bottom, and placed within the two former down to within 3 ft. of the total depth. After the whole of the freezing tubes had been placed the lining tubes

were withdrawn, and the former were then connected to the inner and outer collectors for the circulation of the brine. The cooling liquid was ammonia, and the freezing brine consisted of a solution of 26 per cent. of chloride of magnesium at a temperature of -17°C . (1.4°Fahr.). The time taken to form the ice-wall in the Castlereagh shaft was 185 days, and the ice-wall was maintained for 353 days, a total of 538 days. In the Theresa shaft it took 392 days to form the ice-wall, and 186 days to sink through it, a total of 578 days. The ice-wall formed all round the circumference of the shafts varied from 3 to $4\frac{1}{2}$ ft. A thick bed of concrete was placed in the bottom of the shafts to prevent water from coming up through and interfering with the freezing. After the sinking was completed, and the tubing inserted to the required depth, one of the refrigerators was disconnected from the ammonia circuit in order that it might be used to thaw the frozen ground and allow the withdrawal of the freezing tubes. This was effected by passing steam through the spiral tubes and heating the circulating brine in the freezing tubes. The withdrawal of the tubes was completed on 25th May 1906, and the sinking of the shafts to a depth of about 600 yds. completed by the ordinary method. It seems to the author as if it would have been more economical and speedier to have frozen the strata at first before any sinking at all was done, as this would have saved the large outlay for pumping plant and other expenses.

Cost of Freezing System.—No authentic figures have ever been published as to what the actual cost of sinking shafts by this system amounts to, but it will no doubt vary greatly according to the depth to be frozen and the quantity of water contained in the strata. One writer gives the approximate cost for two shafts 20 ft. diameter as follows * :—

To a depth of	300 ft.,	.	.	.	.	.	£17,500
„	„	600	„	:	:	:	28,000
„	„	1200	„	:	:	:	50,000

The figures include the providing of all boring and refrigerating plant, labour and stores in connection with the boring and running of the refrigerating plant, and the maintenance of the ice-walls while the shafts are being sunk and secured. They do not, however, include steam and water.

Accessories to Shaft Sinking.—The operation of raising the excavated material during sinking is usually done by kibbles or buckets, which may be made either of iron or wood. The best form of sinking kibble is that in which the arms are fixed on trunnions with a catch at the top. Fig. 96 illustrates this kind of kibble. The great advantage of using one of this sort is that it can be completely and easily emptied without requiring to be detached from the winding rope, or even lowered on to a scaffold, for if it be swung clear of the

* *Trans. Inst. Min. Engs.*, vol. xli. p. 353.

shaft it can readily be emptied at any desired point by knocking up the catch *a*, which releases the arms and allows the body of the bucket to revolve on the trunnions *b b*. Fig. 97 shows the dimensioned construction of a kettle suitable for a large sinking. Sometimes kibbles constructed of wood and bound with iron are used

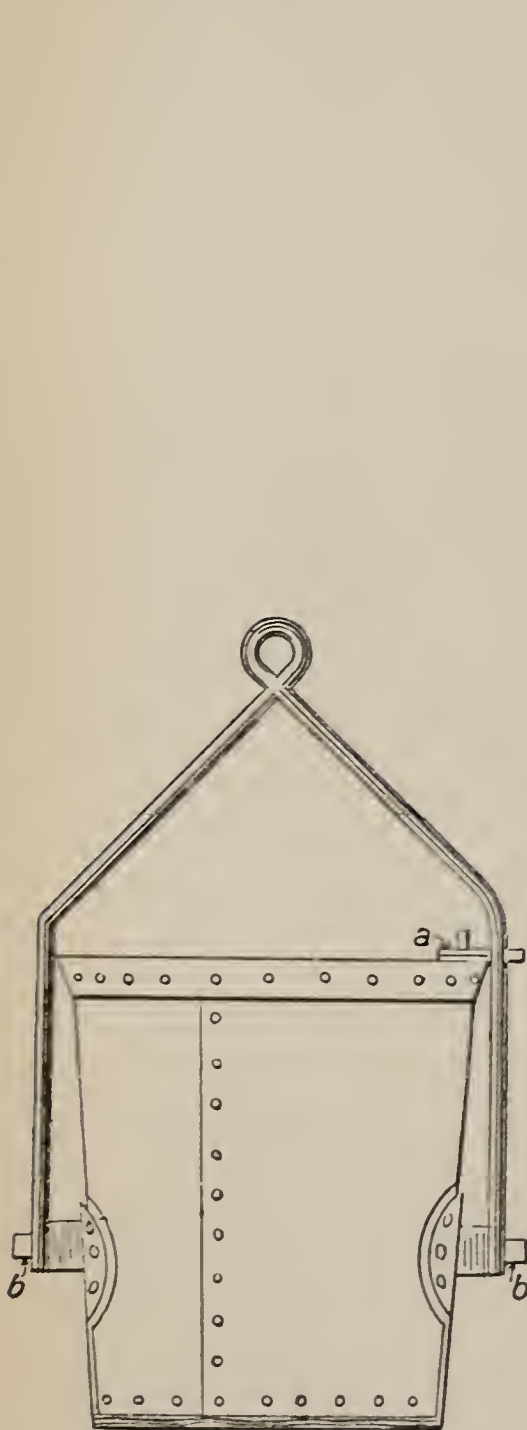


FIG. 96.—Iron kibble.

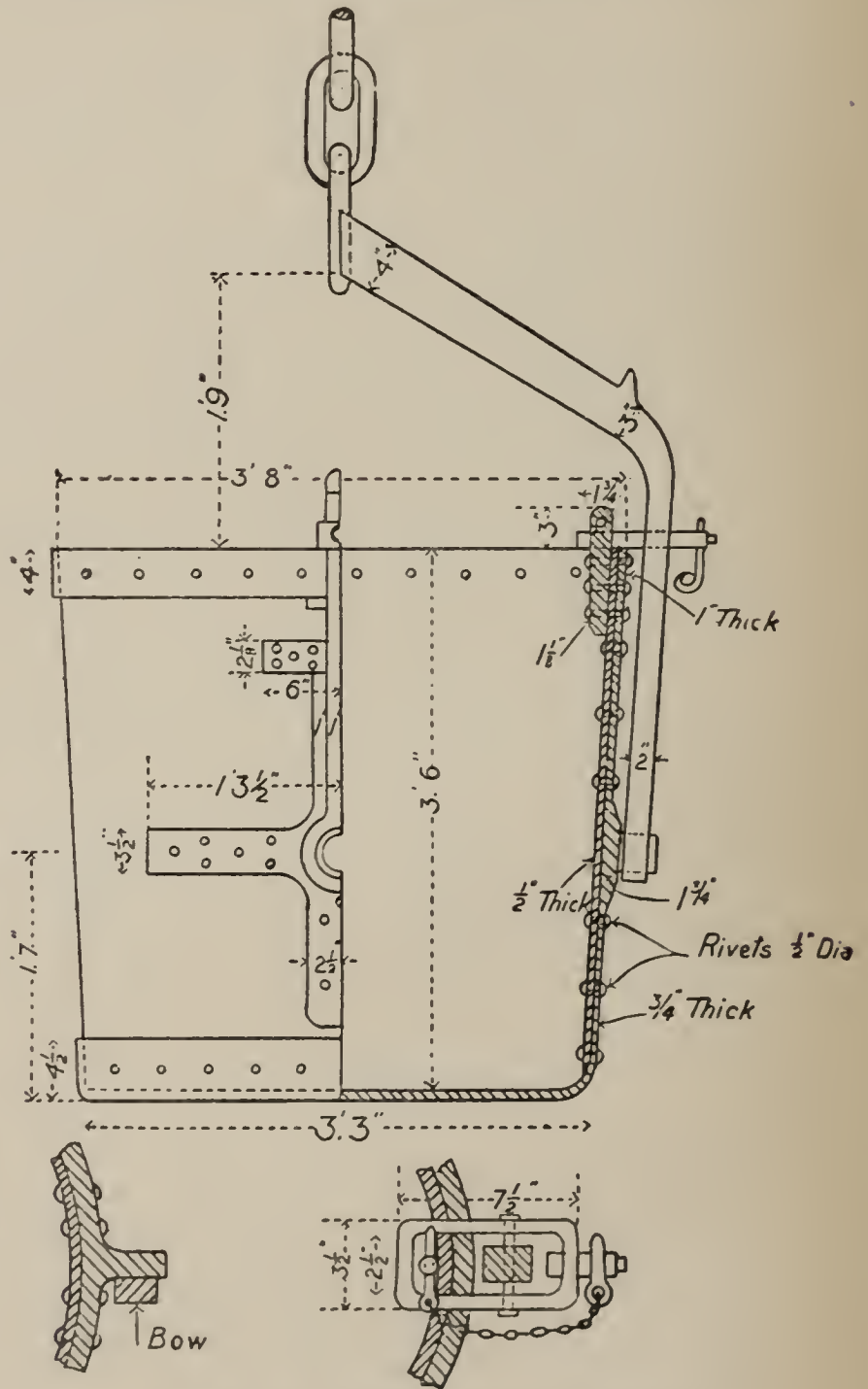


FIG. 97.—Sinking kettle or hoppit.

instead of iron ones, but they are not so handy nor yet so durable as those made of iron. Fig. 98 shows a wood kibble which is well adapted for raising water and ordinary material.

Before the kibble is raised from the bottom of the pit, the sinker in charge ought to examine it to see that the fastenings are secure, and that no stones are likely to fall off, or that none are sticking to the outside of the kibble, liable to be knocked off during the ascent

of the kibble, and possibly injure those who are working in the shaft. The same precautions should be adopted at the surface when the kibble is being lowered into the pit.

Safety Riders.—In the majority of shafts in process of sinking, the kibble is raised without being guided in any way; the common method being for the engineman to raise it a few feet from the pit-bottom when it is filled, and it is steadied for a few moments by one of the sinkers, and then drawn right away to the surface. This method acts very well if there is plenty of space in the shaft, and the depth not great; but when the depth becomes considerable, and cross-buntons require to be fixed, particularly in rectangular shafts, there is danger of the kibble catching these, and doing much damage to the sides of the shaft, and causing injury to the men at the pit-bottom. To obviate risks, guides are sometimes carried down as the sinking proceeds, and a rider employed to run between the conductors and guide the kibble, and also to keep it from swinging.

Fig. 99 shows the construction of a rider, made wholly of iron, to suit wire rope guides. Such a rider runs upon four bushes connected to the arms, the winding rope passing through an opening in the centre sufficiently large for the rope to pass through freely, but too small to permit the capping to do so. At a short distance from the pit-bottom conductors are fastened, or a projection is fixed to them, so that they may grip the rider when it reaches that point. The rope continues to descend through the central opening until the kibble reaches the pit-bottom, while the rider is securely held above. On the upward journey the rope runs through until the capping strikes the rider, which is then carried up to the surface, guiding the kibble during its ascent. Figs. 100, 101 show the details of the bush and gland which run on the rope at *a a* and *b b*.

When wood conductors are used a differently constructed rider is required. A form of conductor which is simple and efficient under these circumstances consists of four pieces of wood *e e* (figs. 102, 103), connected by two upright pieces *f f* firmly bolted together. The space between these is filled by pieces of wood *g g*, and only an

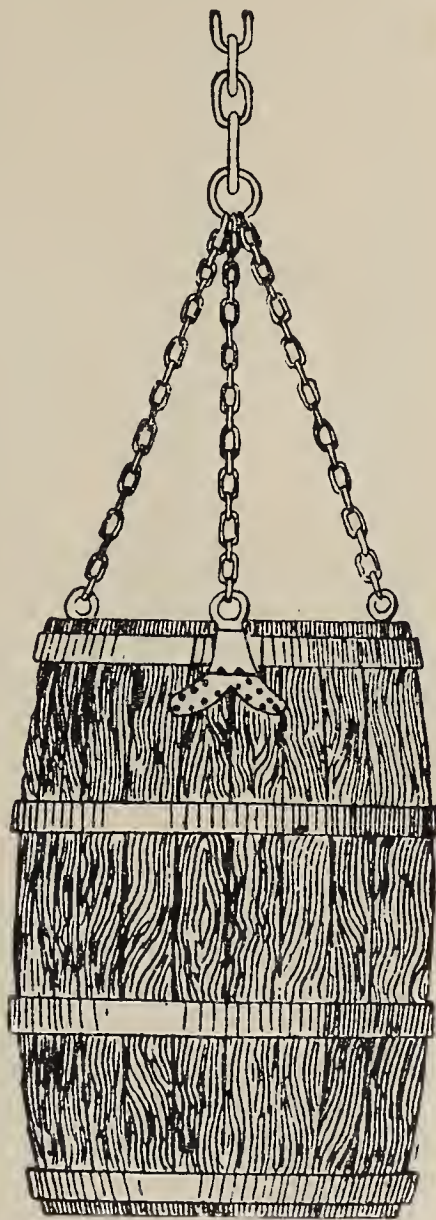
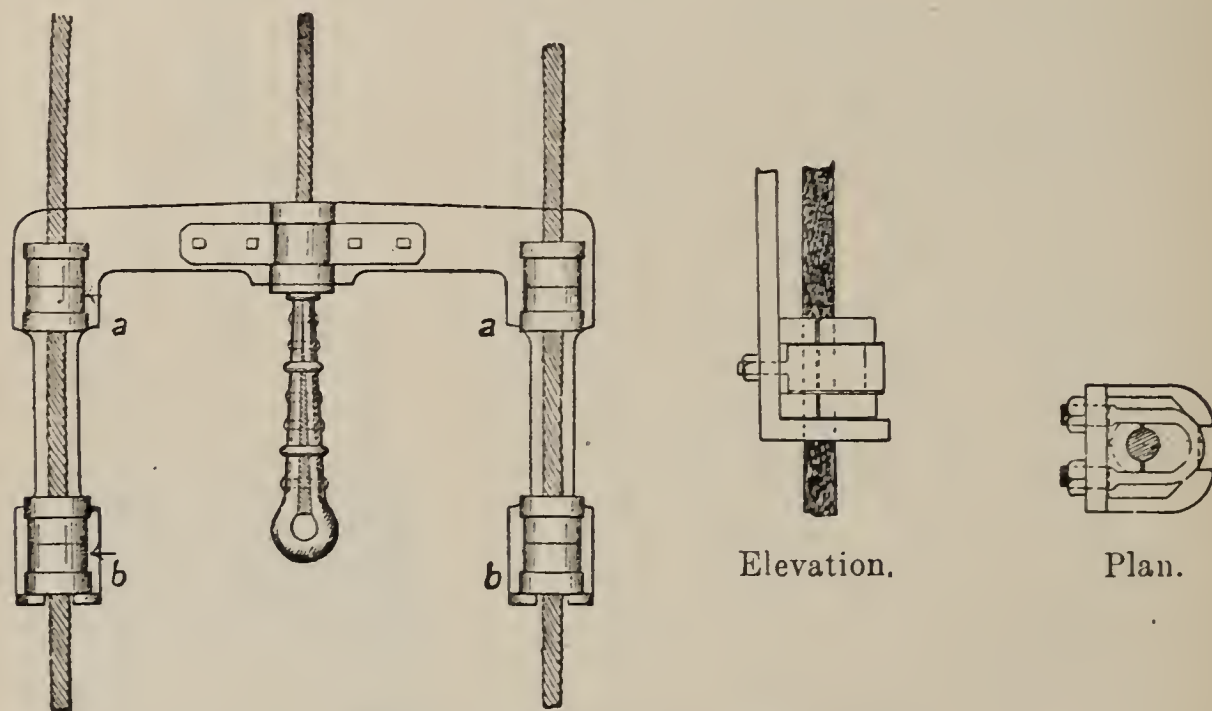


FIG. 98. — Wooden kibble.

opening about 4 in. square is left in the centre for the winding rope to pass through. Near the bottom of the guides two cleats are fixed, *h h*, for the rider to rest on, while the kibble proceeds to the pit-bottom. On the capping of the rope one or two pairs of glands *i* are fixed, for the purpose of catching the rider and carrying it to the surface during the ascent of the kibble. The advantages of using a rider during sinking are, that the winding of the kibble can be carried on at a much greater speed than if no rider be used; while, as before stated, it is prevented from swinging about and so endangering the men working below.



FIGS. 99, 100, and 101.—Iron rider.

The guides, if composed of wire ropes, should be frequently examined and kept well lubricated, particularly during frosty weather. Great care ought to be taken to keep ice from forming on the guides, as such obstructions prevent the rider from running freely, and it may then stick in the shaft and perhaps fall away and do much injury. Fatal accidents have occurred through the rider sticking and then dropping away suddenly.

Ventilating Shafts during Sinking.—While sinking is going on a sufficient supply of air must be provided at the bottom of the pit, to clear away the smoke due to blasting, and enable the men to work. This may be accomplished—

- (1) By dividing the shaft by means of a close brattice and connecting one side to the engine chimney or stack.
- (2) By carrying down a column of steam pipes and allowing steam to escape through a jet or nozzle in the closed compartment of the shaft.
- (3) By ventilating the pit, by erecting either a temporary or the permanent fan and connecting it with the pit.

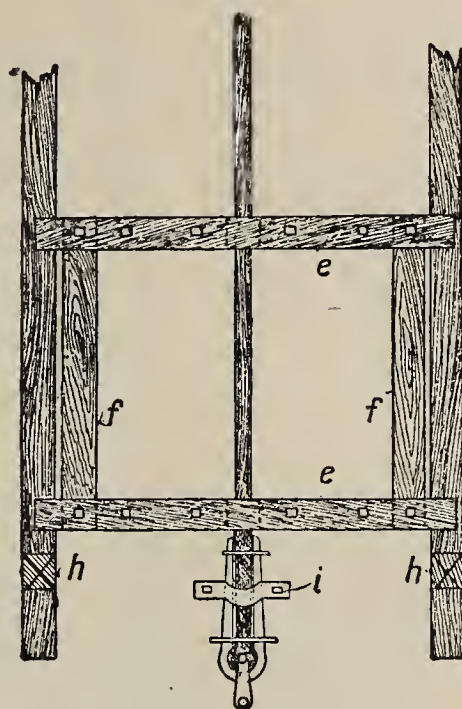
The first method is sometimes used, but, of course, would not be suitable if fire-damp were expected to be given off freely. Connecting the air drift to the flue of the chimney stack acts in the same way as a ventilating furnace underground by heating the air current, and thereby causing a circulation of air in the shaft.

The steam jet is a simple and handy way of ventilating shafts during sinking, and can be very easily applied, particularly if steam pipes have to be carried down to pumps in the shaft. Often the heat given off by these pipes is quite sufficient to ventilate the shaft without the aid of a steam jet.

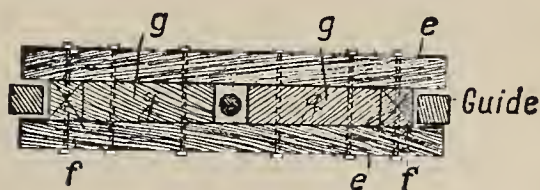
Probably the best method is, however, to use a small high-speed fan to force air down to the bottom of the pit through wooden boxing or sheet-iron tubes, and allow it to return up the whole area of the shaft outside the boxing or tubes. When a fan is used in this way, *i.e.* compressing, the shaft bottom is more quickly cleared of smoke after blasting, and allows the workmen to return with less loss of time than if the fan was working exhaustively, as by the latter system the whole of the smoke would have to be drawn off before the bottom of the pit was cleared. If wooden boxing is used, it may be constructed of flooring deals, $\frac{3}{4}$ or 1 in. thick, dovetailed into each other so as to make the boxing as nearly air-tight as possible. To make this more thorough the jointing should be treated with red lead or given one or two coats of paint. Boxing of this description should be 2 to 3 ft. square, and secured in the shaft by bearers. Sometimes pipes of large diameter made of thin sheet iron are used instead of boxes. The tubes are made in lengths of 9 or 10 ft., 18 to 24 in. diameter, and $\frac{3}{16}$ to $\frac{1}{4}$ in. thick, the sections being joined as shown in fig. 104.

The column of tubing should be carefully supported in the shaft by means of collaring, or secured by clamps to bearers. Fig. 105 shows an arrangement of fan with tubes for ventilating a pair of shafts while being sunk. Sheet-iron tubes are now largely employed for this purpose, as they are stronger than wood boxing, easier handled, and on the whole give better results.

Enlarging Shafts.—Shafts sometimes become too small for the



Elevation.



Plan.

FIGS. 102 and 103.—Rider for wood conductors.

amount of work required to be done in them, and require to be enlarged. If winding has to be completely stopped and underground operations abandoned while such enlargement is taking place, the best method would be to entirely fill up the shaft with some light, loose material, and start the enlargement from the surface, and carry on sinking in the usual way. If the regular work of the colliery has, on the other hand, to be carried on, and the shaft contains pipes, etc., which it is undesirable to interfere with, then enlarging a shaft is not such an easy matter to accomplish. Each case must of course be dealt with according to the circumstances. The enlargement of a shaft of which the writer has personal knowledge was carried out in the following manner.

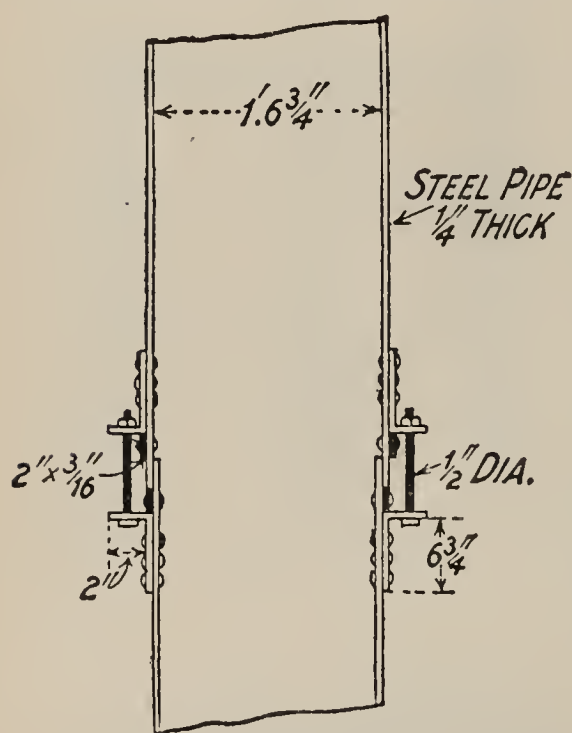


FIG. 104.

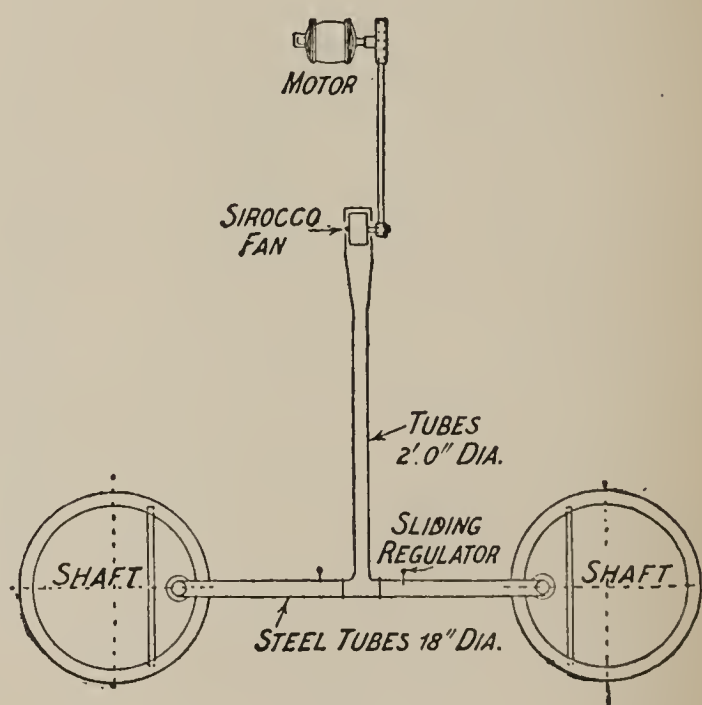


FIG. 105.

The colliery consists of two rectangular shafts, one being used as an up-cast and the other as a down-cast. The lining of the former showed signs of giving way, and the shaft had also departed from the vertical, while repairs of the wood lining had made it smaller than it was originally. It was determined to renew the whole of the lining, render the shaft vertical and enlarge it somewhat, while at the same time the whole of the winding was to be carried on at the down-cast.

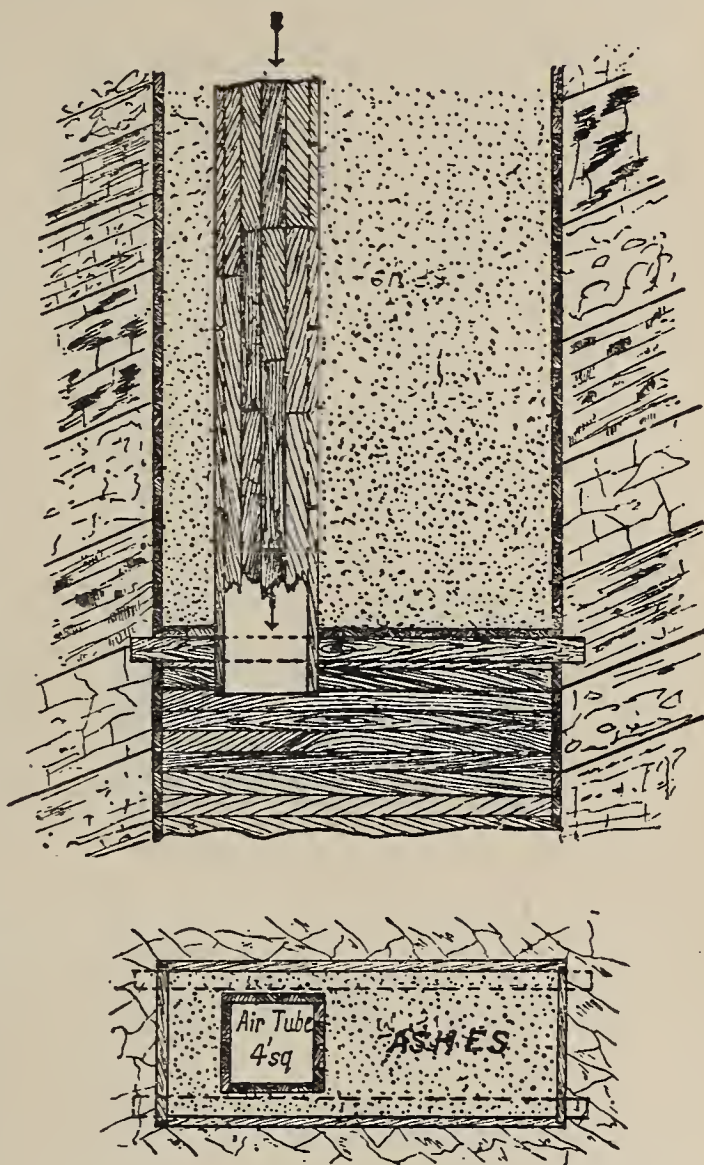
To have filled up the pit completely would have stopped ventilation, and consequently stopped work by the colliers. The enlargement was therefore carried out in stages of 10 fms. or so, by putting in a scaffold, resting on strong beams, in the shaft, this scaffold being completely closed with the exception of an opening of about 4 ft. square, to allow of a wood box passing through for the purposes of ventilation (see figs. 106, 107). This air-box, con-

structed of planking 9 in. $\times$ 3 in., firmly fitted together, was carried down past the scaffold for a short distance, and also up to the surface, and connected to the fan. The shaft was now filled in with ashes to the surface. The enlarging of this portion of the shaft was then proceeded with and new lining put in. The other sections of the shaft were dealt with in a similar manner until the whole shaft had been renewed to a depth of nearly 200 yds. The work was carried out expeditiously, and the whole of the output was dealt with at the other shaft. This method may be easily understood from figs. 106, 107.

Another method which the author has seen used for enlarging shafts is to use, instead of the wooden boxing described above, a wrought-iron or steel cylinder, 3 or 4 ft. diameter and 18 or 20 ft. long, an old flue of a Lancashire boiler serving well for the purpose (figs. 108, 109). A strong scaffold is put in, leaving an opening for the tube to move through, and the pit is filled with ashes or other débris as already described. The tube is hung by a steel wire rope led from a steam winch on the surface, and as each section of enlargement is carried on, the tube is lowered so as to always keep the top of it a little distance above the filled-in débris. By this

method a scaffold requires to be put in and the shaft enlarged in sections of 18 or 20 ft., or according to the length of the tube used. This system has the advantage that the iron tube is not so easily damaged as the wood boxing if blasting has to be resorted to.

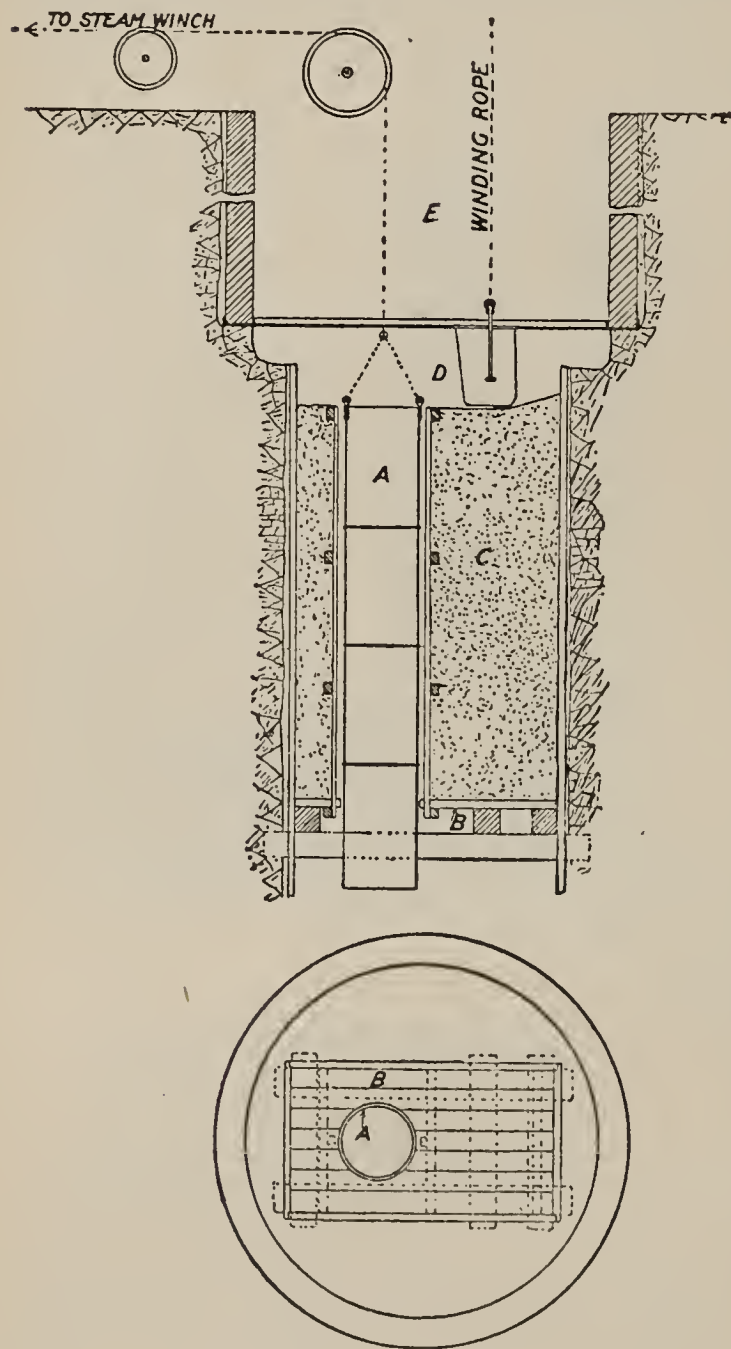
At the Strangeways Hall Colliery, near Wigan,* the up-cast shaft was enlarged in somewhat similar way from a diameter of 10 to $15\frac{1}{2}$ ft. The work of enlarging had to be carried out, and at the same time the shaft kept available for winding men and coal. Coal was drawn from 6 a.m. to 2 p.m., and the process of enlarging done



FIGS. 106 and 107.—Timbering of shafts.

* *Trans. Inst. Min. Engs.*, vol. xxxv. p. 330.

from 2 p.m. to 6 a.m. Commencing from the surface, the shaft was excavated to 24 ft. square, and pitch pine timbers 18 in. square put in every 7 ft. in depth, with 7 in. $\times$ 3 in. packing deals behind these timbers. As the loose ashes and soft ground were taken out, the old brickwork was left standing to act as a fence and to keep



FIGS. 108 and 109.

the smoke from the men, being taken down as the sinking proceeded, about 7 ft. at a time, until solid ground was reached. After the hard strata were reached, the shaft was excavated circular in shape to an outside diameter of 17 ft. When blasting had to be done, an iron tube built of $\frac{1}{2}$ -in. plates, $18\frac{1}{2}$ ft. long and 8 ft. 3 in. diameter, was slung in the shaft by means of two crabs on the surface attached by wire ropes 1 in. diameter, with the large winding rope as an additional safety. To prevent stones from falling down the shaft through the space between the tube and the brickwork of the shaft, a segment was bolted on to the bottom of the tube. The tube was also used to fix the boring machines against while shot-holes were being drilled, care being taken not to drill the holes below the bottom of the tube. The tube was always kept 4 ft. below the bottom of the holes to be

fired. Skeleton rings were placed 4 ft. apart, with backing deals behind, to protect the sides until the new brickwork was put in. When it was necessary to start the building up of a section of brickwork, the tube was lowered below the level of the bricking ring. The whole depth of the shaft, 2160 ft., was successfully enlarged in this way.

Deepening Shafts.—When the upper seams of a colliery get worked out, it may become necessary to deepen the shaft in order to

win the lower seams. This may be carried out in a variety of ways according to circumstances.* One of the simplest methods is to cover the sump over with a movable scaffold, which is put in position while coal is being wound and taken out during the period the sinking is proceeding. A spare length of rope is attached at one end to the cross-bar of the cage, and the other end fixed to a sinking kibble. Sinking is carried out in the ordinary way, the débris being tipped from the kibble at the shaft inset into empty tubs kept ready for the purpose. If the débris cannot be stowed in the workings, sufficient tubs to last the sinking shift must be kept in readiness, and afterwards drawn to the surface and emptied. The above system can, of course, only be carried out when there is not much water given off, or when the water can be led to a lodgment apart from the shaft. If coal-winding is entirely suspended, then the deepening of the shaft can be carried on as an ordinary sinking, the cages being taken out of the shaft altogether.

Sometimes a number of the tubs are fitted with lugs or eye-pieces at the corners, to which four chains can be attached. The chains are attached to the spare length of rope slung below, and fixed to the cage as described above. The débris is filled direct into the tubs, and they are raised to the inset, where they are drawn clear of the shaft by means of a pair of folding doors fitted with rails to suit the gauge of the tubs. The lower end of the spare rope is fitted with a spring hook, so that it can be detached and attached quickly. This system saves the labour of emptying the débris from the kibble.

APPENDIX.

SPECIFICATION FOR SINKING AND WALLING ——— PIT FOR THE ——— COAL CO., LTD.

1. The pit to be sunk from the point to which it is already sunk to a depth of — fathoms, or any less distance in the option of the manager, to be — ft. in diameter in the clear of the walling, and to be perfectly plumb, and truly circular in form.

2. The pit to be walled throughout with brickwork well bedded in and jointed with lime or cement mortar, all joints in the inner or shaft side of the walling being carefully drawn with the trowel. The brickwork to be properly bonded, every third course consisting entirely of headers, to be not less than 9 in. thick, increasing to 14 in. or more in thickness where such increased thickness may be deemed necessary by the Company's manager.

3. The mortar to consist of lime or cement mixed with sand, mine dust, or engine ashes, in such manner and in such proportions as the Company's manager may direct.

* For detailed information as to deepening and enlarging shafts, see *Modern Coal-Mining*, by Prof. D. Burns and G. L. Kerr, Part III. pp. 361-374.

4. The brickwork to rest upon curbs of cast iron or wood, which shall be laid upon carefully prepared beds at such distance apart as the nature of the strata sunk through may render necessary. Where required by the Company's manager these cast-iron or wooden curbs shall be left in to form water garlands or rings, in which case the brickwork shall be set back from the plumb of the walling so as to leave a clear water space of not less than $2\frac{3}{4}$ in. in width. The brickwork above this to be gradually overhung, so that it may coincide with the plumb of the walling at a height of 5 ft. or thereby above the water garland. All curbs of cast iron or of wood not required to serve as water garlands to be removed when closing up the length of the walling immediately underneath.

5. The brickwork for a height of not less than 6 ft. above each curb to be built solidly against the strata, each course being carefully grouted with mortar; at intervals of 6 ft. in the circumference of the pit, or at such other intervals as the Company's manager may direct, water runs or 'hassans' shall be formed in the brickwork, so that water given off the strata may be conducted down into the garlands.

6. The space behind the brickwork above or between these solid belts to be carefully packed with concrete or with rock stone, with or without mortar as may be directed, and no shale or other soft material shall be used for this purpose. The packing shall be put in on the completion of every second or third course, so as to ensure that the whole space is properly packed.

7. All the curbs and water garlands to be properly and accurately put in, and set by centre line from top of pit, and to be put in at such places during the sinking as the Company's manager shall direct.

8. The contractor shall find and pay all sinkers, banksmen, tippers, enginemen, bricklayers, and all other labourers requisite for carrying on the whole work with the utmost despatch, but all such shall be subject to the approval of the manager.

9. The Company will provide the necessary engine power and steam to the engines, tackling, timber, tar, nails, oil, tallow, stores for engines, bricks, and other materials, but not tools. The contractor must give due notice, so that a proper supply of all requisite materials may be at hand. The contractor will provide all tools for carrying on the work, as well as backskins and hats. The contractors will provide all blasting materials, which will be supplied by the Company to them at market prices. The contractor must satisfy himself as to the condition and strength of the machinery, boilers, and plant on the ground, and of all material and tackling provided for him, it being understood that, if dissatisfied, or in the event of an accident, the contractor may stop work until the necessary alterations or repairs are made; and he shall have no claim against the Company for loss or damage should the work be stopped through the plant, etc., supplied to him by the Company proving insufficient for the purpose for which they were supplied. The engines and all other plant to be kept clean and in good repair by the contractors.

10. The Company shall furnish all bratticing, rhones, or tubes that may be found necessary for ventilation, also all pumps and pipes necessary to remove water from the pits, but the contractor shall, except as after specified, fix the same in the shafts and maintain them.

Should the Company desire to drive mines and put down bores to run off water so as to permit the pit being sunk dry, the contractor shall stop work when required, to permit of such being done, and shall have no claim for stoppage of his operations.

11. The sinking to be carried on continuously on week-days and nights, and as many men employed in the pit as may be necessary to carry on the work with the utmost despatch and to the satisfaction of the Company or their manager, but there shall be not fewer than six experienced sinkers in the bottom of the pit during each shift.

The material excavated from the pit, so far as required, will be laid down by the contractor so as to level up the surrounding surface, and the surplus put in waggons, or alternatively taken by the Company on being emptied on bogies at pit mouth.

12. The contractor shall keep a careful section of the strata passed through by the pit, and shall deliver such section to the Company's manager as required.

13. The sinking shall be measured on each alternate Tuesday, and payments made on account on the following Friday, under deduction of 10 per cent., which shall be retained by the Company until the due completion of the contract has been certified by the Company's manager.

14. No extra work shall be paid for except on the production of a written order for the execution of the same signed by the Company's manager.

15. The contractor shall have a competent foreman on the premises during each shift, and he binds himself to carry on all the operations in conformity with, and to cause all the necessary inspections and reports to be made as provided by the Coal Mines Regulation Acts and the special rules for the time being in force at the collieries; and he shall be responsible to the proper authorities for the safety of all persons employed in or about the pit, or about the work or machinery in connection therewith, as required by said Acts and special rules.

16. The contractor shall also bind himself to commence operations immediately after the acceptance of his offer, and to sink the pit and complete the whole work in the best and most tradesman-like manner within a period of eighteen months from the commencement of operations, under deduction from the contract sum of twenty pounds sterling per week for every week that the work may remain unfinished beyond that period. The said deduction shall not be construed to be of a nature of a nominal penalty, but as ascertained damages payable to the Company, and forming an inherent condition of the contract price.

17. The whole work to be carried on rapidly and continuously without unnecessary loss of time, and to the entire satisfaction of the Company or their manager, who shall be sole judge of the character and efficiency of the work. The Company, by their manager or others, shall have power to inspect the work at all times, and should the contractor fail to execute or carry on the work to the satisfaction of the manager, or comply with any of the stipulations herein before specified, or should he allow the work at any time to be at a standstill for forty-eight hours, except in case of accident or under the sanction of the manager, or reduce the number of men in the shifts without the manager's consent, the Company shall have power to retain 10s. for each shift the sinking is stopped or retarded, or in their option, and without any legal proceedings, to take the work out of the contractor's hands, and carry it on themselves or relet it to others, and take all such measures as they may deem necessary for the prosecution and completion of the work at the expense of the contractor, who shall have no claim against the Company for any saving which they may effect by letting the work to another contractor or otherwise.

18. Should any dispute or difference arise between the contractor and

the Company in connection with, or during the performance of, the work, either with regard to the nature or extent of the work estimated for, manner of execution, rate of progress, or from any other cause whatever, the same shall be referred to

whose decision shall be final and binding on both parties; and no stoppage or suspension of work, or of any part thereof, pending the consideration and decision of such dispute or difference, shall take place unless with his consent.

19. A formal deed of contract, embracing all the clauses of this specification, and all other usual and necessary clauses, shall be executed between the Company and the contractor at the joint expense of the contracting parties, the original contract being deposited with the Company, and should the contractor require a copy, the same will be furnished to him at his own expense.

20. Tenders to be lodged with the Company at _____ on or before _____.

The Company do not bind themselves to accept the lowest or any other tender.

The Coal Company reserve power to lay down a compressor to operate boring drills for the contractor's use. In such an event he will accept a lesser fathomage rate as may be mutually arranged.

On the other hand, the Company agrees to pay to the contractor a sum of fifteen pounds sterling per week for every week by which he completes the work earlier than the time stipulated for.

6th August 19

FORM OF TENDER.

I hereby undertake to execute the work, conform in every respect to the foregoing specification, at the following rates:—

Sinking per fathom—first 50 fathoms,	.	.	.	.	£
„ „ „ second 50 fathoms,	.	.	.	.	
„ „ „ remainder,	.	.	.	.	
Extra per fathom when water exceeds 8 kettles per hour,	.				
Walling per fathom,	.	.	.	.	

Any advance or reduction in miners' wages that may take place during the carrying on of the work, affecting the — district as a whole, shall have effect on the above-named rates for sinking as follows:—A rise or fall of — per cent. on the 1888 standard shall imply an increase or reduction of — on each — of the contract rates, and so in proportion for a greater or less advance or reduction than — per cent.

(Signature of Contractor.)

(Signature of Agent or Manager }
on behalf of the Company.) }

The foregoing draft of specification is only given as a guide, and should be altered or added to, to suit the circumstances of each individual sinking.

Two copies of the contract should be signed, one to be kept by the Company and the other by the contractor.

CHAPTER IV.

EXPLOSIVES.

Definition.—An explosive is a substance the decomposition of which results in the sudden expansion of its components into a volume of heated gases many times exceeding its original bulk.

The strength of an explosive depends upon the volume of gases liberated, the rate at which decomposition proceeds, and the temperature of ignition. The gases liberated by the ignition of gunpowder, for instance, amount to about 2000 times the original volume of the powder used. The force exerted by ordinary blasting powder has been ascertained to be about 22,000 foot-pounds per sq. in.

The actual work performed by any explosive used in blasting operations is limited by incomplete combustion, compression, etc., by waste of energy in cracking and in heating material not displaced, and by the escape of gases through the shot-hole and through fissures in the rock.

The efficiency of explosives, *i.e.* the proportion borne by the work done to the theoretical energy liberated, has been estimated to range from 4 to 33 per cent.

Classification of Explosives.—Explosives may be classified in different ways, such as rending and shattering, or high and low, but the usual systems adopted are: (1) according to method of igniting; (2) according to composition. Under the first head they may be subdivided as follows:—

Explosives the decomposition of which is due to simple combustion, as in the case of ordinary gunpowder.

Explosives in which detonation occurs simultaneously throughout their mass, as Ammonite, Amvis, Bellite, Roburite, etc.

Explosives which partly detonate and partly burn, such as Carbonite, Kynite, Gelignite, etc.

Classification according to Composition (Cundhill):—

Gunpowder, ordinarily so-called.

Nitrate mixtures other than gunpowder.

Chlorate mixtures.

Nitro-compounds containing nitroglycerine, including the dynamite series.

Nitro-compounds not containing nitroglycerine, such as guncotton, etc.

Miscellaneous explosives.

Gunpowder is largely used. It is cheap, comparatively slow in action, and therefore suitable for coal and soft rocks, and less dangerous than some of the nitro-compounds. On the other hand, it is very dangerous in the presence of fire-damp and coal-dust, and its use is now prohibited in certain collieries by order of the Home Secretary.*

Gunpowder, if exploded in large quantities, is also dangerous to life, especially in badly ventilated workings, owing to the large percentage of carbon monoxide it gives off, and no explosives which give rise to this gas ought to be used for extensive blasting in mines, because of the risk of injury to health, and also because even small traces of carbon monoxide have been proved to render mixtures of coal-dust and air highly explosive, a point frequently overlooked in experiments with explosives. On firing $1\frac{1}{4}$ lbs. of blasting powder, over 3 cub. ft. of combustible gas, consisting chiefly of carbon monoxide, would be produced, and this, when mixed with pure air, would give over 10 cub. ft. of an explosive, or, at least, a rapidly burning mixture. The approximate composition of ordinary gunpowder is: Nitrate of potassium (saltpetre), 75 per cent.; carbon, 15 per cent.; sulphur, 10 per cent., along with small percentages of potassium sulphate and water.

When gunpowder is exploded 56 per cent. of solid matter is formed and 44 per cent. of gas, or roughly, the solid matter is to the gaseous as 6 to 4. Ordinary blasting powder explodes at 600° F.

By the explosion of ordinary powder the following gases are produced † :—

	Volumes per cent.		Volumes per cent.
Carbon dioxide, . . .	32·15	Sulphuretted hydrogen, .	7·10
Carbon monoxide, . . .	33·75	Marsh gas, ...	2·73
Nitrogen, ...	19·03	Hydrogen, ...	5·24
			100·00

Bichel ‡ gives the following analysis of the decomposition products of gunpowder :—

Gases per cent.	Solids per cent.
22·8 CO ₂	30·6 K ₂ CO ₃ (potassium carbonate)
10·3 CO	6·4 K ₂ SO ₄ (potassium sulphate)
10·3 N	19·6 K ₂ S ₃ (potassium sulphide)

From the foregoing it will be seen that gunpowder gives off a large percentage of carbon monoxide, which, as already stated, is

* Sec Coal Mines Explosives Orders, 1897–1904.

† *Ore and Stone Mining*, Sir C. Le Neve Foster, sixth edition, p. 223. C. Griffin & Co., Ltd.

‡ *New Methods of Testing Explosives*, by C. E. Bichel, 1905, Table III. C. Griffin & Co., Ltd.

very objectionable. The sulphur is also objectionable, and is by many makers reduced to a minimum. Gunpowder is very effective in breaking down coal, and is readily kept in good condition.

Gunpowder, where permitted, is now almost universally used in the form of compressed pellets or cartridges for blasting in coal mines. In this form it is much more effective, and not so dangerous as when used in the loose state.

The following are typical blasting agents of the gunpowder type:—

	Special Bulldog Powder.	Bobbinite No. 1.
Nitrate of potassium, ...	84-86	62-65
Charcoal, ...	12-13	17-19½
Moisture, ...	...	2½
Carbonate of magnesium, . . .	2½-3½	...
Sulphur, ...	...	1½-2½
Sulphate of ammonium, . . . }	...	13-17
Sulphate of copper, . . . }	...	

To be compressed to a pellet, density 1.42, and, in the case of bobbinite, to be coated with paraffin wax melting at 120° F. Both explosives to be fired by electric fuse containing 5 grains of gunpowder, or with equivalent efficient explosive.

Other varieties of gunpowder, introduced of recent years, are the following:—

Constituents.	Argus Powder.	Earthquake Powder.
Nitrate of potassium, ...	70-82	78-81
Carbon, ...	17-20	19-28
Distilled or pure sulphur, . .	½-1	⅓ (optional)
Oxalate of ammonium, ...	...	...

Constituents.	Elephant Brand.	Oxalate Powder.
Nitrate of potassium, ...	74-76	63-73
Carbon, ...	14½-15½	12-15½
Pure or distilled sulphur, . .	9-11	...
Oxalate of ammonium, ...	...	13¼-16½

Chlorate Mixtures.—Explosives containing chlorate of potash are regarded as too dangerous for mining purposes, being peculiarly sensitive to slight shocks, blows, etc.

Nitrate Mixtures other than Gunpowder.—In this class of explosives nitrate of sodium (Chili saltpetre) is substituted for potassium nitrate. Such mixtures are cheaper, but are absorbent, or deliquescent, *i.e.* they take up moisture from the atmosphere and are therefore unsuitable for mining purposes.

Nitro-compounds containing Nitroglycerine.—In this class are included all those 'high' explosives which are so useful in mining, and particularly in blasting operations in hard rock. Nitroglycerine

is a light yellow, oily liquid, having a specific gravity of 1.6. It freezes at 40° F., and explodes with great violence at 360° F., or when subjected to a sudden shock. It is less sensitive to blows and detonation when frozen than when in the liquid state. Its use in the pure state is forbidden in Britain.

Blasting Gelatine.—This is one of the most powerful explosives used in mining. Its manufacture is both difficult and dangerous, but when once made it is one of the safest of explosives. It contains 93 per cent. to 95 per cent. of nitroglycerine, and 5 per cent. to 7 per cent. of nitro-cotton.

It is less rapid in detonation than dynamite, and is quite insoluble in water, in which it may be kept without deterioration. In its plastic state it is less sensitive to shocks or blows than dynamite, but when frozen it is more so. Its relative sensibility to detonation compared with dynamite has been accurately ascertained, 0.8 grain of 'cap mixture' being required to explode a given charge of No. 1 dynamite, while the best blasting gelatine requires, for the same charge, 3 grains.

Relative efficiency of different explosives with same charge:—

Blasting gelatine (93 per cent. nitroglycerine and 7 per cent. nitro-cotton), ...										1000.00
Nitroglycerine, ...										907.14
No. 1 dynamite, ...										642.85
No. 2 dynamite, ...										378.57
Gunpowder (extra strong), ...										194.28

It will thus be seen that blasting gelatine is about three times more efficient than ordinary dynamite, and about five times stronger than gunpowder.

Dynamite.—This explosive is manufactured by impregnating diatomaceous kieselguhr, a spongy earth obtained from Germany, with nitroglycerine.

Its average composition is: Nitroglycerine, 75 per cent.; kieselguhr, 25 per cent.

When in a proper condition dynamite, blasting gelatine, and other explosives containing nitroglycerine are plastic, may be safely handled, and are very convenient for use as explosives. Irregularly shaped holes are easily charged with them, and they do not explode at ordinary temperature either by spark or flame, but require detonation. When dynamite cartridges are at a temperature below 32° F. they will only detonate with difficulty. When their temperature falls below 40° to 42° F. they are not in a safe condition, owing to their increased sensitiveness to shock. When in a frozen condition they should only be thawed by the warming-pans provided by the makers, and not heated in tin cans over fires or carried about in trouser pockets, as is too often done by miners.

Relative Efficiency of Gunpowder and Dynamite.

For Equal Weights.		For Equal Bulks.	
Gunpowder	=1·00	Gunpowder	=1·00
No. 1 dynamite	=3·75	No. 1 dynamite	=6·00
No. 2 dynamite	=2·00	No. 2 dynamite	=3·30

The use of dynamite results in economy of labour and tamping, loose sand being sufficient. It can be used in watery rock, and gives off little smoke, but dangerous fumes, and workmen ought not to return too quickly to a place where charges of dynamite or other nitroglycerine explosives have been exploded.

Permitted Explosives.—The Secretary of State for the Home Department has had conferred on him by section 6 of the Coal Mines Act the power to prohibit *any* explosive, on his being satisfied that it is, or is likely to become, dangerous, from being used in any mine, or in any class of mines, either absolutely or subject to conditions. “In all coal mines in which inflammable gas has been found within the previous three months in such quantity as to be indicative of danger,” and “in all coal mines which are not naturally wet throughout, the use of any explosive other than a permitted explosive is absolutely prohibited in all roads, and in every dry and dusty part of the mine.” *

Before an explosive can be admitted to be a permitted explosive it must undergo specified tests in the Government Testing Station at Woolwich. The following is a complete list of permitted explosives as authorised by the Explosives in Coal Mines Order, February 1912:—

Nitrate Mixture (any preparation, other than gunpowder ordinarily so-called, formed by the mechanical mixture of a nitrate with any form of carbon or with any carbonaceous substance not possessed of explosive properties, whether sulphur be or be not added to such preparation, and whether such preparation be or be not mechanically mixed with any other non-explosive substance):—

Ammonal B	Dahmenite A	Virite
Aphosite	Electronite	Westfalite No. 1
Bobbinite	Ripping ammonal	Westfalite No. 2

Nitro-compound (any chemical compound possessed of explosive properties, or capable of combining with metals to form an explosive compound, which is produced by the chemical action of nitric acid—whether mixed or not with sulphuric acid—or of a nitrate mixed with sulphuric acid, upon any carbonaceous substance, whether such compound is mechanically mixed with other substances or not).

Division I. (comprising any chemical compound or mechanically

* Explosives in Coal Mines Order, 21st February 1910 (1 (a), (b)).

NITRO-COMPOUND.

Composition.	Abbicite.	Arctic Carbonite.	Arkite.	Britonite.	Cambrite.	Carbonite.	Celtite.
Nitrate of ammonium	78-82	..	..	..	..	..	..
Nitro-glycerine	9-11	14·5-16·5	51-54	25-27	23-27	25-27	56-59
Wood-meal*	8-10	28-31	6-8	39-43	36-42	39-42	8-9
Charcoal*	..	..	..	..	..	..	..
Nitro-hydrocarbons	..	9-12	..	..	..	..	..
Nitrate of potassium	..	40-44	21-23	31-34	25·75-32	} 30-36 {	} 17-21 {
Nitrate of barium	..	..	..	..	3·25-4·5		
Carbonate of calcium	..	0-0·5	..	..	} 0·5 {	} .. {	} .. {
Carbonate of sodium	..	..	..	0-0·5			
Nitro-cotton	..	..	3-4	..	..	..	2·5-3
Oxalate of ammonium	..	..	14-16	..	8-0	0·5	11-13
Chalk	..	..	0-0·5	..	..	..	..
Sulphuretted benzol	..	..	..	..	0·5	0·5	..
Starch*	..	..	..	..	..	..	..
Sulphate of magnesium	..	..	..	..	..	..	..
Chloride of potassium	..	..	..	..	..	..	..
Paraffin oil	..	..	..	..	..	..	..
Vaseline	..	..	..	..	..	..	..
Collodion cotton	..	..	..	..	..	..	..
Di-nitro-toluol	..	..	..	..	..	..	..
Castor oil	..	..	..	..	..	..	..
Mineral jelly (free from acid)	..	..	..	..	..	..	..
Moisture	0·5-2·5	3	..	..	..	..	0·5-1·5

Composition.	Monobel Powder.	Nobel Ammonia Powder.	Nobel Carbonite.	Normanite.	Oaklite No. 1.	Oaklite No. 2.	Phoenix Powder.
Nitro-glycerine	9-11	7-9	25-27	32·5-34·5	24-26	9-11	28-31
Nitrate of ammonium	78-82	82-86	..	..	..	78-81	..
Carbonate of calcium	..	..	..	..	..	..	..
Chalk	..	..	..	..	..	..	..
Nitrate of potassium	..	..	28-32	42·5-46·5	32·5-35·5	..	30-34
Nitrate of barium	..	..	3·5-4·5	..	..	..	..
Carbonate of calcium	..	..	} 0·5 {	{ .. {	..	..	..
Carbonate of sodium	..	..			..	..	..
Wood-meal*	8-10	6-8	39-42	7-9	33·5-36·5	8-10	33-37
Charcoal*	..	..	..	1-2	..	..	..
Sulphuretted benzol	..	..	0·5	..	..	..	..
Nitro-cotton	..	..	..	1-2	0·5-1·5	0·25-0·75	1
Oxalate of ammonium	..	..	..	10-12	..	..	..
Carbonate of magnesium	..	..	..	..	0·5	0·25	..
Nitrate of sodium	..	..	..	..	..	..	..
Tri-nitro-toluol	..	..	..	..	..	..	..
Castor oil	..	..	..	..	..	..	..
Di-nitro-toluol	..	..	..	..	..	..	..
Sulphate of magnesium	..	..	..	..	..	..	..
Bi carbonate of sodium	..	..	..	..	..	..	..
Moisture	0·5-2·5	0·5-2	..	..	3·5-6·5	1-2·5	2-6

* Dried at 100° C.

DIVISION I.

Clinite.	Clydite.	Cornish Powder.	Dominite.	Dragonité.	Excellite.	Extra Carbonite.	Fracturite.	Haylite.	Kolax.	Kynite.	Kynite (condensed).
45-48	23-27	54-57	57.5-60.5	34-37	80-84	34-36	51.5-53.5	25-27	24-26	25-27	24-26
..	35.5-41.5	5-7	3.5-5.5	11-13.3	7-9	31-35	5-7	12-14	29-32	39-42	5-7
..	..	..	..	..	..	..	..	..	..	..	..
..	..	..	..	..	..	..	..	..	..	..	..
..	..	17-20	17.5-19.5	43-46	..	24-27	21-25	19-21	25-27	..	..
..	29.5-36	..	..	..	..	3-5	..	19-21	4-6	30-36	31.5-34.5
..	} 0.5 {	..	..	..	..	..	..	..	..	..	0.5
2.5-3.5	..	3-4	3-4.5	2-3	..	0.5	..	..	..	..	..
..	8	..	7-10	..	..	0.1-0.5	3-4	0.5-1.5	..	..	..
..	..	..	..	..	..	..	14-16	10-12	..	..	..
..	0.5	..	..	..	..	..	..	..	..	0.5	..
41-45	..	..	..	..	..	..	..	..	8-10	..	30-32.5
..	..	16-18	..	..	..	..	..	..	..	..	..
..	..	..	3-5	..	..	..	..	..	..	..	..
..	..	..	0.5-1	..	..	..	..	..	..	..	..
..	..	..	..	5-6	..	..	..	..	..	..	..
..	..	..	..	..	0.5-1.5	..	..	..	..	..	..
..	..	..	..	..	2.5-3.5	..	..	..	..	..	..
..	..	..	..	..	0.5-1.5	..	..	..	..	..	..
..	..	..	..	..	..	..	..	..	..	..	..
6-9	..	1	2	..	2	2-5	..	6-8	4-6	..	3-6

Pit-ite.	Pitseal Powder.	Rexite.	Rippite.	Russelite.	Samsonite.	Stomonal.	Stonax.	Stow-ite.	Swalite.	Tutol.	Victorite.
25-27	7-9	6.5-8.5	59.5-62.5	40-42	57-60	7-9	50-53	58-61	58.5-61.5	24-26	25-27
..	82-86	64-68	..	..	..	83-86	..	..	..	..	..
..	..	..	..	..	..	..	..	..	..	..	..
..	..	..	..	..	..	..	..	..	..	..	..
..	..	..	18-20	24-26	17-19	..	16-18	17-20	16-18	32-34.5	..
31-35	..	..	..	..	..	..	..	..	..	1-3	32-36
} 0.5	{	..	..	1	..	..	..	..	..	..	} 0.5
		..	..	..	..	..	..	..	..	..	
40-43	6-8	3-5	3.5-5.5	3-5	5-7	6-8	5-7	6-7	4.5-6.5	35-38	38.5-41.5
..	..	..	..	..	..	..	..	..	..	..	..
..	..	..	..	..	..	..	..	..	..	..	0.5
..	..	..	3.5-4.5	1.75-2.75	3-4	..	3-4	4.5-5	3.5-4.5	..	..
..	..	..	9-11	22-24	12.5-14.5	..	8.5-10.5	11-13	12.5-14.5	..	..
..	..	..	..	..	..	..	..	..	..	..	..
..	..	13-15	..	..	..	..	..	..	..	..	..
..	..	6.5-8.5	..	4 5-6.5	..	..	..	..	..	..	..
..	..	..	0.5-1.5	..	..	..	..	..	..	..	..
..	..	..	..	..	..	..	4.5-6.5	..	..	..	..
..	..	..	..	..	..	..	5-6	..	..	..	..
..	..	..	..	..	..	..	..	..	..	0.5	..
..	0.5-2	0.5-1.5	1	1	1.5	0.5-2	1	..	1	2-6	..

NITRO-COMPOUND.

Composition.	Ammonite.	Amvis.	Bellite.	Curtisite.	Dreadnought Powder.	Faversham Powder.
Nitrate of ammonium	87-89	88-91	82-85	86-90	73-77	88-93
Di-nitro-naphthalene	11-13	..	..	..	..	..
Wood-meal*	..	4·6	..	..	..	..
Di-nitro-benzol	..	} 4-6 {	15-18	..	..	..
Chlorinated naphthalene	..			..	..	..
Tri-nitro-toluol	} ..	..	..	7-9	3-5	9-11
Di-nitro-toluol		..	..	3 5	..	..
Mono-nitro-naphthalene	..	..	..	..	14-17	..
Nitrate of sodium	..	..	..	..	4-6	..
Chloride of ammonium	..	..	..	..	0·1	..
Oil red	..	..	..	..	..	..
Amido compound	..	..	..	..	..	..
Tumeric	..	..	..	..	..	..
Oxalate of copper	..	..	..	..	..	..
Oxalate of ammonium	..	..	..	..	..	..
Graphite matter	..	..	..	..	..	..
Colouring matter	..	..	..	..	..	..
Metallic Aluminium	..	..	..	..	..	..
Curcuma charcoal	..	..	..	..	..	..
Flour*	..	..	..	..	..	..
Moisture	0·5	0·5	0·75	0·5	1	1

* Dried at 100° C.

CHLORATE MIXTURE.

DIVISION I.

Composition.	Permonite.
Perchlorate of potassium	31-34
Nitro-glycerine	3-4
Nitro-cotton	0·1-0·5
Nitrate of ammonium	39-43
Tri-nitro-toluol	11-13
Starch (dried at 100° C.)	5-9
Wood-meal (dried at 100° C.)	1·5-3·5
Moisture	2·5

DIVISION II.

Fortex.	Good Luck.	Minite.	Nationalite.	Negro Powder.	Odite.	Rendite.	Roburite No. 3.	St Helen's Powder.	Tritanite No. 1.	Withnell Powder.
76-80	80-84	87-90·5	90-94	86-90	86-90	87-91	86-89	92-95	85-88	88-91
..	..	..	..	..	..	..	..	..	..	..
..	0·5-1·5	..	..	..	..	..	9-13	..	..	..
..	..	..	..	..	..	..	2	..	..	..
..	..	9-11	7-9	9-11	{ 11-13 }	9-11	..	3-5	6-8	4-6
..	..	..	..	..	..	..	..	..	..	..
..	..	..	..	..	..	..	..	..	..	..
..	..	..	..	..	..	..	..	..	..	..
20-23	..	..	..	..	..	..	..	..	..	..
..	9·5-11·5	..	..	..	..	..	..	..	..	..
..	5·5-6·5	..	..	..	..	..	..	..	..	..
..	..	0·5-2	..	..	..	..	..	..	..	..
..	..	..	..	1-3	..	..	..	..	..	..
..	..	..	..	0·1	..	..	..	..	..	..
..	..	..	..	..	..	..	..	..	..	..
..	..	..	..	..	..	..	..	..	4·5-6·5	..
..	..	..	..	..	..	..	..	..	..	4-6
1	1	0·5	1	1	0·5	2	0·5	1	1	1·5

CHLORATE MIXTURE.

DIVISION III.

Composition.	Amasite.	Colliery Steelite.
Perchlorate of ammonium . . .	32-36	...
Nitrate of sodium . . .	29-33	...
Myrobalans (dried at 100° C.) . .	30-36	...
Agar agar . . .	0·15-0·5	...
Chlorate of potassium . . .	...	72·5-75·5
Oxydised resin . . .	...	23·5-26·5
Castor oil . . .	...	0·5-1·5
Moisture . . .	3·5	1

mixed preparation which consists either wholly or partly of nitro-glycerine, or of some other liquid nitro-compound):—

Abbicite	Extra carbonite	Pit-ite
Arctic carbonite	Fracturite	Pitsea powder
Arkite	Haylite	Rexite
Britonite	Kolax	Rippite
Cambrite	Kynite	Russelite
Carbonite	Kynite condensed	Samsonite
Celtite	Monobel powder	Stomonal
Cliffite	Nobel ammonia powder	Stonax
Clydite	Nobel carbonite	Stow-ite
Cornish powder	Normanite	Swalite
Dominite	Oaklite No. 1	Tutol
Dragonite	Oaklite No. 2	Victorite
Excellite	Phoenix powder	

Division II. (comprising any nitro-compound which is not comprised in the first division):—

- Ammonite	Fortex	Rendite
Amvis	Good Luck	Roburite No. 3
Bellite	Minite	St Helen's powder
Curtisite	Nationalite	Titanite No. 1
Dreadnought powder	Negro powder	Withnell powder
Faversham powder	Odite	

Chlorate Mixture (any explosive containing a chlorate).

Division I. (comprising any chlorate preparation which consists partly of nitroglycerine, or of some other liquid nitro-compound):—

Permonite.

Division II. (comprising any chlorate mixture not comprised in the first division):—

Amasite

Colliery Steelite

Other explosives are being tested from time to time and added to the list. It must be understood that the above list of permitted explosives does not form a guarantee by the Home Office that the explosives are safe under all conditions; it only signifies that those named have passed the Woolwich test, a test which many mining authorities think is quite inadequate for the purpose of defining a safety explosive. At best it is only a test of comparative safety, and the mine-owner is left to choose the explosive that he thinks may be safest to use under the conditions prevailing at any given colliery. To assist mine-owners, however, the Home Secretary issued a notice in October 1899, intimating an additional test to which explosives already upon the 'Permitted List' might be subjected. The proposed test was more severe than the original one, and explosives which passed it will be placed on a 'Special List.' This special test has now become the ordinary test.

NITRATE MIXTURES.

Composition.	Ammonal B.	Aphosite.	Bobbinite.*	Dahmenite A.	Electronite.	Ripping Ammonal.	Vitrite.	Westfalite No. 1.	Westfalite No. 2.
Nitrate of ammonium . . .	93-95.5	58-62	..	91.5-93.5	71-75	84-87	30-40	94-96	90-92
Metallic aluminium . . .	2.5-3.5	..	..	..	..	7-9	..	..	..
Nitrate of potassium . . .	..	28-31	62-65	..	..	..	33-38	..	3-5
Charcoal	2-3	3.5-4.5	17-19.5	..	..	2-3	10.5-12.5	..	..
Wood-meal	..	3.5-4.5	..	..	7-10	{ ..	..	..	..
Starch	..	..	..	..			..	..	..
Sulphur	..	2-3	1.5-2.5	..			4-5	..	..
Sulphate of ammonium . . .	..	..	13-17	{ ..	..	..	..	..	..
Sulphate of copper	..	..			..	..	..	..	..
Naphthalene	..	..	..	4-6.5	..	..	..	..	..
Bichromate of potassium . .	..	..	..	1.5-2.5	..	3-4	..	..	..
Nitrate of barium	..	..	..	..	18-20	..	..	..	..
Oxalate of ammonium . . .	..	..	..	..	..	..	9-12	..	..
Rosin	..	..	..	..	..	..	..	4-6	4-6
Moisture	1	1.5	2.5	1	0.5	1	1-2	0.5	0.5

* Bobbinite (2nd definition) has proportions of rice or maize starch and paraffin wax instead of sulphate of ammonium and sulphate of copper.

Detonators.—These are generally made of a small copper cylinder or shell containing a small quantity of fulminate of mercury, or a mixture of fulminate with chlorate of potash, the proportion of the latter varying from 5 to 10 per cent.

The Home Office issued an order in 1897 regarding the standard charge for detonators, which is to consist of a mixture of fulminate of mercury 80 per cent., and chlorate of potash 20 per cent., or some other explosive mixture of the fulminate class of not less strength than the above.* Different explosives require detonators of different strength to explode them, and the manufacturers of explosives generally recommend a certain class or strength of detonator for use in blasting the different explosives which they produce. These strengths are usually denoted by numbers, and the following are those commonly in use:—

	No. 1.	No. 2.	No. 3.	No. 4.	No. 5.	No. 6.	No. 6½.	No. 7.	No. 8.
Charge per 1000 in grammes, . . .	300	400	540	650	800	1000	1250	1500	2000
Individual charge in grains, . . .	4.6	6.2	8.3	10	12.3	15.4	19.2	23.1	30.9

For rendering explosives—*e.g.* some of the blasting powders, Nobel Ardeer powder, etc.—a detonator of the strength of No. 3 is commonly employed; for explosives of the nitroglycerine class No. 6 or No. 6½ is the most suitable. Miners and shot-firers should therefore be provided with the class of detonator most suitable for the explosive they

* For sizes of detonators required to be used with different explosives, see the Explosives in Coal Mines Order.

are using. As the whole success and safety of shot-firing almost entirely depends on the detonators, it is of the utmost importance that not only the proper strength of detonator be used, but also that the detonators be of a good quality. A common source of annoyance and danger in using inferior or under-strength detonators is miss-fires, which should be avoided at all hazards. Miss-fires, however, may occur with the best quality of detonators if they are not carefully handled and stored. Detonators should be kept in a dry place, and under no circumstances should they be placed in sawdust, as is sometimes done, for they absorb the moisture from it and soon become useless. They should also be bought in such quantities

AUTHORITY TO HAVE CHARGE OF DETONATORS.

_____ 191
 _____ Colliery
 _____ Seam or District
 Authority to _____
 employed as _____
 to have charge of detonators.

Received this _____
 day of _____

191 the counterpart of this

Certificate.

Signed _____

Occupation _____

Initials of Manager _____

COAL MINES REGULATION ACT, 1911.
 EXPLOSIVES IN COAL MINES ORDER
 OF 21ST MAY 1912.

_____ Colliery.
 _____ Seam or District, No. _____ Pit.
 _____ 191

_____ employed as
 _____ you are hereby appointed to
 have charge of the detonators for use at this
 colliery in accordance with the Explosives in
 Coal Mines Order of 21st May 1912.

Signed _____
 Manager.

I hereby accept the above authority to have
 charge of detonators at this colliery.

Signed _____

Section 1 (e) of the Explosives in Coal Mines Order of
 21st May 1912 stipulates that in every mine—

- (i.) Detonators shall be under the control of the manager of the mine, or some person or persons specially appointed in writing by the manager for the purpose, and shall be issued only to shot-firers or to officials specially authorised in writing by the manager.
- (ii.) Shot-firers and other authorised persons shall keep all detonators issued to them until about to be used for the charging of a shot-hole in a suitable case or box, securely locked, separate from any other explosive.

as will suffice for short periods, as they soon deteriorate if kept in store. In taking them into the mine, detonators ought to be carried in a securely locked case or box separate from the other explosives.*

No one is allowed to take detonators into the mine unless a duly authorised person, and the detonators must be kept in a securely locked case separate from other explosives. On p. 112 is given a copy of an authority form issued to persons authorising them to carry detonators.

Fixing the Detonator.—To fire a shot, a piece of fuse, of sufficient length, is taken, cut clean, and inserted into the open end of the detonator, which is then inserted into the cartridge of explosive, taking care to have the detonator so placed that the explosive covers it. After placing the detonator in position, it should be securely tied to the cartridge with a piece of string. A cartridge with a detonator so attached is called a 'primer.' When placing the charge in the shot-hole the detonator end of the cartridge should be towards the mouth of the hole, one detonator only being used for each charge.

Precautions against Fire-damp Explosions.—In order to avoid the danger of fire-damp explosions, arising from shot-firing, in fiery mines, strict observance should be paid to the following rules with regard to the detonator :—

- (1) Explosives should always be exploded by a sufficiently powerful detonator. Newly introduced explosives should be tested first to ascertain the strength of the detonator required.
- (2) The fulminating portion of the detonator must be properly enclosed, for only such caps can be depended on as do not suffer from leakage.
- (3) Detonators should be tested to see if they are in good condition.
- (4) Wherever possible, detonators should be specially re-dried before being used.
- (5) If explosion is to take place in wet or damp ground, the point of junction with the friction fuse should be well protected by some waterproof covering.†

Friction-detonators, *i.e.* detonators which are fixed to the fuse when manufactured, are safer during transport, less liable to be jarred in tamping, and safer to handle.

Firing the Charge.—Except where blasting is done by electricity, the charge is fired either by Germans, squibs, or safety fuses, the latter being most largely employed in coal-mining. Germans or squibs consist of small cylinders of cardboard or stiff paper filled with gunpowder, and are inserted into the shot-hole. To the outer end a piece of cotton wick, saturated with oil, is attached. When the charge is ready for firing, the wick is lighted.

Safety Fuses.—Many safety fuses have been devised. They

* See Explosives in Coal Mines Order, 21st May 1912 (1 (e) (ii.)).

† *Trans. Fed. Inst. Min. Engs.*, vol. x. pp. 550-557.

mostly consist of a fine column or central core of gunpowder, surrounded by flax, cotton, or similar materials. Taped fuses are protected by an external varnished coating, and are adapted for use in wet ground. In using such a fuse, the charge to be fired must be enclosed in a cartridge, into one end of which the fuse is introduced. The guttapercha fuse is surrounded by a coating of that material, and has also an outside coating of waterproof varnished cloth, so as to preserve the guttapercha, as the latter becomes very brittle when exposed to air. Fuses should be kept or stored in a dry place, so that the core of powder may not be affected by damp, and care should be taken that they do not come in contact with greasy or oily matter, as this rapidly penetrates the outer covering to the gunpowder and prevents the proper burning of the fuse. Ordinary fuse burns at the rate of about 3 ft. per minute.

The disadvantages of using ordinary safety fuses are: (a) Uncertainty of burning speed of the fuse; (b) danger of miss-fires through defective fuse; (c) dangers of shots hanging fire; (d) ignition of explosive gases from 'spit' of safety fuses; (e) ignition of explosive gases from burning fuses; (f) dense smoke given off from the burning fuses.

Firing Shots by Electricity.—In many collieries, especially those in which safety lamps are used, or which are dry and dusty, it has now become the custom to fire the shots by electricity. There can be no doubt but that this system of firing shots, when properly carried out, is very much safer than blasting with the ordinary fuse and detonator, as it allows the workmen to retire to a place of safety before the charge is exploded, and there is much less danger from 'hang-fire' or 'miss-fire' shots, or from the premature explosion of the charge.

In all coal mines which are dry and dusty, or in which inflammable gas has been found within the previous three months in such quantity as to be indicative of danger, all shots must now be fired by an efficient electrical apparatus. Where permitted explosives are required to be used, a competent person must be appointed—in writing—as shot-firer (Explosives in Coal Mines Order, 21st May 1912). On p. 115 is given a copy of a form usually employed for appointing shot-firers.

Electric Fuses.—In exploding charges of explosives by electricity, fuses, or detonators with wire attachment, are used. These fuses are generally of two kinds, viz. high- and low-tension fuses. High-tension fuses have their terminal wires bridged by a chemical paste or priming powder of relatively high electrical resistance. When a current is sent along the cables "the electrical energy at the fuse (wire) terminals is (owing to the insufficient conductivity of the bridging composition) converted into heat energy; the heat cannot dissipate with sufficient rapidity, therefore the temperature rises to the point of ignition of the bridge or priming. The latter bursts

AUTHORITY TO FIRE SHOTS.

COAL MINES REGULATION ACT, 1911.
EXPLOSIVES IN COAL MINES ORDER
OF 21ST MAY 1912.

_____ 191

_____ Colliery

_____ Seam or District

Authority to _____

employed as _____

to fire shots.

Received this _____

day of _____

191 the counterpart of this

Certificate.

Signed _____

Occupation _____

Initials of Manager _____

_____ Colliery.
_____ Seam or District, No. _____ Pit.
_____ 191
_____ employed as

_____ you are hereby appointed to
fire shots at this colliery, in accordance with
the Explosives in Coal Mines Order of 21st
May 1912.

Signed _____
Manager.

I hereby accept the above authority to fire
shots at this colliery.

Signed _____

Section 1 (e) of the Explosives in Coal Mines Order of
21st May 1912 stipulates that in every mine—

- (i.) Detonators shall be under the control of the
manager of the mine, or some person or per-
sons specially appointed in writing by the
manager for the purpose, and shall be issued
only to shot-firers or to officials specially
authorised in writing by the manager.
- (ii.) Shot-firers and other authorised persons shall
keep all detonators issued to them until
about to be used for the charging of a shot-
hole in a suitable case or box, securely locked
separate from any other explosive.

into flame and in turn fires the explosive compound at the end of the copper cap"; or, simply, the explosion is caused by the electric current heating the priming compound to ignition point. In the low-tension fuse the terminal wires are connected or 'bridged' by a short bridge of fine iridio-platinum wire (the wire is an alloy of 80 per cent. platinum and 20 per cent. iridium). The current in passing through this bridge raises its temperature sufficiently high (due to the same cause as in the high-tension fuse) to ignite the priming; the priming in turn, as above, fires the fulminate of mercury compound. The use of low-tension fuses has now become almost universal, as they are more trustworthy and safer; they can

be more easily tested, and are more certain in action than high-tension fuses. High-tension fuses are better adapted for firing a number of shots simultaneously, and they have certain other minor advantages; but in coal-mining it has been found that low-tension fuses are generally more satisfactory, although costing a little more. In the Dortmund district of Germany, fuses called 'Spaltglühzündor,' occupying an intermediate position between high- and low-tension fuses, are used for firing single shots. They are essentially low-tension igniters, but the platinum bridge is replaced by a finely divided conducting substance, such as graphite or coal-dust, mixed with the igniting substance, and the resistance opposed to the current is far greater than that of a small platinum wire, being generally about 5000 ohms. The quantity of current required to ignite the few particles of dust between the wire terminals is very slight, not exceeding at most $\frac{1}{100}$ th of an ampère. Such igniters are said to combine the advantages of both the high- and low-tension fuses without their disadvantages.

Exploders.—For igniting or exploding the fuse, small electric machines or exploders are used, that most largely employed being known as a magneto-exploder. It consists essentially of an armature, revolving between the poles of a set of permanent electromagnets of hardened steel. The armature, which is wound to a high resistance, is made to revolve rapidly by means of a rotary crank connected to geared wheels in contact with the armature spindle. By this means an electric current of high potential is generated sufficient to explode the fuse. A fluidic or dry battery and secondary battery exploders are also used for the electric firing of shots in mines.

Firing Cables or Conducting Wires.—In order to allow a safe distance between the blasting charge and the shot-firer, a suitable length of cable, for conveying the current, must be employed to connect the exploder to the fuses. The length required will depend upon the nature of the blasting, *i.e.* whether it is in rock or coal, and the place where the shots have to be fired, but the minimum length should not be less than about 60 ft. for coal, and a longer length for stone-work in narrow drifts. The most suitable length, however, to be used in every colliery cannot be arbitrarily stated, but must be fixed from what is found to be best by practical experience.

Simultaneous Firing.—When blasting with electricity in shaft sinking or stone drifts, the shots are ignited simultaneously in order to obtain the maximum rending effect, although it is questionable if this effect is always got from such a method of blasting. Some persons, indeed, hold that independent firing gives much better results, for the reason that if all the shots are ignited at once it cannot be expected that one shot will help the other; but if the centre charges, say in a stone drift or at the bottom of a sinking pit, are fired first, so as to loosen the middle portion, the side charges

should then operate under the most favourable conditions. To enable shots to be fired independently with electric blasting, a system has been brought into use by the adoption of a combination of electric ignition and tape-fuse. By this system a retarding action is got by inserting a piece of tape-fuse between the electric igniter and the detonator. For simultaneous firing two systems of connecting the fuses to the exploder are usually employed, known as the series and parallel systems. In the series system the line and fuse wires are coupled consecutively, one wire of the fuse being connected direct to the cable, the other wire connecting the first shot to the second, the second to the third, and so on until all the charges are joined up, after which the remaining wire is coupled up to the second cable. Low-tension fuses are generally fired by this system.

With the parallel system the two firing cables are connected to the last charge, forming a parallel line, and then the wires of the other fuses are coupled up to them alternately.

Where electricity is not employed for blasting, simultaneous firing of a number of shots can be carried out by using the Bickford patent volley-firer in conjunction with the same maker's instantaneous fuse.* This appliance consists of a little instrument devised so as to contain an ordinary safety fuse at one end and at the other a set of instantaneous fuses, the set varying according to the number of charges to be blasted. Between the end of the safety fuse and the ends of the instantaneous fuses is inserted an explosive disc, the action of which is such that, on the communication of fire from the safety fuse, the whole of the instantaneous fuses are immediately ignited, the latter burning at the rate of about 100 ft. per second, giving practically the same result as with electric firing. Any number of shots up to sixteen can be fired simultaneously by this apparatus.

Blown-out Shots.—These may be a source of great danger in mines which are dry and dusty and where fire-damp is given off. They ought to be carefully guarded against in blasting. Blown-out shots are brought about chiefly by (a) insufficiently and badly tamped holes; (b) an insufficient charge of explosive for the work to be done; (c) the shot-holes being drilled beyond the line of holing or kirving. The charge should be well, but not excessively, tamped with good surface clay or fireclay, free from stones or hard nodules, the tamping being firmly rammed back with a wooden stemmer. The hole should not be drilled beyond a point 6 to 8 in. from the back of the holing, and in stone-work especially, where the holing has often to be blasted out, the holes should be drilled at a suitable angle and length according to the kind of explosive used.

Position of Shot-holes.—The most suitable position for the charge depends upon various circumstances, a proper knowledge of which can only be obtained by actual practical experience. All 'joints,'

* The Bickford patent igniter is now prohibited in mines where permitted explosives are required to be used.

'backs,' 'lypes,' 'partings,' etc., must be carefully avoided, and the position of the hole for the charge so placed that the resistance in every direction may be as nearly equal as possible from the expected plane of fracture.

As an explosion takes effect along the line of least resistance, if there are any joints or cracks near the hole, they will determine the direction of fracture, and the charge will have comparatively little effect in any other direction. In the case of a sump-hole, for instance, in a sinking pit, the line of least resistance will be the shot-hole itself, and in such a case a heavier charge of gunpowder than ordinary must be used, and it will have to be well stemmed, or a strong explosive occupying little bulk must be employed, such as dynamite.

Cost of Blasting.—This will vary greatly according to the kind of explosive used, the system of firing adopted, and the hardness of the coal or strata the shots are fired in. The cost for blasting will vary from about 0·35d. to 1d. per ton, or an average of about 0·7d. per ton in coal. Regarding the cost of firing shots in gaseous mines—exclusive of explosive and labour—by different systems, *i.e.* firing by low- and high-tension electric safety fuses, and by the Bickford patent lighters and safety fuses *—Mr Frank W. T. Brain, in his evidence before the Departmental Committee on the use of electricity, gave some figures which are instructive, and are here reproduced :—

I. *Cost of firing 1000 Shots when using High-Tension Electric Detonators.*

1000 4 ft. wires, electric high-tension, No. 6 detonators, complete,	£5 0 0
50 yds. firing cable, costing 10s., used, say, for 2000 shots,	0 5 0
Magneto exploder, costing 35s., plus repairs 10s., used, say, for 30 shots per day, two years,	0 2 6
Total,	<u>£5 7 6</u>

Cost per shot = 1·29d.

II. *Cost of firing 1000 Shots when using Low-Tension Electric Detonators.*

1000 4 ft. wires, electric low-tension, No. 6 detonators, complete,	£5 16 0
50 yds. firing cable, costing 10s., used, say, for 2000 shots,	0 5 0
Magneto exploder, costing 35s., plus repairs 10s., used, say, for 30 shots per day, two years,	0 2 6
Total,	<u>£6 3 6</u>

Cost per shot = 1·48d.

* See Report of Committee, p. 88.

III. *Cost of firing 1000 Shots when using Bickford's Patent Safety Fuse and Igniters.*

4000 ft. Bickford's fuse,	.	.	.	.	.	£4	3	4
1000 Bickford's igniters,	.	.	.	.	.	3	2	6
1000 No. 6 detonators,	.	.	.	.	.	1	15	0
Total,						£9	0	10

Cost per shot = 2·17d.

From these figures it will be seen that the high-tension system is 0·19d. per shot, or approximately 13 per cent. cheaper than the low-tension system, and firing by the Bickford fuse and igniters shows a difference of 0·88d. and 0·69d. per shot respectively for high- and low-tension fuses, or a difference in favour of electricity of 68 per cent. and 46 per cent. respectively, for high- and low-tension fuses. It must be distinctly noticed that these figures are a mere comparison of the cost of material alone.

On the whole, firing by electricity is cheaper. Regarding the total cost of blasting, *i.e.* including explosives, fuses, and labour, it will vary greatly according to the kind of explosive used, the system of firing adopted, and whether the colliery is a non-fiery one or a fiery, dry, and dusty one. Naturally in the latter class of mine the cost will be somewhat higher than for non-fiery mines where open lights are used. The average cost for blasting with gunpowder is about 0·6d. per ton, and for safety explosives 0·9d. per ton of coal got, or a difference of 0·3d. per ton in favour of gunpowder, so that the cost for blasting is increased about 50 per cent. when safety explosives are used. With gunpowder the percentage of round coal got, in a number of experiments, averaged 62·2 per cent., and for safety explosives 62·0 per cent., so that so far as this is concerned there is not much difference between gunpowder and some of the safety explosives. In blasting rock there is, however, an estimated gain of 25 per cent. in using the latter.*

It may be taken for granted that many of the safety or permitted explosives are 50 to 100 per cent. dearer than blasting powder, but of course, on the other hand, a much smaller charge of such explosive will be required to do the work than if gunpowder was used.

* Paper by Henry Hall, H.M.I.M.

CHAPTER V.

MECHANICAL WEDGES, ROCK DRILLS.

Mechanical Wedges in Coal-mining.—In underground excavations the coal seam or rock can very rarely be removed by the aid of picks alone, unless in very soft strata. In ordinary seams wedges are used to assist in bringing down the coal after it has been 'holed'—the commonest form employed being known as the 'feather-shaped' wedge.* This wedge is also an adjunct to blasting in many mines; the coal being first loosened by explosives and afterwards wedged down.

Elliott Multiple Wedge.—In mines where blasting is prohibited, some mechanical method must be adopted for bringing down the coal or rock without the aid of explosives. The Elliott wedge is designed for such a purpose, and may be said to be an adaptation of the old plug and feather. The construction and use of the wedge will be understood from figs. 110, 111. To use the wedge a hole must first be bored out deep enough to hold the wedges. Into this hole are then inserted two portions of the wedge *a a*, tapered in front and increasing in thickness towards the further end. These pieces are constructed with the front portion turned back so as to grip the hole and prevent them from being driven out of position. Two other long-tapered wedges *b b* are now driven into the hole, and if these fail to bring down the coal, a third wedge, *d*, may be driven in between them and thus exert further pressure.

The advantages claimed for this wedge are that only a small hole requires to be bored, that the expansive force developed is great, the weight of wedges, etc., is small, and the first cost low. The wedges are made in two sizes: for holes $1\frac{3}{4}$ in. diameter, 2 ft. 6 in. long; and for holes 2 in. diameter, 3 ft. long.

Burnett's Roller Wedge.—In using the multiple wedge a great amount of power is necessarily lost in overcoming the friction of the parts sliding over each other, and the Burnett roller wedge has been designed to obviate this loss. In this appliance, the wedges, instead

* See *Elementary Coal-Mining*, third edition, p. 28, by G. L. Kerr. Chas. Griffin & Co. Ltd., London.

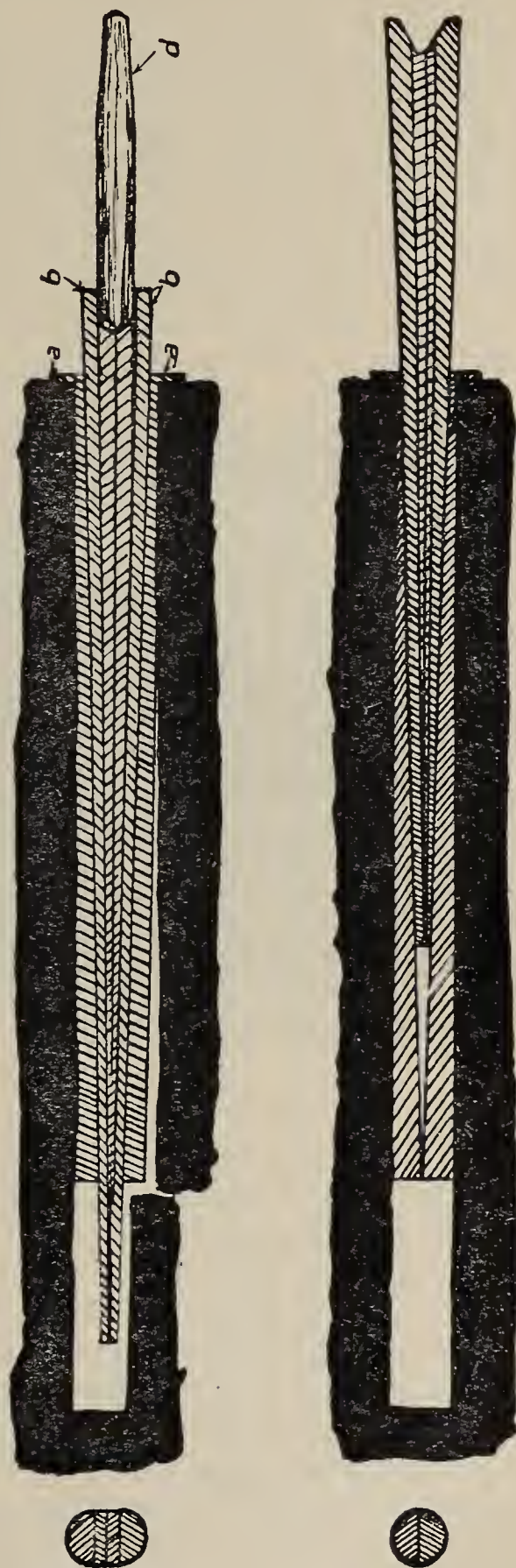
of being in sliding contact with the feathers or cheeks, are in rolling contact only; and as the rollers are arranged on each side of the wedge, it will be understood that the latter travels twice the distance covered by the rollers on the stationary cheeks or feathers — the centres of the rollers having a velocity ratio equal to only half that of the advancing wedge. The power is thus doubled and the friction greatly reduced.

Hydraulic Wedges. — Water at high pressure has been used as a means of blasting, but, except with the Tonge hydraulic cartridge, the success of hydraulic blasting has been questionable.

Tonge Hydraulic Cartridge. — This cartridge has been introduced into a number of collieries in the Lancashire coal-field, and has given such satisfactory results that in one or two mines it has entirely superseded blasting by explosives.

The cartridge consists of a cylinder of steel 20 in. long by 3 in. in diameter, and having eight small duplex rams fixed radially along it. By a suitable arrangement of passages, a communication is made between each of these rams whereby simultaneous action can be obtained. By an ingenious contrivance a greater traverse is obtained by these rams than the diameter of the cylinder. This traverse is essential for the complete forcing down of the coal. Thin liners are used to prevent the rams cutting into the coal.

The cartridge is operated by a hydraulic pump, to which it is connected by a pipe. The pump is of special design, is mounted on



FIGS. 110 and 111.—Elliott multiple wedge.

an adjustable stand, and fitted with water tank. The water required for the whole of the operation is about one and a half pints, but most of this returns to the tank at its completion, and can be used again. At the commencement of the operation the small handle is used, and when pressure has increased, an extension handle is slipped over it, and greater power is thus exerted. A pressure of 3 tons per square inch can be reached, and this represents a total pressure on the coal of over sixty tons. This is found adequate in ordinary seams, and the standard sizes are made for this duty. Special cartridges are made for extraordinary conditions.

After the coal has been undercut, either by hand or coal-cutter, a hole $3\frac{1}{4}$ in. diameter is drilled about 3 or 4 ft. deep into the coal

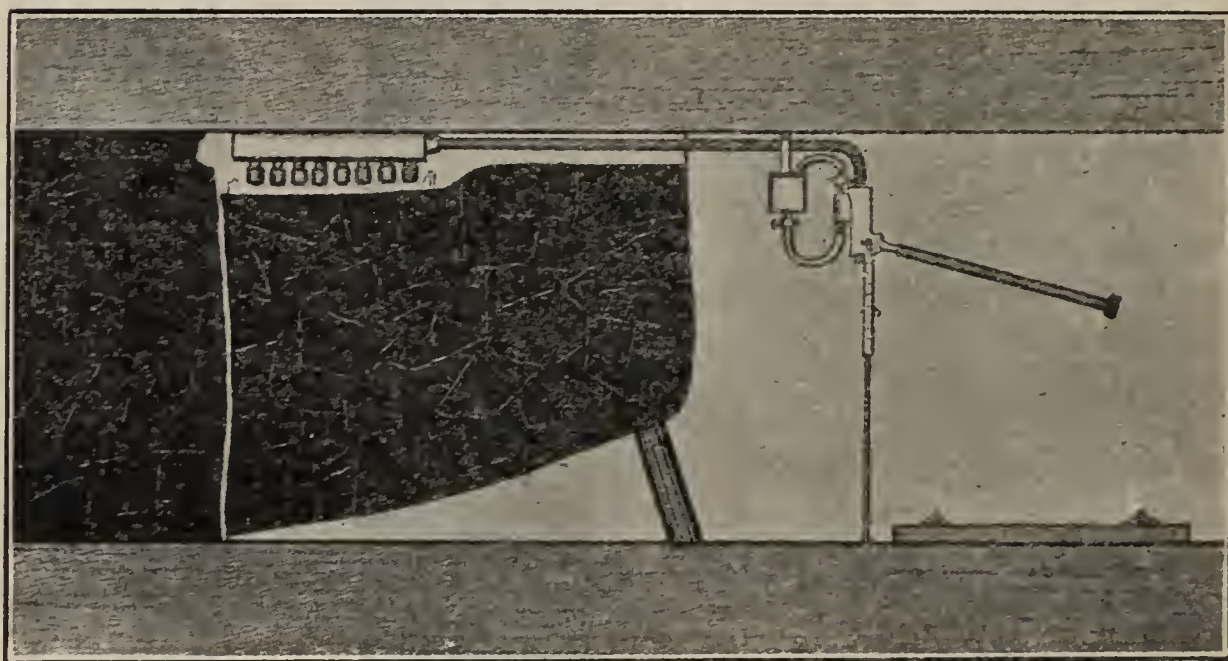


FIG. 112. —Tonge hydraulic cartridge, showing rams extended after use.

(slightly less than depth of holing). This is done by means of ordinary machine and spiral drills. The hole is put in parallel with the roof, and as near as possible along the parting to which the coal ordinarily comes off, or just below it. It is then cleared out. The cartridge, with one or more liners, and having pipe and pump attached, is pushed to the back of the hole, and the pump is left in position for the attachment thereto of the stand, which is adjusted to the required height. The water tank is filled, and hung on the pipe, the rubber suction pipe coupled, and the taps turned. The coal sprags are all left in tight. The pressure being fully on, the coal is heard to be rumbling and cracking. This is allowed to continue until the back portion of the coal is broken off, after which the sprags are slightly slackened. By a continuance of the pumping, the pressure is brought to bear at the front of the face, and continues to spread until the operation is completed, when the sprags are then withdrawn. The whole operation occupies about

ten minutes. This system naturally secures absolute immunity from explosion, miss-fires, poisonous fumes, etc., while it is claimed that the coal is not so shattered. It has been in use for years at the Atherton Collieries, 19,000 'thrusters' having been made in one seam in a year, producing 40,000 tons from a 3-ft. seam.

Rock Drills.—Rock drills may be divided into two classes, viz.—(1) hand drills, (2) machine drills. In the first class the work is performed by manual power alone, while in the second class other methods are employed, such as hydraulic pressure or compressed air. Drills either of the hand or machine type are of two classes: (a) rotary, and (b) percussive.

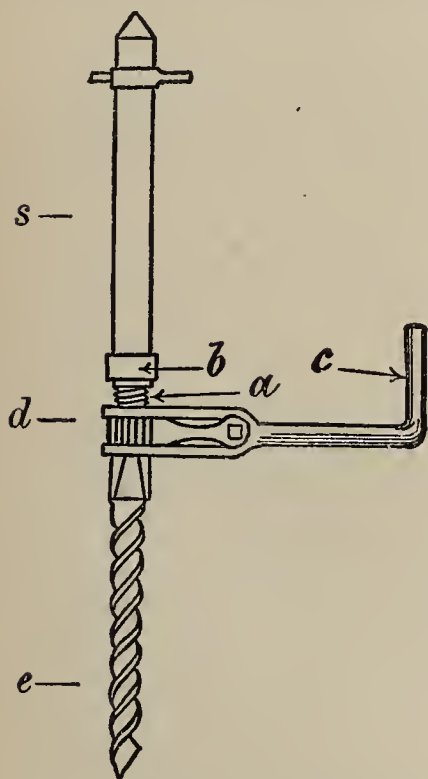


FIG. 113. —Hand drill.

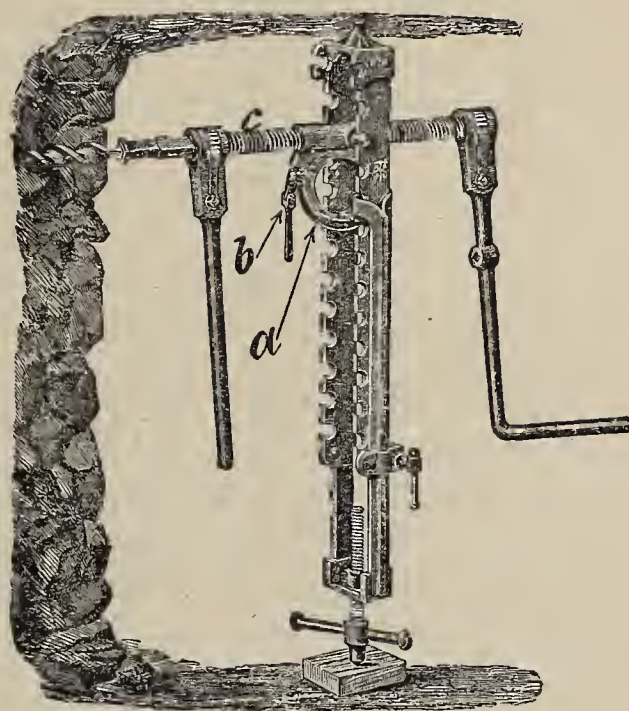


FIG. 114. —'Conqueror' machine.

Hand Drills.—The commonest type of this class is that known as the ordinary 'ratchet' boring machine, which is now so extensively used in all kinds of mining. Such drills are worked on the rotary principle. It consists of a screw-spindle *a* (fig. 113), terminating in a hollow square, into which the drill *e* can be inserted, and working through a screw-nut collar *b*, which is fixed to, or composes part of, a hollow sheath *s*, for the screw-spindle to work in. When a hole requires to be bored, a prop is set firmly up, and the drill fixed against it with the boring-bit *e* inserted into a small hole or cut made with the pick in coal or rock. The handle *c*, which works on the ratchet *d*, is then turned and bores out the hole by a grinding action. When the full length of the drill is out, it is again worked back into the hollow sheath and a longer drill substituted, the same process being repeated until the hole is bored to the required depth. These machines are very handy, and will penetrate very hard stone.

In the 'ratchet' machine a prop has to be used to support the machine, and where hard rock has to be bored it is often difficult to keep the drill in position. To overcome this difficulty and save loss of time many machines are now provided with adjustable stands. In the 'Conqueror' boring machine such a stand is provided, and the screwed nut and hollow sheath is done away with, the screw-spindle actuating the drill working through a long screw thimble *a* (fig. 114), which can be fitted on any part of the stand to suit the height of working and position of hole. An iron sole-piece is used on which the machine may stand, the height of which can be varied by the adjusting screw, which at the same time tightens it between the roof and floor. There is also an adjusting sliding piece *b* for regulating the position of the boring bit. The drill can be worked either with one or two handles, according to the hardness of the material and the power required.

Machine-power Drills.—This class of machine has proved of great service in certain kinds of work. These drills are not so largely employed in coal-mining as in tunnelling, metal-mining, and quarrying, but there are often cases, such as the driving of mines in hard rock, sinking shafts, etc., where they can be used with great advantage as auxiliaries in coal-mining. Possibly the fact that they require to be supplied with motive power such as steam or compressed air, the use of which involves considerable first cost in plant, and that skilled men are required to work them properly, will be against their extensive use in small coal-mines. There are, however, many collieries where compressed air is used for numerous purposes, amongst others for driving coal-cutting machines, and, in such circumstances, machine drills might be advantageously used, as for instance in drilling shot-holes in the ripping when it is composed of hard rock.

In order to obtain the greatest benefit from the use of rock drills in sinking or tunnelling, it is necessary that a given length be driven by each series of holes, and that all the holes of each series should be fired simultaneously, either by electricity or by a quick-burning fuse. The special point in favour of the use of rock drills in hard stone is that the same work is effected in about half the time that it could be done by ordinary methods. The following average weekly results have been obtained in sinking shafts by the aid of rock drills, the rate of wages being that prevailing in 1894.*

Shaft 18 ft. diameter.	Depth sunk Feet.	Cost per Yard.					
		Wages.			Explosives.		
		£	s.	d.	£	s.	d.
Very hard limestone with partings, ...	30	10	4	9	1	18	0
Coal measures, shales, and sandstones, ...	39	7	12	6	0	19	0

* *Trans. Inst. Min. Engs.*, vol. viii. pp. 18-19.

The depth sunk by each series of holes was :—

Hard limestone,	.	.	.	4½ ft. in 18 hours.
Hard sandstone,	.	.	.	5½ „ 18 „
Shale and sandstone,	.	.	.	6½ „ 16 „

The weight of stone lifted with each round of shots averaged, for hard stone 130 tons, and for moderately hard stone 150 tons.

At the Houldsworth Colliery, near Ayr, in driving a drift through green basalt there was a gain of 75 per cent. in speed of cutting by using machine drills, and a reduction in cost of £1, 17s. 7d. per fathom as compared with hand drilling, the details of which are shown in the following table :—

Method of Drilling.	Length of Drift Driven.	Time of Driving.	Average Length Driven per Week.	Cost per Fathom.
	Ft.	Weeks.	Ft.	£ s. d.
Hand, .	180	23	7·8	10 18 5·3
Machine, .	192	14	13·7	9 0 9·6

These figures are significant, as the cost of sinking would have been at least 25 to 30 per cent. higher if the ordinary methods had been adopted, while the rate of progress is very much greater, which is an important factor in all sinkings. Of course there is the additional cost of plant and consumption of fuel to be taken into account. In the first case cited, where four drills were used, an air-compressor having a cylinder 16 in. diameter by 24 in. stroke, with a 20 nominal horse-power boiler attached, was required. Each drill when working consumed about 30 lbs. of coal per hour, or taking the working time of the drills at four hours per day, the consumption of four drills would be about 480 lbs.

There are so many different kinds of machine drills in use that it would take up too much space to attempt to describe them all, and all that will here be attempted is to instance a few of the best known. Before doing so, it may be pointed out that the requirements of a good rock drill, as concisely stated by André* in his work on coal-mining, are as follows :—

It should be simple in construction and strong in every part.

It should consist of few parts, especially of few moving parts.

It should be as light in weight as is consistent with the first condition.

It should occupy but little space.

The striking part should be relatively heavy, and should strike the rock direct.

No other part than the piston should be exposed to violent shocks.

The piston should be capable of working with a variable length of stroke.

* *A Practical Treatise on Coal-Mining*, by G. G. André, p. 148.

Sudden removal of the resistance should not cause it any injury.

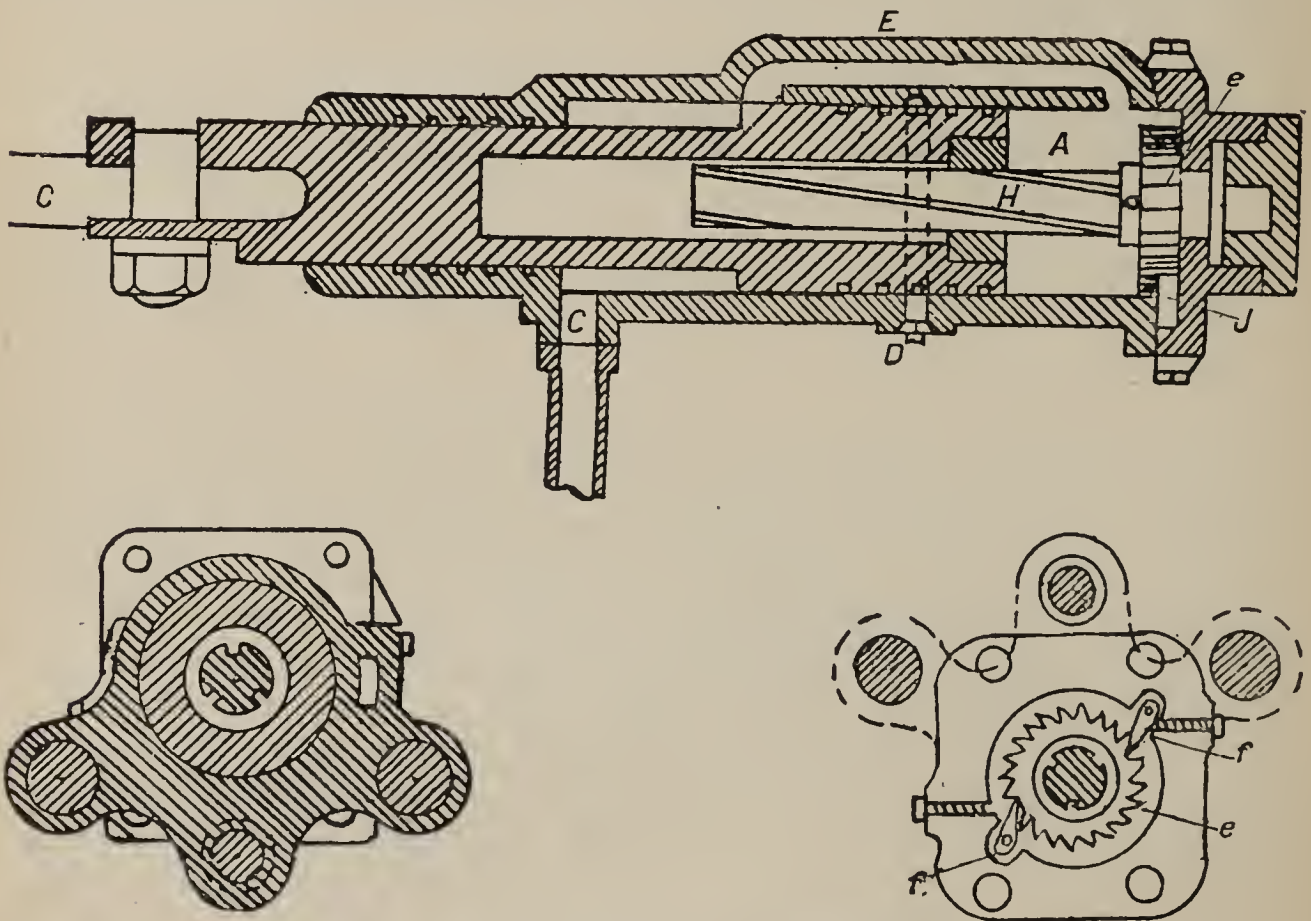
The rotary motion of the drill should take place automatically.

The feed, if automatic, should be regulated by the advance of the piston as the cutting advances.

The machine should be capable of working with a moderate degree of pressure.

It should be capable of being readily taken to pieces.

Darlington Drill.—This is one of the simplest and most effective machines in use. It consists essentially of two parts: the cylinder A (fig. 115), with its cover, and the piston with its rod, to which is attached the boring tool *c*. To give the latter a rotary motion,



FIGS. 115, 116, and 117.—Darlington drill.

there is a spiral or rifled bar *H*, having three grooves, and being fitted at its head with a ratchet-wheel *e*, recessed into the cover of the cylinder.

Two detents *f f* (fig. 117) are also recessed into the cover, and are made to fall into the teeth of the ratchet-wheel by spiral springs. This arrangement of the wheel and the detents allows the spiral bar *H* to turn freely in one direction, while it prevents it from turning in the contrary direction. The spiral bar drops into a long recess in the piston, which is fitted with a steel nut, made to accurately fit the grooves of the spiral. Hence the piston during its instroke is forced to turn upon the bar, but during its outstroke it turns the bar, the latter being free to move in the direction in which the straight outstroke of the piston tends to rotate it. Thus the piston, with the

boring tool, assumes a new position after each stroke. The total length of the Darlington drill is 3 ft., and the weight 100 lbs.

Adelaide Drill.—The Adelaide drill somewhat resembles the Darlington machine just described, but in it the air is used expansively, and it has only one moving part. The piston *c* (figs. 118, 119) works in a cylinder, having ports and passages so arranged that

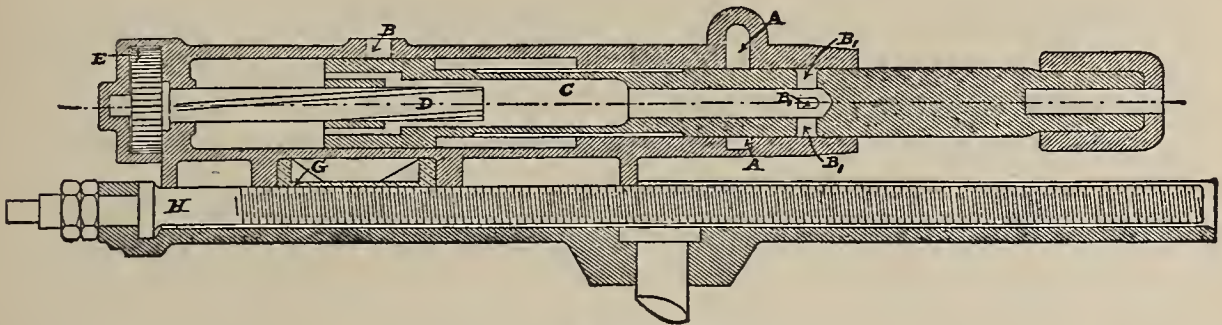


FIG. 118.—The Adelaide drill.

the air or steam is admitted and cut off automatically by the piston itself. Air enters through an annular port *A*, by which means the pressure is equalised on all sides of the piston-rod and unequal wear is avoided.*

The exhaust takes place through the port *B*. The piston itself is made to perform the action of a valve in the following manner: as soon as it reaches the port *B*, free communication is opened with the atmosphere and exhaust takes place, not only here, but also through the ports *B*₁, which have by this time passed outside the cylinder cover. The inlet aperture *A* being always in free connection with the air receiver, the pressure now acts on the small area at the front of the piston, and drives it backward, until this part is also brought into connection with the

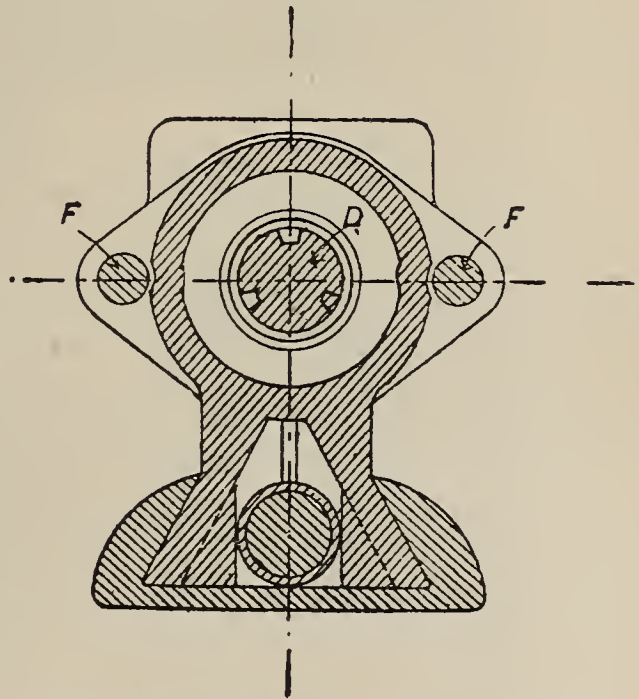


FIG. 119.—Section of Adelaide drill.

exhaust; at the same moment, the ports *B*₁ come opposite the inlet *A*, compressed air enters through the hollow rod *C*, passes into the back end of the piston, and drives the drill rapidly forward against the rock. Admission takes place during half the stroke, the air working expansively during the second half.

To rotate the drill there is a spiral or rifled bar *D*, having three grooves. It is fitted at its head with a ratchet-wheel *E*, which is

* *Journal Brit. Soc. Min. Students*, vol. xiii. pp. 70, 71.

recessed in the cover of the cylinder. Two detents or cams, also fixed into the cylinder cover, are forced by small springs to engage with the teeth on the wheel E. The feed is obtained through the rotation of the screw H in the nut G. As the admission both below and above the piston takes place some time before it arrives at the end of the stroke, a cushion is formed, and the piston is thus prevented from striking the covers.

Ingersoll Drill.—Fig. 120 shows the construction of the Ingersoll eclipse drill which is so largely used in American mines. Either steam or compressed air may be used as the motive power. The machine consists of the cylinder A, with the piston M, and the piston-rod B. Steam or air is admitted by a single spool valve C, and enters the valve chest at *o*, and when the piston has reached the end of its back stroke it passes round the valve to the passage N', and then enters the port P', and ultimately reaches the back end of the cylinder. When the valve is reversed, the air goes past and enters

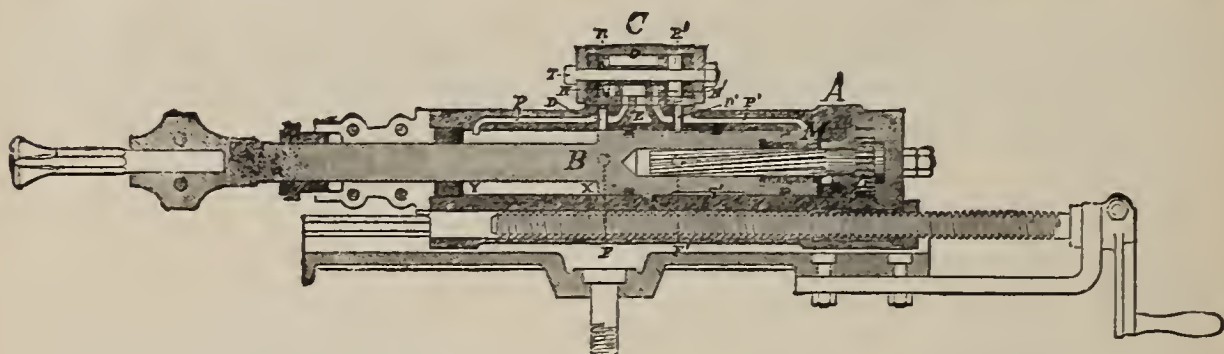


FIG. 120. —The Ingersoll drill.

the port P, to the front end of the cylinder. In the figure the port P is shown communicating with the exhaust E.

In the piston M are two recesses *s s*, which practically makes two short pistons out of one long one. There are two ports, F F', leading to the exhaust, and two small ports, D D', communicating cross-wise with the ends of the valve chest, *i.e.* the port D is connected with the end R' and D' with R. At each end of the cylinder are placed two elastic cushions, H H', to protect the cylinder covers from injury should the piston be suddenly relieved from resistance or the attendant fail to feed forward his machine as the drill advances. The drill is rotated by a rifled bar and ratchet-wheel. This drill can be had with cylinders varying in size from $1\frac{1}{4}$ in. diameter, which is used for light work, and holes 3 to 4 ft. deep and $1\frac{1}{4}$ in. diameter up to 5 in. diameter—the largest size made—which is used for holes 25 to 50 ft. deep and 3 in. to 6 in. diameter.

Hammer Drills.—This type of drill is now being extensively employed in mining work for sinking shafts and driving drifts in stone-work. They are specially suitable for small drifts and shafts, as they take up little space, and the operator can set them up and begin work in about five minutes. Holes can be drilled at any

angle, and can be placed so as to take every advantage due to breaks and fissures in the rock, and within certain limits, *i.e.* for holes $1\frac{1}{8}$ in. to $1\frac{7}{8}$ in. diameter by 4 to 5 ft. deep, accurate and rapid work can be done. The rotation is obtained by hand, and skilled drillers are not necessary, a good miner with a few days' experience being able to operate them quite successfully. They are easier kept in repair than the larger machine drills, and the consumption of free air per minute is very much less.

The majority of these hammer drills are operated by compressed air. Several drills of this class operated by electricity have been put upon the market, but, so far as the writer is aware, none of them have met with much success.

The three best-known types of hammer drills, operated by compressed air, used in British mines, are the 'Siskol,' 'Flottmann,' and 'Simplex-Hardy.'

Siskol Drill.—This machine is made in four sizes, weighing 21 lbs., 32 lbs., 45 lbs., and 60 lbs. respectively. The lightest type is recommended where holes have to be bored near the roof of a seam, and where a heavier machine would be disadvantageous. The second size is for general use, and the two heavier types for sinking purposes, or wherever it is needed to bore vertically downward holes, in which cases the operator has to bear little or none of the weight of the drill.

The action of this hammer drill* is effected as follows:—The compressed air enters the valve casing A (figs. 121, 122) through the holes L and L' on both sides of the valve B. To bring the piston G

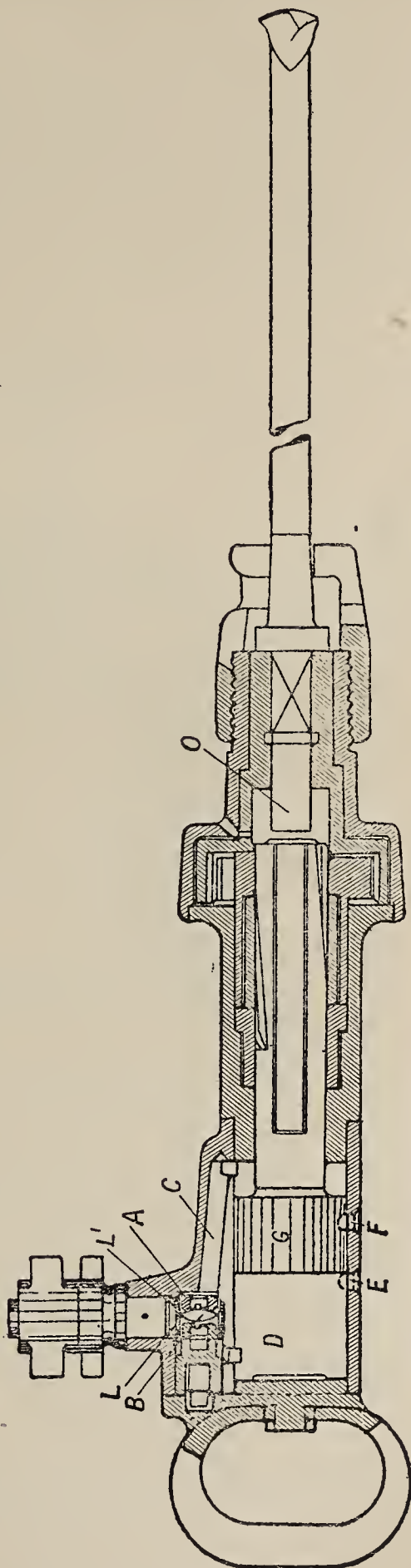


FIG. 121.—Siskol hammer drill.

* *The Colliery Guardian*, 12th November 1909, p. 978.

into the position shown on the drawing, the valve B has been driven to the right side, closing the channel C, the air having entered the cylinder through the port D and driven the piston into the position already shown, by means of which the blow on the borer O has been delivered. As soon as the piston on its forward stroke uncovers the exhaust holes E the air behind the piston immediately escapes, and at the same time, due to the forward stroke of the piston, a slight compression has taken place in channel C. Owing, therefore, to the escape of the compressed air through the exhaust holes E, and to the slight compression in channel C, a

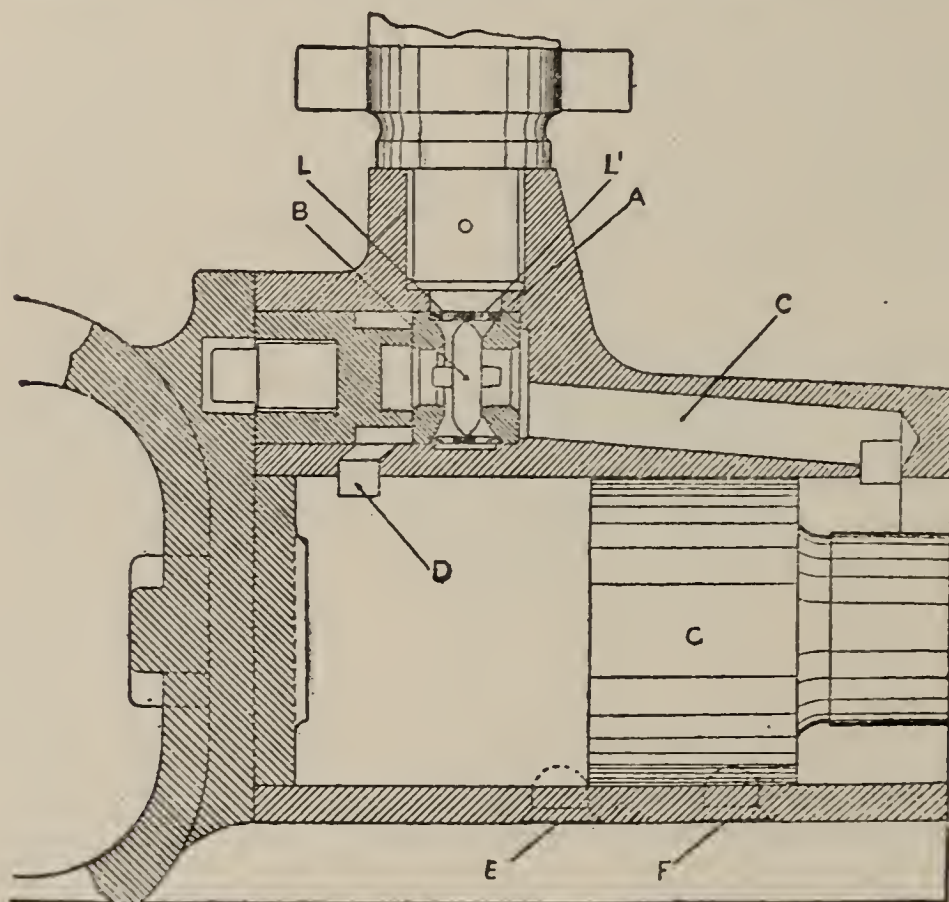


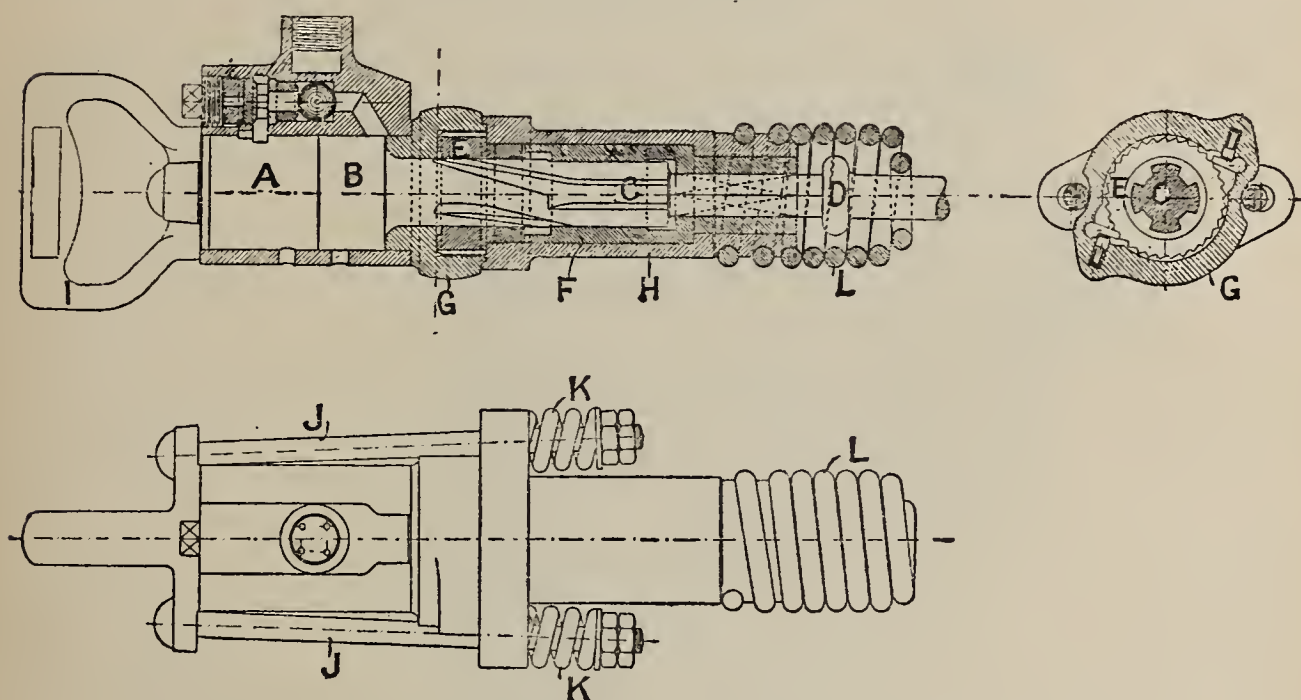
FIG. 122.—Enlarged section of Siskol hammer drill.

difference in pressure is brought about on the sides of the valve B, by which means the valve is thrown over to the left side, thus opening channel C, and thereby enabling the compressed air to get to the front of the piston, and so reversing the action of the drill. When the piston on its backward stroke uncovers exhaust holes F a difference in pressure on the sides of the valve is again brought about, due to the causes already explained.

The number of blows delivered with an air pressure of 60 lbs. or so per square inch is approximately 2000 per minute.

Flottmann Drill.—This hammer drill was introduced into British mines in 1906, and since then has been largely employed in stonework underground and in shaft sinking. Figs. 123, 124 show the construction of the drill. It consists of a small cylinder A, 3 to 3½

in. diameter, inside of which the piston B moves to and fro, striking with its outer and thinner part C the drill D at the rate of about 2000 blows per minute. The thin part C of the piston contains in its rear portion a series of twisted grooves, whilst in the front portion a series of straight grooves are cut. The ratchet-wheel E fits into the twisted grooves, in which twisted female grooves are cut. The straight grooved part of the piston slides inside the drill holder F, containing at the back a series of similar grooves, and at the front a shank for the drill. Pawls pivoted in the ratchet-case G engage the ratchet-wheel and allow it to revolve in one direction only. The drill holder is encased by the front end H, and the handle I serves to close the top part of the cylinder A, as also to handle the machine. The two tie rods J keep the parts in position,



FIGS. 123 and 124.—Improved type of Flottmann hammer drill.

the coil springs K preventing breakages if the piston does not strike the drill, as may happen when the drill is withdrawn from the bore-hole with the machine still running. When the piston is moving towards the handle the pawls prevent the ratchet-wheel from revolving, and the twist forces the piston to turn, which motion is transmitted through the drill holder to the drill. On the downward stroke of the piston the ratchet-wheel, turning free, allows a straight downward blow. The drill is held in the drill holder by means of the coil spring L, which is screwed on to the front end H, and thus forms a simple chuck.

The compressed air for operating the drill is admitted into the cylinder by means of a steel ball valve moving in a casing with about $\frac{1}{16}$ in. stroke backwards and forwards. The ball, which is the only moving part of the valve gear, is enclosed in a steel bush, the ends of which, by means of channels, communicate with the ends

of the cylinder. Side-ways into the bush holes are drilled, through which the compressed air passes. When the ball closes the port

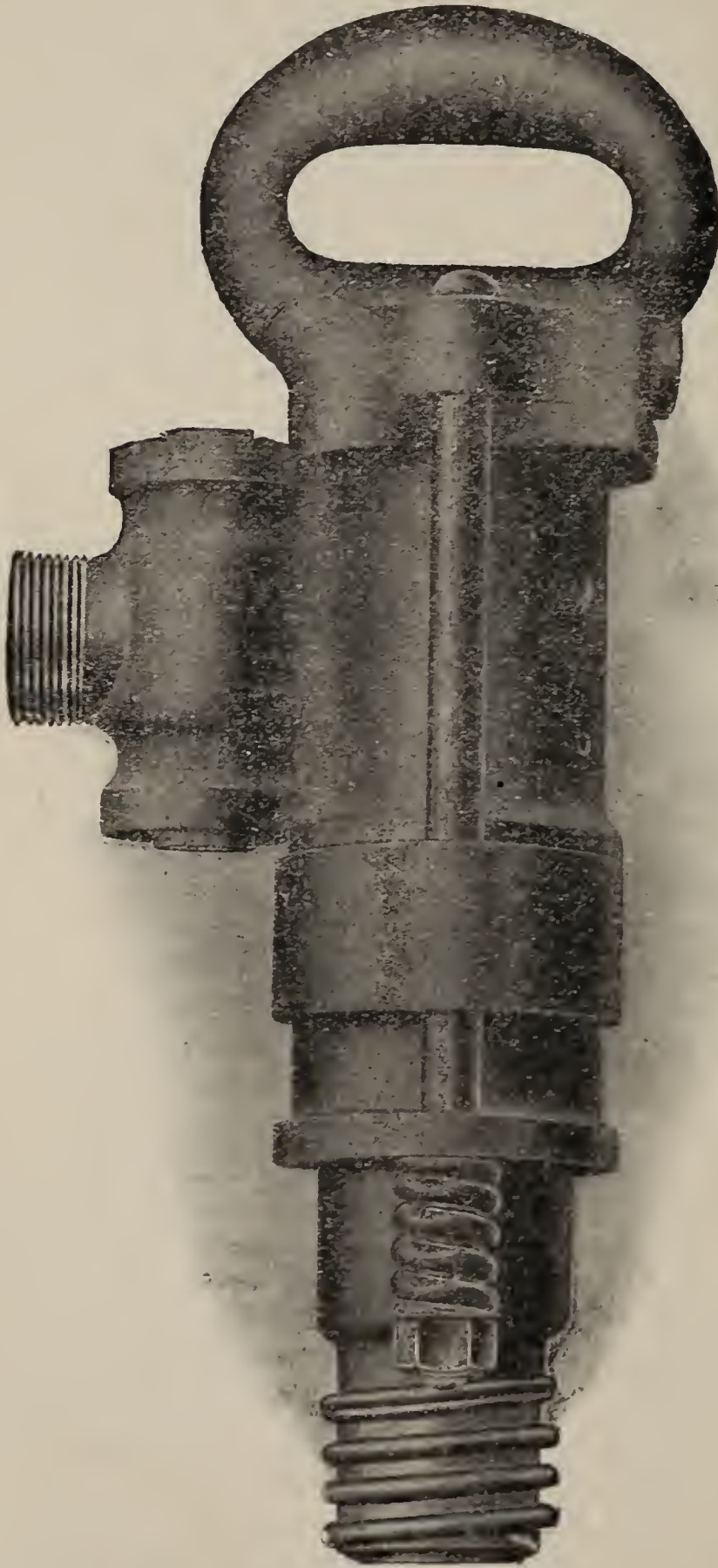


FIG. 125.—Hardy-Simplex hammer drill.

communicating with the back of the piston, the air passes on the other side of the ball into the cylinder and forces the piston backwards. After the piston has moved a certain distance in that

direction, the lower exhaust ports in the cylinder are opened and the air allowed to escape. This escape of air reduces the pressure on that side of the ball, and the greater pressure on the other side, throwing it over against the lower side, opens the part leading to the back of the piston. The piston in moving backwards compresses the air in the back part of the cylinder, and is then thrown forward to deliver its blow with great force on to the end of the drill, at the same time opening the exhaust port at the back of the piston, which action brings the ball back into its first position. The drill is made in several types, weighing from 23 to 40 lbs. complete, and suitable for boring holes from 2 to $3\frac{1}{2}$ in. diameter to a depth of $3\frac{1}{2}$ to 10 ft.

Hardy-Simplex Drill.—This hammer drill is also largely used in mines for boring holes in stone-work, and gives good results. The cylinder and valve box are forged together in one piece of special steel, bored, case-hardened, and then accurately ground. The valve and valve bush are manufactured from special alloy steels, and the valve moves with practically no shock, and is provided with efficient buffering devices. The valve is of about the same length as diameter, to ensure steadiness of working and immunity from any tendency to wedge itself or assume an oblique position.

The rotation of the drill is effected by means of a rifle bar and ratchet-wheel situated at the rear of the machine, in combination with a number of straight grooves cut in the piston or hammer, working through a steel guide nut situated in the chuck or drill holder. The drills are retained by means of a cap, which is provided with a round thread of great strength, screwing over the front of the drilling machine, and embracing the collar or the drill shank. The cap is slotted its full length, thereby enabling the machine to be at once disengaged from the drill steel should this for any reason become necessary. It is claimed that this cap is a far more rigid and convenient attachment than many of the ordinary spring devices used. To dismantle the cylinder and component parts it is only necessary to unscrew two nuts. The drill can bore holes up to 12 ft. in depth, and the most suitable air pressure is 40 to 75 lbs. per square inch. The same machine is suitable for solid, hollow, or spiral drills, as exhaust air only is used for blowing out the dust.

Drills of the hammer type have also been introduced to operate with electricity, but they are not so satisfactory as the air drills, and so far have not been much employed in Britain. The 'Locke' and 'Siemens and Halske' are drills operated by electric current.

CHAPTER VI.

COAL-CUTTING BY MACHINERY.

COAL-CUTTING by machinery was introduced practically into British mines over forty years ago, at a time when there was a great scarcity of workmen and wages were high in the mining districts. As long ago as 1869 coal-cutting machines, of the Winstanley type, were being operated at collieries in the Wigan district, and about the same time, or shortly thereafter, Messrs William Baird, Limited, the well-known Scottish coal and ironmasters, introduced a coal-cutting machine of their own design to undercut the coal at their Gartsherrie Collieries, near Coatbridge. Since that time mechanical cutters have been used to some extent in the different mining districts, but until comparatively recently coal-cutting by machines has had only a limited application in British mines. Within the last decade, however, there has been a great increase in the number of machines in our mines, and year by year coal-cutting by machinery will have a more widely extended application all over the coal-fields of Great Britain.

Some of the chief difficulties which have retarded the more extensive employment of coal-cutting machines in Britain have been badly designed machines, inferior workmanship, and the use of inferior material. This has to a great extent been largely remedied, and there are now some excellent machines at the disposal of colliery managers, suited to work under almost any ordinary conditions. Another difficulty which has had to be contended with has been the labour question, the workmen in many districts having a prejudice against the use of coal-cutting machines, on the ground that the introduction of such machinery will dispense with the need of hand labour. This phase is usually temporary, and is the result of misapprehension of the influence of machine mining on employment and wages. Where the men have had experience of machines they have found them altogether beneficial to their interests; and when there is a choice of working in machine-wrought or hand-wrought sections, they usually prefer the former, for they earn higher wages for less laborious and less irksome work. There has also been the difficulty of getting a sufficient supply of skilled men to operate and supervise the machines, and consequently in many cases incompetent men are

operating machines with unsatisfactory results. This difficulty has been greatly lessened with the increase of skilled operators.

The following table shows the number of coal-cutting machines at work in the various mining districts in the United Kingdom for 1910, and the different types of mechanical cutters used, as given in the Annual General Report and Statistics of Mines and Quarries (Part II., 1910).

NUMBER OF MACHINES, AND OF COLLIERIES WHERE USED, MOTIVE POWER EMPLOYED, AND QUANTITY OF MINERAL OBTAINED BY THEIR USE IN THE VARIOUS INSPECTION DISTRICTS DURING THE YEAR 1910.

District.	No. of Collieries where Machines are at Work.	Number of Machines.	Worked by		Coal and other Mineral obtained. Tons.
			Electricity.	Compressed Air.	
Scotland, ...	168	581	425	156	5,873,455
Newcastle, ...	31	286	35	251	1,387,077
Durham, ...	19	119	42	77	691,219
Yorkshire and North Midland, ...	99	481	235	246	4,991,033
Manchester and Ireland, ...	16	153	12	141	506,361
Liverpool and North Wales, ...	22	129	18	111	983,224
South Wales, ...	49	107	51	56	559,791
Midland and Southern, ...	28	103	55	48	886,641
Total in 1910, ...	432	1959	873	1086	15,878,801
Total in 1909, ...	420	1691	777	914	13,769,687

A large proportion of the machines employed are of the well-known disc type, which seem to be best suited for British mines. The following table shows the number of the various types of machines in use.

The quantity of mineral cut by electricity in 1910 was 8,963,460 tons, as compared with 7,490,131 tons in 1909; and by compressed air 6,915,341 tons, as against 6,279,556 tons.

There has been a large increase, both in the number of machines used and in the output of mineral, as compared with the year 1909.

Classification of Machines.—Coal-cutting machines may be divided into the following classes:—

- a. Heading machines, such as the Stanley heading machine.
- b. Disc machines, such as the Gillott & Copley machines.
- c. Chain machines, such as the Jeffrey machine.
- d. Bar machines, such as the Hurd machines.
- e. Percussive machines, such as the Ingersoll-Sergeant machines.

a. Heading Machines.—This class of machine has, up to the present, been used only to a very limited extent, the only machine

employed with success in British mines being the Stanley heading machine.

It consists of an iron frame carried on two wheels set one in advance of the other. On this frame is fixed an engine with two cylinders placed vertically, the pistons being connected to the engine shaft by two cranks. This engine shaft carries geared wheels at each end, and through these is geared to the principal cutting shaft,

MECHANICAL COAL-CUTTERS EMPLOYED IN THE UNITED KINGDOM IN 1910.

Kind of Machine.	Number in Use.								Total.
	District No. 1.	District No. 2.	District No. 3.	District No. 4.	District No. 5.	District No. 6.	District No. 7.	District No. 8.	
Driven by electri- city:—									
Disc, ...	269	21	20	144	3	8	10	29	504
Bar, ...	137	10	11	38	7	8	34	16	261
Chain, ...	19	4	11	52	2	2	7	10	107
Percussive, ...	...	...	...	1	...	...	...	...	1
Rotary heading, ...	...	...	...	...	...	...	...	...	...
Total, ...	425	35	42	235	12	18	51	55	873
Driven by compressed air:—									
Disc, ...	114	38	18	148	28	66	6	8	426
Bar, ...	...	...	11	24	3	7	24	3	72
Chain, ...	...	4	4	10	1	1	4	3	27
Percussive, ...	41	209	44	64	109	37	20	31	555
Rotary heading, ...	1	...	...	...	...	...	2	3	6
Total, ...	156	251	77	246	141	111	56	48	1086
Total in 1910, ...	581	286	119	481	153	129	107	103	1959
Total in 1909, ...	489	223	135	410	125	139	69	101	1691

which passes through the centre of the frame. On the end of this shaft a cross-head is fastened, to which are attached, at right angles, two cutter bars upon which are fixed the cutters. These cutters revolve in a circle, and cut out a circular core of coal about 5 ft. in diameter. The main shaft has a screw thread cut nearly its whole length, by which, aided by suitable gearing, the cutters are advanced. The cutter bars project about 3 ft. beyond the cross-head ; and this length controls the depth of each cut. The machine is ‘anchored’ to the sides and floor, by means of screwed arms, to maintain it in position, and to keep the cutters against the face. When a cut the

length of the arms has been made, and the coal removed, the cutting motion is put out of gear, and the advancing motion put into gear, by which the whole machine is propelled forward ready to start a new cut. The whole machine, which is worked by compressed air, weighs between three and four tons. It can cut usually between 12 to 15 ft. in a shift of eight hours, including the time taken to bring down the coal and move the machine forward. Compared with hand labour, the cost of cutting by this machine is high, but the great advantage of using it is the rapidity with which a field of coal can be opened out, the cutting rate being double that of hand labour. The type of machine just described can only be used in thick seams, *i.e.* seams above 5 ft. thick.

A double or duplex type of this machine is now made to enable a wider road to be cut to admit of two lines of rails being laid down

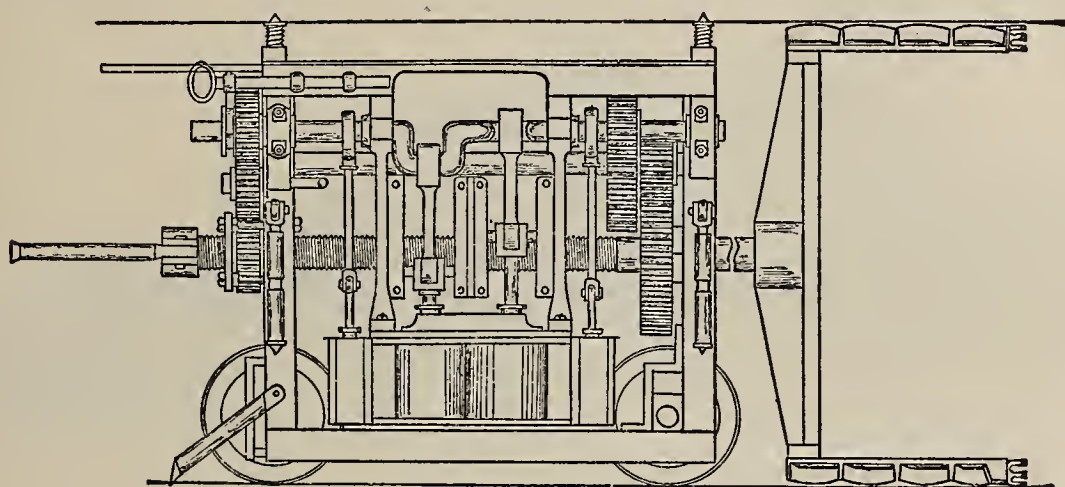


FIG. 126.—Stanley heading machine.

for endless rope haulage. It can also be applied to seams varying from $3\frac{1}{2}$ to 7 ft. thick, and the width cut from 8 to 12 ft. In the newer type the cylinders and driving gears are placed at the top instead of at the bottom of the machine, thus leaving a good clear space at the floor level, between the feeding screws, for the cut coal to be passed through.*

A machine of this type was employed for heading work at the Charity Colliery, Bedworth, and the average rate of cut per week was 93 ft., at a cost, including everything, of about 3s. 4d. per foot. Each machine made a cut 9 ft. wide and 5 ft. 4 in. high.

b. Disc Machines.—There are now a considerable number of different machines of this type used in British mines, but they are all constructed on practically the same principle, *viz.* that the under-cutting is done by means of a disc, 3 to 6 ft. diameter, on the periphery of which are fixed a number of picks or cutting tools, the disc working in the same way as a circular saw, but placed horizontally instead of vertically, and made to rotate by suitable gearing. These machines

* *Trans. Inst. Min. Engs.*, vol. xxx. p. 600 ; also vol. xxvi. p. 538.

are exclusively used for longwall working, where they can be set to under-cut a long length of face in one operation.

The Gillott & Copley is one of the best-known types of disc machine, and has been successfully used for many years. It is worked by compressed air, and has a frame about $5\frac{1}{2}$ ft. long by $2\frac{1}{2}$ ft. wide, which carries two cylinders 8 or 9 in. diameter, and about the same length of stroke. The cutter-wheel, which varies from 3 to 4 ft. in diameter, carries on its circumference about twenty-five steel teeth, and cuts to a depth of 3 to $3\frac{1}{2}$ ft., the height of cut not being more than 3 in. This machine works at a low speed, the cutter-wheel making only about five or six revolutions per minute. The machine runs along the face on rails, and is fitted with the usual wire rope and drum arrangement. The disadvantage of this machine is that it does not cut level with the floor, the portion left on having to be taken up by hand. It can, however, be arranged to cut level,

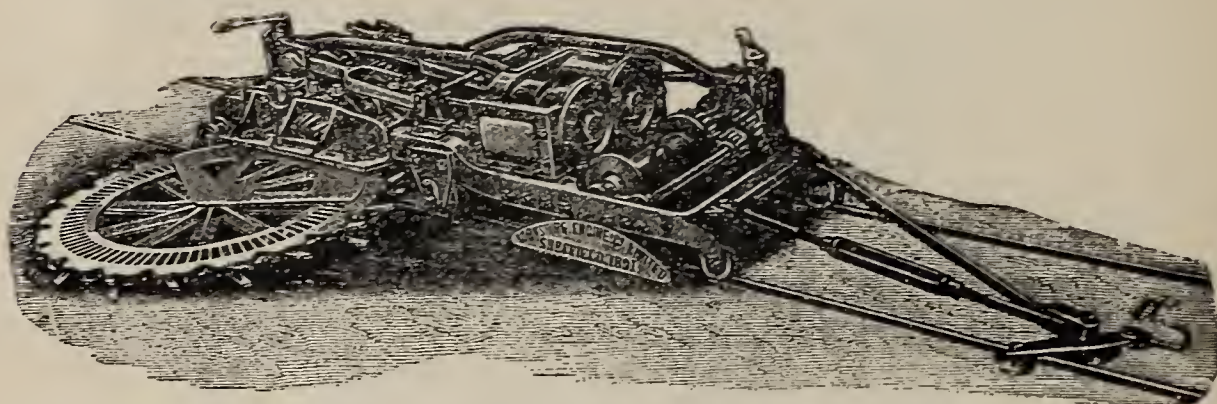


FIG. 127.—The Gillott & Copley coal cutter.

but it must be provided with a guard to keep the disc from taking the cuttings back into the holing.

Where the holing is done in the coal, this machine does very well, but is not so efficient when cutting in fireclay as some of the other disc machines. Where the under-cut only requires to be about 3 ft. or so, this machine gives good results under favourable conditions for working; where a deeper cut is required, some of the machines with larger discs, such as the Diamond machine, are generally used.

The Rigg & Meiklejohn machine is somewhat like the Gillott & Copley cutter, and holes the coal in the same way, the picks being fixed into a circular disc. It is made so that it can cut at the very bottom of the coal. The construction of this machine will be understood by reference to fig. 128. It is placed on a steel frame xy , carried on four wheels, and kept in adjustment by screws at each corner. The two air cylinders dd are bolted to the frame, these cylinders being about 8 in. diameter and 10 in. stroke. A jib b , bolted to the side of the frame, carries the disc a , with eight or ten snugs ee , on which the picks are bolted. A bevel pinion c , with twenty-seven teeth on the engine shaft, is geared into a circular rack f of sixty teeth on the disc. The disc makes about sixty-two revolutions

per minute, while the engine makes about 140 revolutions. The picks, 5 to 6 in. long, made of hard steel, are in sets of four, and shaped so that the whole thickness of holing, about 4 in., is fully occupied by the pick points. The disc revolves on a bearing, and is kept in position by a bottom plate, bolted with stud bolts to the jib. The machine is reversible, and cuts either way. The engines are reversed by turning the eccentric pulleys round on the shaft, no links being used. The machine, whilst cutting, is hauled along by gearing from the main driving shaft, on which there is bolted a split worm which works into a pinion on the under shaft. The under shaft extends from the engine sole-plate to the front of the machine in which the

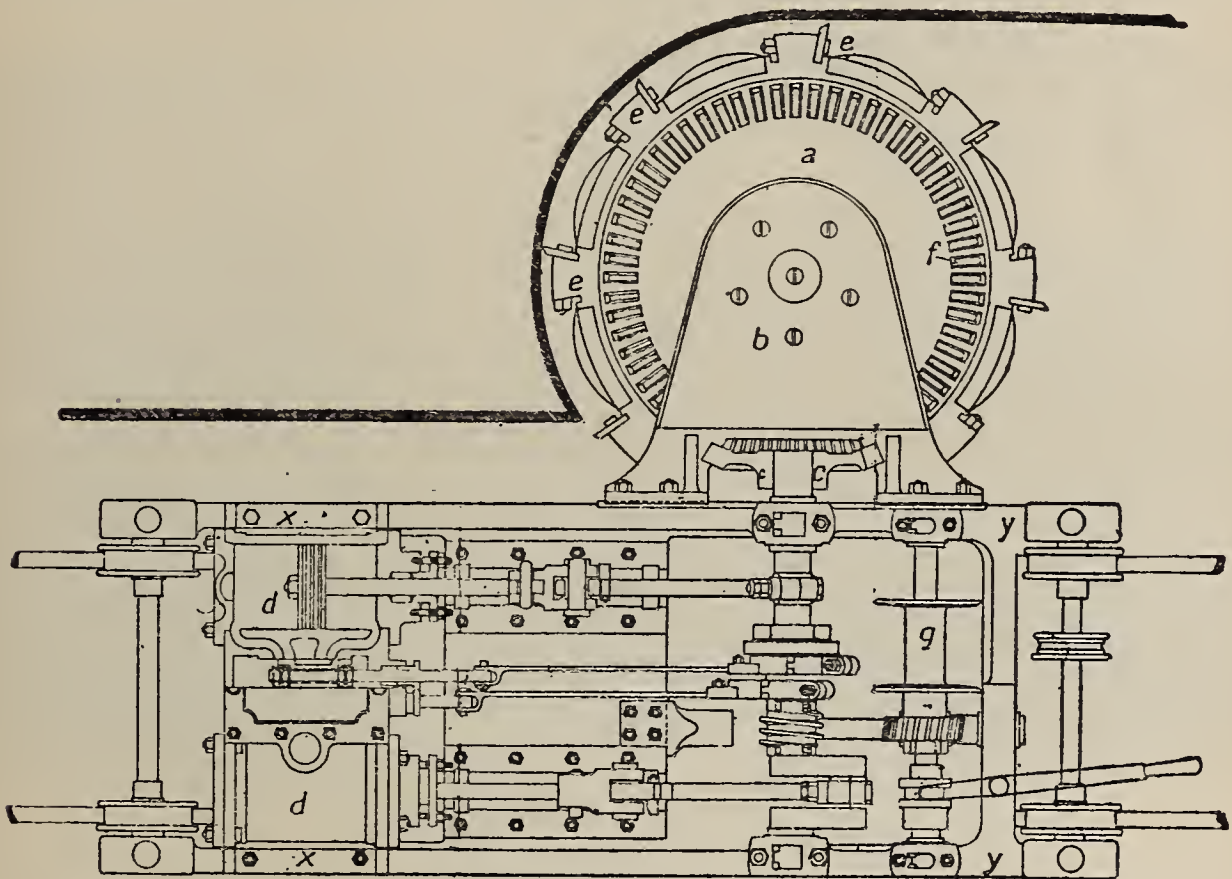


FIG. 128.—Rigg & Meiklejohn machine.

bearings are seated. A solid worm on the under shaft is geared into a pinion on the hauling shaft, on which the drum or chain wheel *g* is placed. A $\frac{3}{8}$ -in. chain is coiled once or twice round the drum and is made fast to a prop 60 ft. ahead. The depth of cut is about 3 ft. to 3 ft. 6 in., the height of the machine 22 in., and the weight about 18 cwt.

This is a very good machine for thin seams, as in cutting it makes its own floor. It is equally suitable for cutting in either a coal or fireclay holing, and has been extensively used with successful results at a large number of collieries for a good many years. The motive power used is compressed air, but it is now being constructed to work also with electricity.

Electric Power Machines.—The above two machines are types with

compressed air as the motive power, but by far the larger number of machines are now constructed to be operated with electric power. This system has many advantages, where the conditions are suitable, and is referred to further on.

Clarke & Steavenson Machine.—The Clarke & Steavenson machine (fig. 129) is constructed on much the same principle as the last two described, with the exception of some minor details. Unlike the Gillott & Copley, or Rigg & Meiklejohn machines, it has been constructed to work with either compressed air or electricity as the motive power, the larger number now being made to be driven electrically. Two types of this machine are at present in use, the standard type for cutting in seams 26 in. and over, and the low type

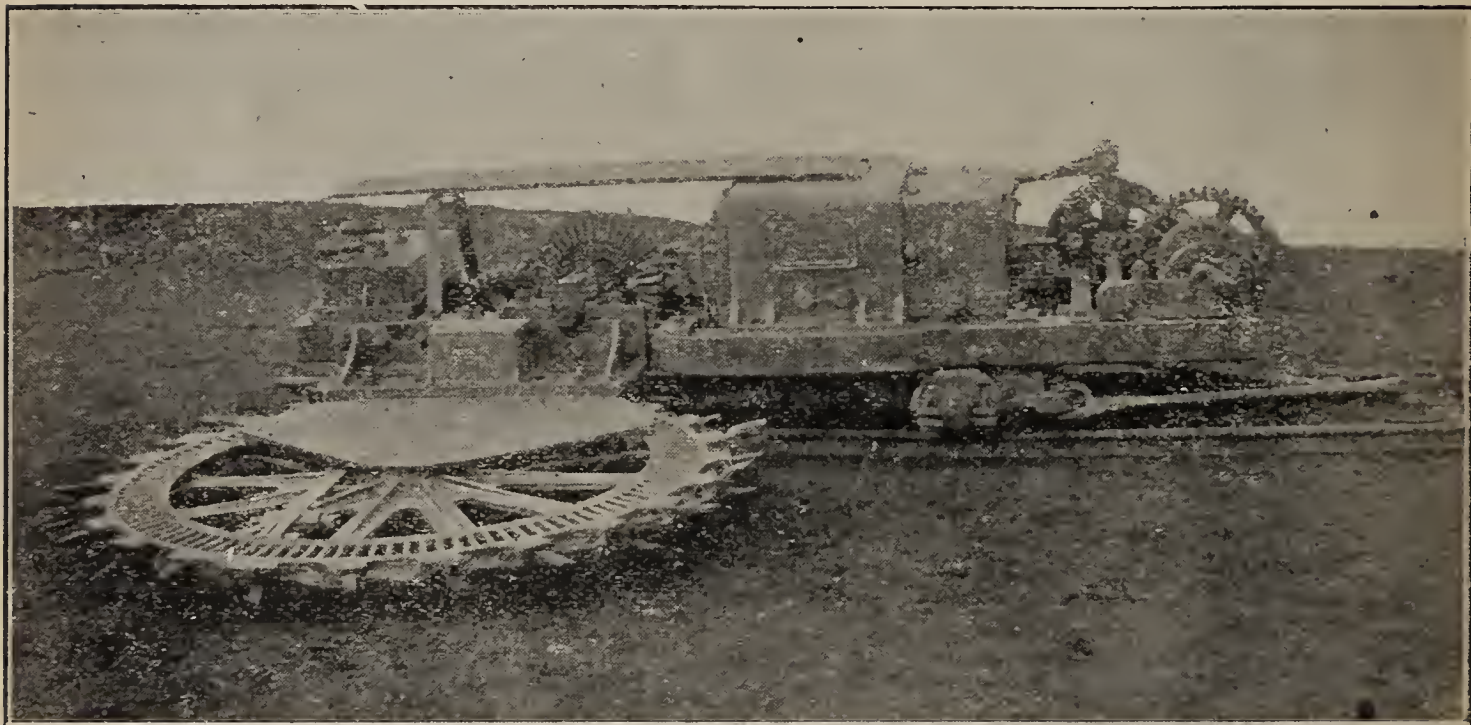


FIG. 129.—Clarke & Steavenson machine.

for cutting in seams under 26 in., the latter machines being capable of cutting in seams 20 in. in height. The Clarke & Steavenson machine, like those already described, is provided with a disc fitted with cutters which are alternately of the straight and forked type, or singles and doubles, and are secured to the disc by being fitted into boxes cast on its periphery and held in position by cotter bolts and nuts, or by split pins. The number of cutters or teeth in the cutting wheel is twenty, of which ten are straight, or singles, and ten forked, or doubles. The discs used are either 4 ft. or 6 ft. in diameter, according to the depth of the under-cut required, the former making an under-cut of 3 ft. 6 in., while the latter makes an under-cut 5 ft. 6 in. in depth.

The standard type is 2 ft. 2 in. in height, measuring from the top of the rail to the top of the casing, while in the special low type the height is 1 ft. 5 in. In the Rigg & Meiklejohn machine the

cutting wheel is placed almost at the front of the machine, but in the Clarke & Steavenson it is placed at the rear, which is a great advantage, as the material from the cut is delivered at a point close to the driver. As a consequence, if the machine is working beyond

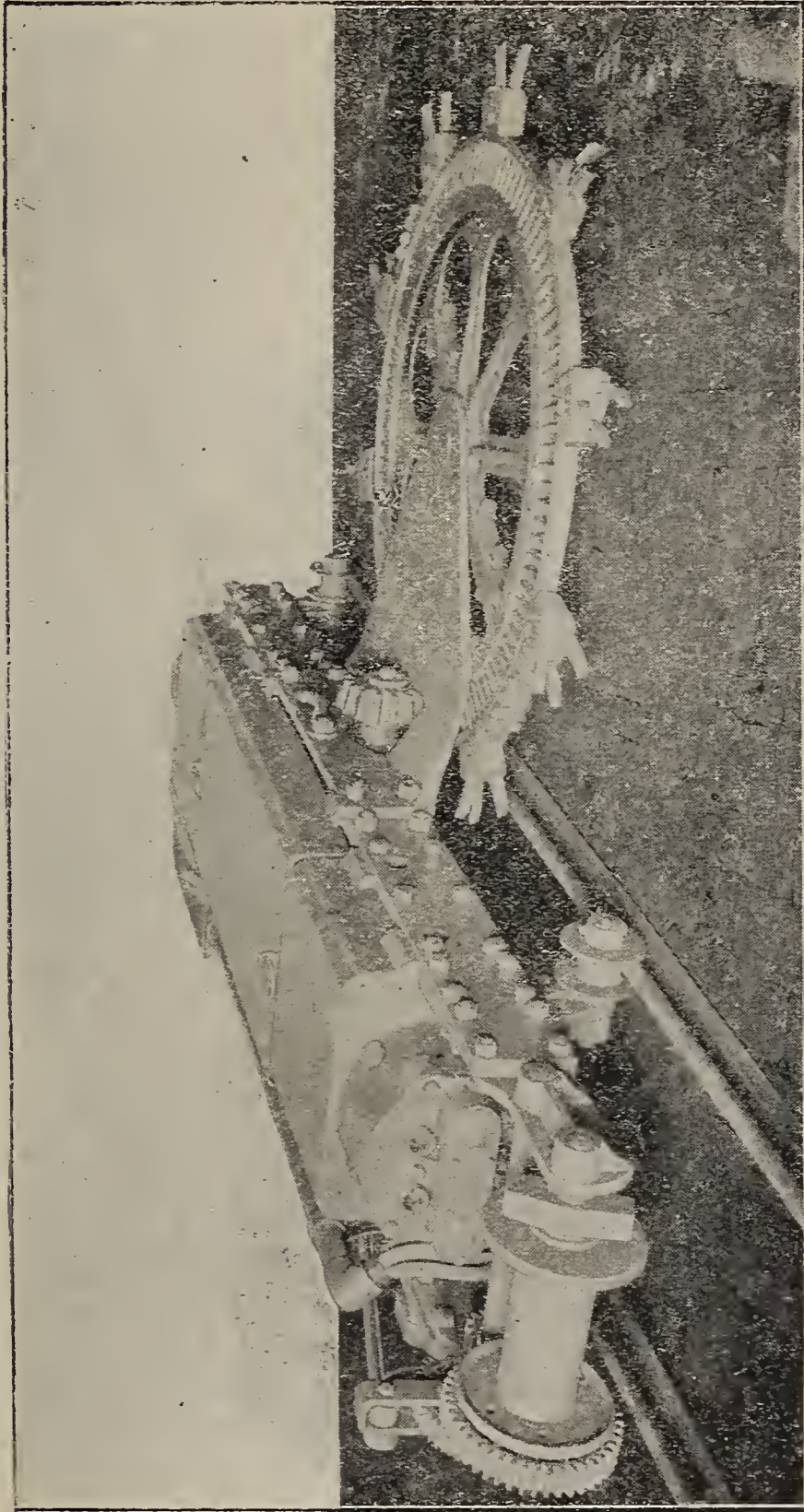


FIG. 130.—Diamond machine, constructed to run on rails.

its capacity and is hard pressed, the driver can materially assist it by clearing back the cuttings from the wheel. This at first was considered a disadvantage, as it was thought that it would prevent the machine from cutting forward and backwards if required, but it

has been found that by changing the cutters in the disc (*i.e.* putting in cutters suitable for the direction in which the machine is to travel) the machine can cut back equally as well as forward. At the same time that the cutters are changed the long bridle must also be changed to the other end, and the haulage rope drawn through under the machine. The Clarke & Steavenson machine has proved to be a good all-round coal-cutter for working under very various conditions.

Diamond Machine.—The Diamond machine (fig. 130) is another well-known machine of the rotary disc type. A large number of these machines are at work in the Midland coal-fields, and have been found to give good results in seams where a deep under-cut is required. The Diamond Coal-Cutter Company, who manufacture the machine,

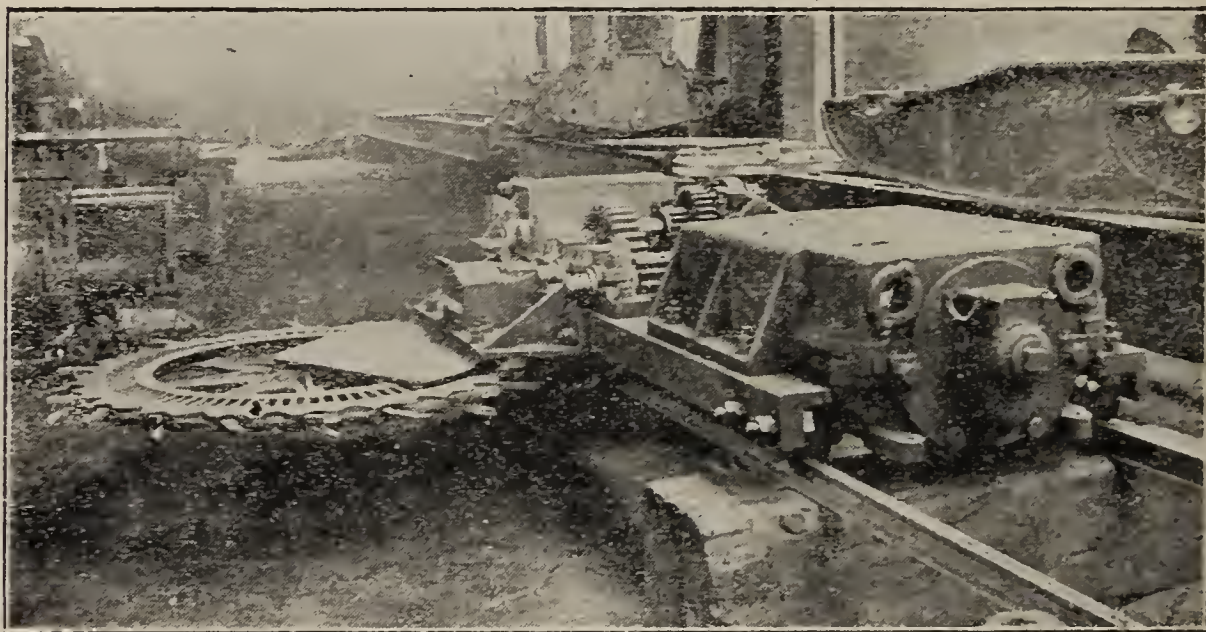


FIG. 131.—Anderson-Boyes machine.

were the first to introduce deep under-cutting, the depth of the under-cut varying from $4\frac{1}{2}$ to $6\frac{1}{2}$ ft., the latter depth of cut being employed in seams of medium thickness, *e.g.* 4 to 5 ft. Diamond machines are also made to cut in thin seams of 2 ft. and under, with an under-cut of 3 to $3\frac{1}{2}$ ft., as is usual in such seams. Two types of this machine are now constructed, those fitted with cylinders to work with compressed air, and those fitted with motors to work with electricity. Several new features have been introduced into these machines. In the compressed air machines the power-cylinders, instead of being placed side by side at one end of the machine, as is the usual practice in the other disc cutters described, are placed one at each end of the machine, an arrangement which has an important effect in assisting to balance the large cutter-wheel. This arrangement also decreases the width of the machine, which is a point of no small importance, especially where cutting with machinery has to be done in seams with a bad roof. Another feature of this machine is that by

fitting on an extra pair of axles and wheels and turning the machine over, it can be made to cut at any height in the seam, and it can cut in either direction. The cutting wheel, which is of the usual disc pattern, is attached to a strong bracket fixed to the framework of the machine.

Anderson-Boyes Machine.—The Anderson-Boyes machine (figs. 131 and 131A) is another of the longwall machines of the disc type, made by the firm of Messrs Anderson, Boyes & Company, of Motherwell, N.B. The machine, which is electrically driven, has been mainly designed by Mr Daniel Burns, M.E., who has had a large practical experience in the installation of coal-cutting plant in Scotch collieries, especially thin seam collieries, and the design of the machine is the outcome of that experience, the improvements which he has introduced being suggested by the defects in other coal-cutters which are now at work in our coal-fields. Of course, every machine has some imperfections, and the ideal machine to work under varying conditions is as yet in the experimental stage. The following is a description of the machine:—The frame, which is of heavy angle steel $4\frac{1}{2}$ in. by $4\frac{1}{2}$ in. by 1 in., extends along each side, and is braced by the motor box at one end and the switch box at the other, both being attached to the frame by heavy fitted bolts.

The cutter wheel is 4 ft. 6 in. in diameter, and carries twenty cutters, ten doubles and ten singles. The peripheral speed of the cutting wheel is about 300 ft. per minute, and the reduction gear is carried on three shafts. In order to keep the pinions closely in gear the plummer blocks are all cast in one piece, the arrangement of the gearing for the cutter wheel being such as admits of the use of a short stiff bracket which carries the wheel and arc-plate. The haulage gear is fitted low down in the body of the machine just behind the switch box, and is arranged so as to take the rope upon the under side, whether the machine is cutting back or forward. An arrangement is fitted to the ratchet-wheel which admits of the feed being thrown off or regulated while the machine is running. This is fixed



FIG. 131A.—Anderson-Boyes disc type coal-cutting machine.

in a position which is easily accessible to the person driving the machine, and is an appliance which should save a great deal of time where the cutting has to be done in under-clay that varies much in hardness. The most important part about the machine is the motor, which is specially constructed for the work, and is of much greater power than those most commonly used in coal-cutters, being 39 horsepower if the machine is series wound, the class of winding most generally used for coal-cutters; if shunt wound it gives 38 horsepower.

A large number, between two hundred and three hundred, of the Anderson-Boyes machines are at work in thin seam collieries in Scotland and the North of England, giving excellent results.

Jeffrey Machine.—The Jeffrey machine is an American type of disc machine which has been introduced into this country. It is constructed in two types, viz. to be driven by compressed air and to be operated by electricity. The construction of both types is nearly similar, in the latter a motor being substituted for the air cylinders. The electrical machine consists chiefly of two parts, the motor and feeding gear on its frame, and the cutting wheel. The frame of the machine is of the usual rectangular shape, and made of steel, with the haulage drum for moving it along the face placed at the forward end. The motor is placed in the middle of the frame, and the cutting wheel is placed at the extreme rear end. By means of a hand wheel the disc can be tilted up or down so as to follow any unevenness in the floor, or pass any obstruction in the holing, such as ironstone balls, which may be encountered. A device is also provided for altering the speed of travel of the machine without changing the speed of the cutter wheel. Another novelty in connection with this machine is that it only requires a single rail to run on, this rail being placed under the centre of the machine, the side thrust being taken up by special sleepers and light screw-jacks. The machine is operated from the front end, the feed being driven by an eccentric through a ratchet and pawl on to the haulage drum, and there is an arrangement for enabling the feed to be stopped, started, or adjusted without it being necessary to stop the motor. This enables the machine to clear itself should it get hampered by a fall of coal. Three rates of cutting are provided for—8, 16, and 25 in. per minute, or *vice versâ*, should such lowering of speed be required, according to whether the under-cut is soft or hard. The machine is constructed to cut either forwards or backwards, making it suitable for working in a short length of face. The high-speed wheels gearing the armature down to the cutting wheel and feed eccentric are enclosed in a casing arranged so as to run in oil, which reduces the noise of the machinery while cutting is being done, and so enabling the attendants to hear any movements in the coal or roof more readily. The cutters, which are all singles, are secured in position by suitable boxes on the periphery of the wheel. One half of the cutters

are set with their faces upwards, and the other half downwards. Different sizes of cutting wheels are used according to the depth of under-cut required, the smallest size cutting 3 ft. 6 in., and the largest cutting 6 ft. under, the height of the cut being 4 to 6 in., according to depth. The electric machine is fitted with a 25 horse-power motor, and weighs about 34 cwt., the principal dimensions being 8 ft. 2 in. long, 3 ft. 8½ in. wide, and 19 in. high from the top of the rail.

We have only described a few of the disc types of machines, as the space required to include them all would be much too large for a text-book on mining. Other disc machines in use are the 'King,' 'Samson,' 'Crescent,' 'Birtley-Falcon,' 'Scott and Mountain,' etc., machines all of which are constructed on similar lines to those described, but each having some special feature of its own.

c. Chain Machines.—These machines are of comparatively recent date, having been first introduced into American mines in 1894; but since that time they have rapidly made their way into a large number of mines in the United States, and are now also being used to some extent in British mines.

Two types of chain machines are in use; those made for cutting in narrow work, *i.e.* in pillar and stall workings, and machines like the Morgan-Gardner for longwall work. The principle of the chain machine is almost that of the band-saw, with the addition of suitable mechanism for moving the machine while it is cutting. The machine consists of three principal parts:—(a) A fixed carriage which can be clamped fast to the side of the working place by screw-claws, the cut being effected laterally, the claw holding the carriage in front being set at an angle of 45° so as to take up the thrust; (b) a movable frame in the shape of a trapezium that rolls longitudinally on the carriage, the short base of which carries the motor, while the long base is in contact with the face, having over its whole periphery a groove through which passes the endless chain; (c) the endless chain, moving in the groove of the frame, passing at the back round a driving pinion, and in front over two small guide pulleys. The cutters, which are often set upwards and downwards alternately, are fitted to the links of this chain. Nearly all of these chain machines are now driven by electricity.

Jeffrey Chain Machine.—The Jeffrey chain machine (fig. 132) is designed to work either in pillar and stall or longwall workings, and can be built to work by compressed air or electricity. It consists of three principal parts, the bed frame, the sliding chain cutter frame, and the motor carriage. Upon the frame are mounted the feed racks and a cross-bar on which rests the jack for taking the backward thrust. The cutter frame consists of one steel centre rail, the cutter head, and two side guides for the cutter chain. The cutter frame is triangular in shape, making it necessary to use only three wheels, two in the cutter head and the sprocket wheel, for conveying power to the cutter chain. The driving and feeding mechanism, consisting

of steel pinions and wheels for working the machine, are mounted on the carriage. The cutter bits, which are fixed in the ends of the cast steel cutter links, are straight, with a slight hook at the cutting end. When it is set working, the chain begins to travel round, while at the same time the cutter head and frame are advanced into the coal. The machine having cut to the full depth, the direction of travel is reversed and the cutter frame drawn in again. The machine is then slid along a distance about equal to that of the cutter head, and the work proceeds as before. The machines are built to under-cut $5\frac{1}{2}$ and 7 ft., the height of cut being 4 to 5 in.

Two types of the Jeffrey machine are constructed, the ordinary type for working in moderately thick seams, and the low type constructed for operating in thin seams. The construction of both machines is practically alike, the only difference being that in the

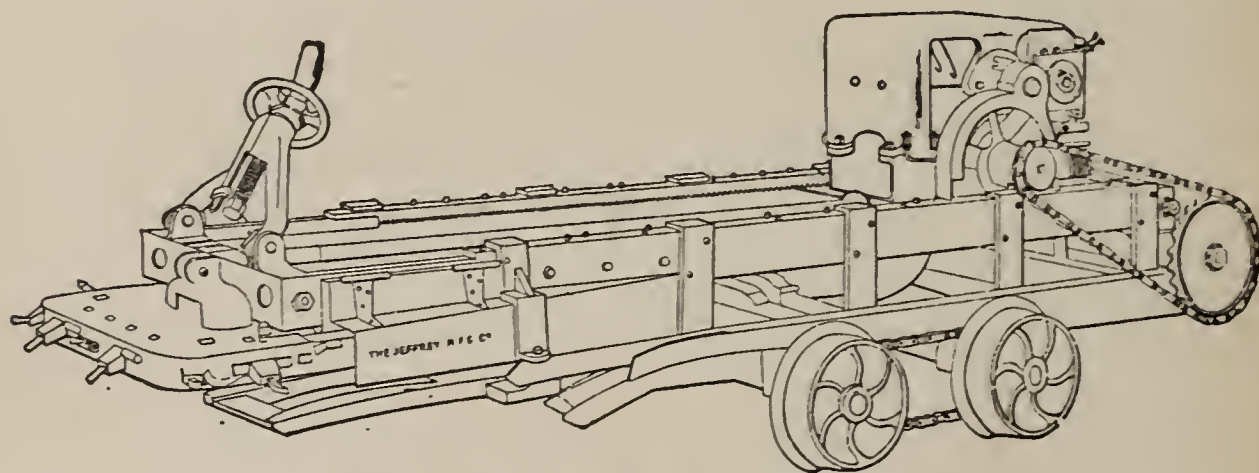


FIG. 132.—Jeffrey chain machine.

low type the motor is set much lower down in the carrying frame, which reduces its height.

Another type of the chain machine is the Morgan-Gardner, which in construction resembles very much that of the Jeffrey machine, with the exception of some minor details, and so does not require further description.

Anderson-Boyes Chain Machine.—This machine was introduced quite recently, and was specially designed for cases where the under-cut had to be made in coal or soft fireclay. Its motive power is electricity, and it is constructed both for continuous and three-phase current.

The main gear consists of a large crown bevel wheel attached to a vertical shaft which carries the chain sprocket wheel. The gear is contained in a cast steel gear case, arranged with covers for withdrawal and inspection of the gear, the whole being oil-tight, to admit of the wheels running in oil or grease. The chain which carries the cutters runs in channel steel guides, which are fixed to the sides of a heavy steel jib. Provision is made for adjusting the tension of the chain on the jib by means of a movable jib-head, and also for turning the jib into or out of the cut. The haulage gear has suitable

provision for adjusting the feed, and for throwing it on and off while the machine is running. The machine is mounted on skids, and carried on them by means of adjusting screws arranged for vertical adjustment by means of a ratchet spanner. The height of the machine is 1 ft. 8½ in., the width 2 ft. 11 in., and length 8 ft. 9 in. Like other machines of this class the jib can be adjusted for either cutting at the top or bottom of the seam. The normal depth of under-cut given is 4 ft., but deeper cuts can be got by fitting on a longer jib.

Hopkinson Chain Machine.

— This machine is made to be driven either by electricity or compressed air, and it has been adopted in quite a number of collieries in the Midlands. It is made in two sizes, one 8 ft. long × 2 ft. 2 in. wide × 2 ft. 2 in. high, giving a maximum under-cut of 6½ ft.; and the other 6 ft. 6 in. long × 1 ft. 10 in. wide × 1 ft. 10 in. high, giving an under-cut of 4½ ft. One of the outstanding features in this machine is the use of an extremely strong chain, and the attachment of the cutting tools to it by means of a cutter box, a single and a double tool being carried alternately. It is also probably one of the narrowest machines of its kind, which is an advantage for cutting in seams with a broken or tender roof. The jib can be detached when necessary, and it is mounted on a turn-table, which allows of its being swung through an angle of 180°, thus enabling the machine to cut its way into the

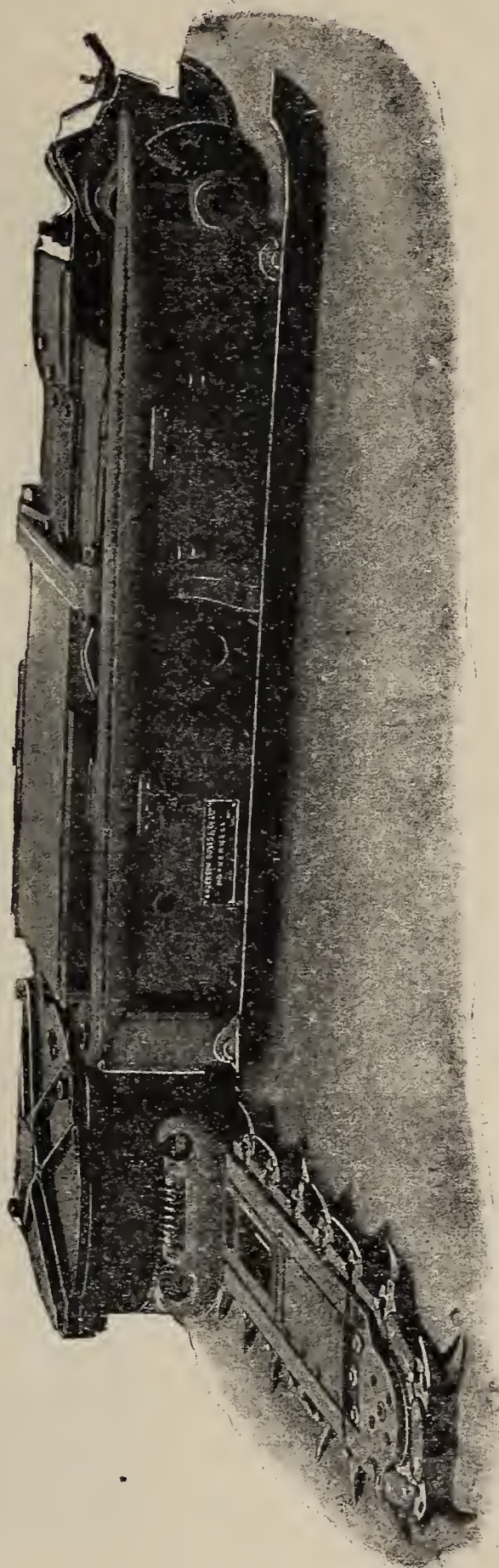


FIG. 133.—Anderson-Boyes chain machine.

coal at the start, and by turning the machine round it can be made to cut its way out at the finish of the machine run. This

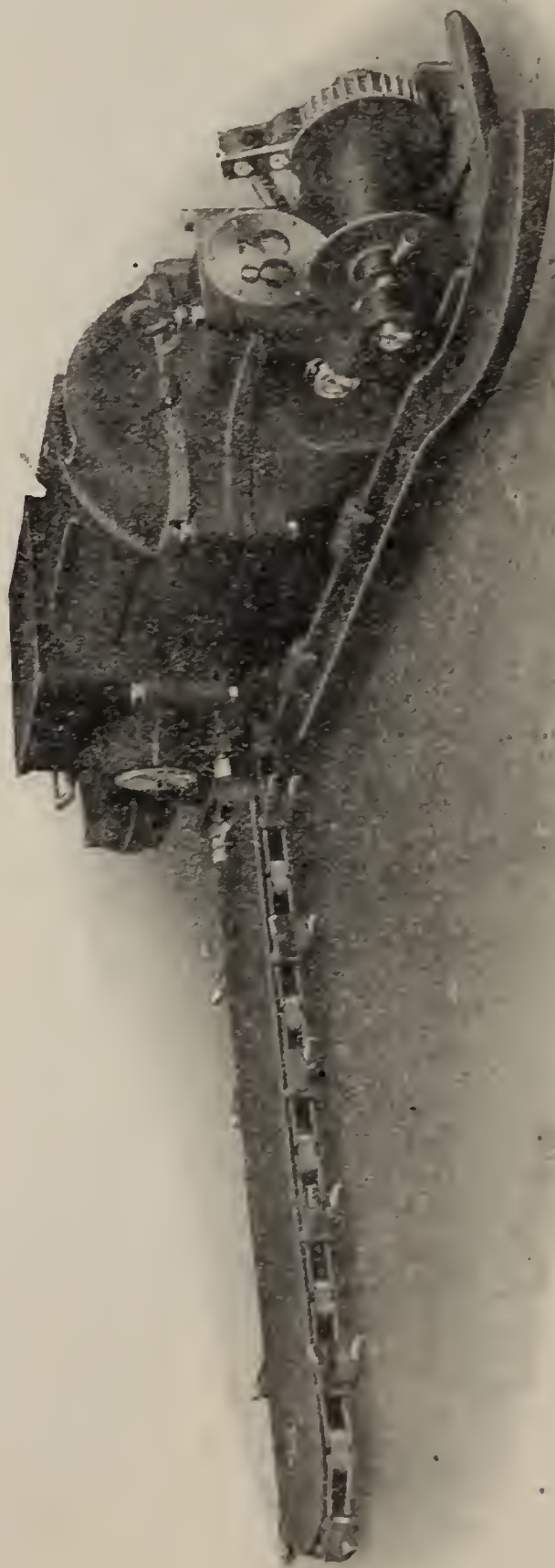


FIG. 134.—Hopkinson chain coal-cutting machine.

avoids the necessity of making jib-holes or 'stables' at the start and finish of the cut.

Other chain machines are the 'Diamond,' the 'Mather & Platt,' the 'Goodman,' etc., which are all used for longwall work, and which are constructed much on the same lines as those already described.

Chain Shearing Machine.—This type of machine is used for shearing or side-cutting the coal in narrow places, such as in pillar and stall working. The construction of the machine is somewhat like the chain under-cutting machine, but with the frame round which the cutter chain passes placed in a vertical instead of a horizontal position. The outside frame is constructed of heavy steel angles with the flanges turned in, which gives a broad wearing surface for the carriage that carries the motor and gearing. The machine is fitted with a motor at the rear end of the machine which drives the shearing chain through spur and bevel gearing.

d. Bar Machines.—This is a type of machine which is very different—so far as the tool used for under-cutting is concerned—from the two classes of machines just described. Instead of the under-cutting being done by a disc or moving chain, a circular tapered steel bar with a number of teeth or cutters fixed in its periphery is employed to do the cutting. For under-cutting in certain seams this type of machine has some advantages over the disc cutter. In seams which are soft and friable, if disc machines are employed a good deal of trouble is often experienced by the under-cut coal coming down on the disc before it can clear itself, or before sprags or holing props can be put in to support the cut coal. The larger the disc used, the more likely is there to be trouble from this cause, as the area of under-cut and unsupported coal will vary as the diameter of the cutting wheel. These falls of coal taking place on the disc cause delay, as the machine requires to be stopped until the fallen coal is cleared away; and where cutting is being done in thin seams with a bad roof where propping is carried close to the machine, the clearing of such coal is often a tedious and difficult process. Where such conditions prevail, the bar machine may be employed with good results, as only a small area is under-cut and left unsupported for a very short time, as the spragging can be done quite close to the cutter bar. Another advantage that is claimed for this type of machine is that less power is required to drive it than a disc or chain machine for an equal depth of under-cut. In actual working practice we do not think there is a very great difference in the power required for driving the two classes of machines, and it is a point on which at present too much value need not be put, for the power required for driving is after all not of so much importance as the time required to cut a given area by a machine.

Other advantages claimed for the bar machine are that there are no bearings under the coal when cutting is proceeding, as in the centre of a cutting disc; that the machinemen are not so hard worked as with disc machines; that the holings are better cleared out; and that the cutters are examined more easily.

'Pick-Quick' Machine.—This is one of the best-known machines of the bar type (figs. 135, 136, 137), and is made with either compressed air or electric driving. It consists essentially of five parts, viz. (a) the cutter bar; (b) gear head; (c) the motor; (d) the switch box; and (e) the hauling gear. The cutter bar is machined out of the solid, and the spiral thread formed on it acts as a worm conveyer to bring out the cuttings. Taper holes with 'feather' ways for the cutter or pick shanks are drilled between the threads. On the inner end of the bar a thrust collar block is fitted, and through this a reciprocating motion is imparted to the bar. The gear head completely encloses the main gearing. A pinion on the motor shaft drives the double bevel wheel, which in turn drives (with a reduction of 2 to 1 in speed) the cutter bar pinion. This pinion is not keyed rigidly to the bar, but is fitted with two 'feathers' projecting into its bore. The feathers engage in long key-ways in the bar, which, while being driven, is thus free to reciprocate along its own axis. Access to the whole gear is obtained by removing the gear case cover. A circular rack in the lower part of the gear case, worked by a pinion, provides for turning the lower part of the gear head with the cutter bar in a horizontal plane for cutting. The reciprocating motion is obtained from a worm upon the boss of the bar-driving pinion geared into two wheels, each of which drives a toggle by means of an eccentric pin. The toggles impart a to-and-fro movement to the thrust block and with it the cutter bar. The combined rotary and reciprocating movement thus imparted to the bar gives a chipping and shearing action which prevents the bar from clogging and the cutters from working into grooves. The whole gear case being completely enclosed, lubrication is ensured by filling the lower portion with oil. At the outer end of the bar bearing a stuffing-box is provided to keep the oil in and the dirt out.

Haulage Gear.—In the switch-box casting, and accessible through the tool box, is a chamber into which a worm on the end of the armature shaft projects. This worm gives motion to a cross-shaft which passes through the side of the box, and carries outside it a disc with pin (adjustable radially) which operates the connecting rod, and by it the haulage gear.

The cutter picks are made in two styles, single and double pointed, the former being used in the body of the bar, and the latter only at its extreme point. For coal holing the cutters are sharpened to a point like an ordinary hand pick, and for fireclay holing is better dressed to a chisel shape about $\frac{3}{8}$ in. wide. The bar is made in different lengths to give an under-cut of 3 to 6 ft., and the depth of cut 4 to 6 in. The cutter bar revolves at a speed of 300 to 500 revolutions per minute. The machine can be mounted either on wheels or skids, and the cutting bar can be adjusted to cut either at the bottom, top, or intermediate positions in the seam.

Other machines of the bar type are the Hurd, Lee, and Simplex

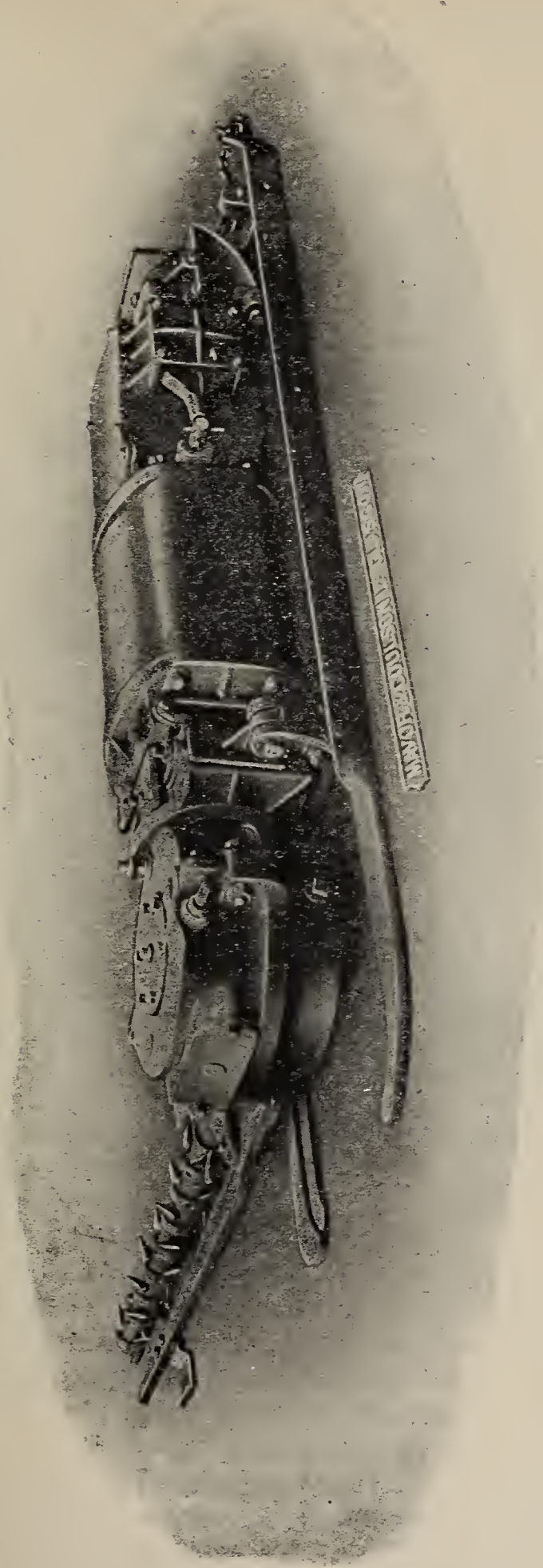


FIG. 135.—The Pick-Quick, small size, with three-phase electric motor, adjusted for under-cutting.

machines, all of which in the main are very similar in construction to that described above.

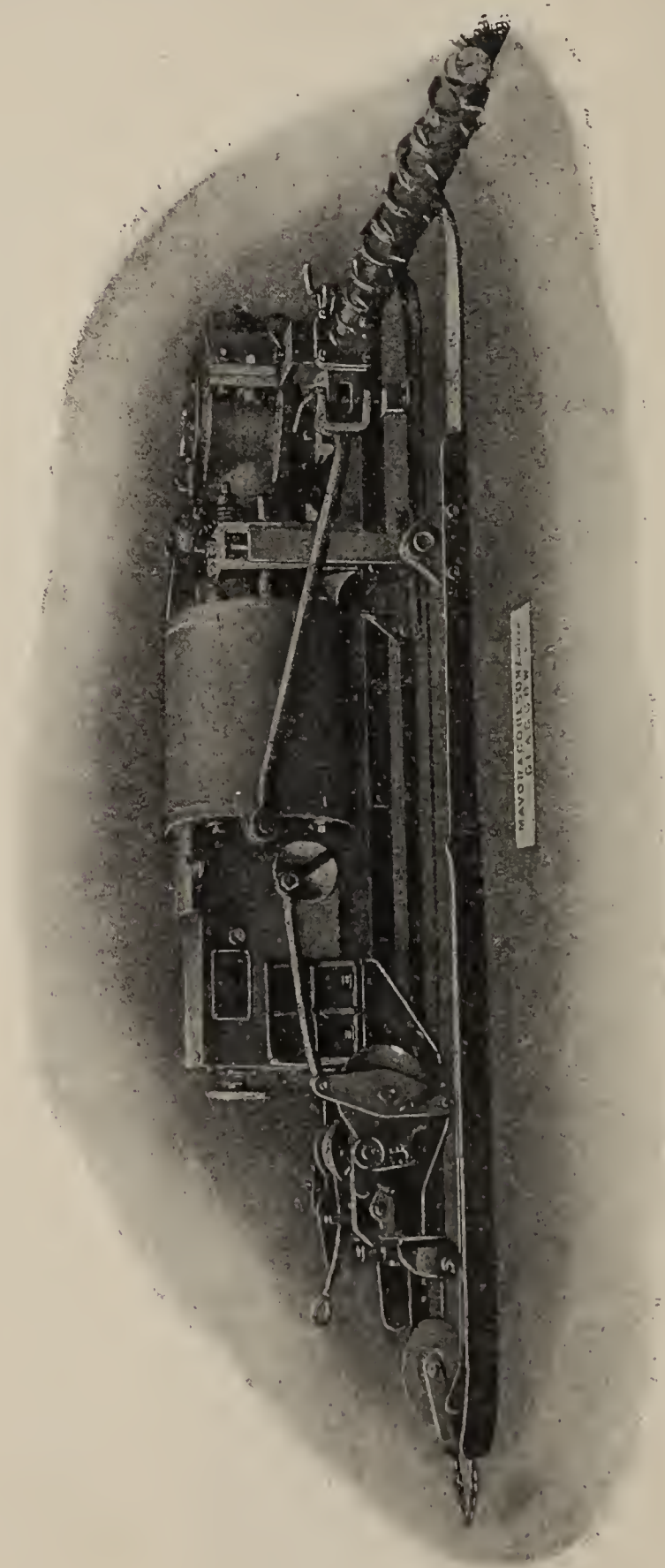


FIG. 136.—Pick-Quick, medium size, with three-phase electric motor, adjusted for intermediate holing.

e. **Percussive Machines.**—The percussive machine, as its name implies, under-cuts the coal by a series of blows in much the same way as a miner would under-cut with his pick, so that their action

is altogether different from the machines previously described in which the under-cutting is done by a continuously rotating disc, chain, or bar. Percussive machines are, therefore, practically

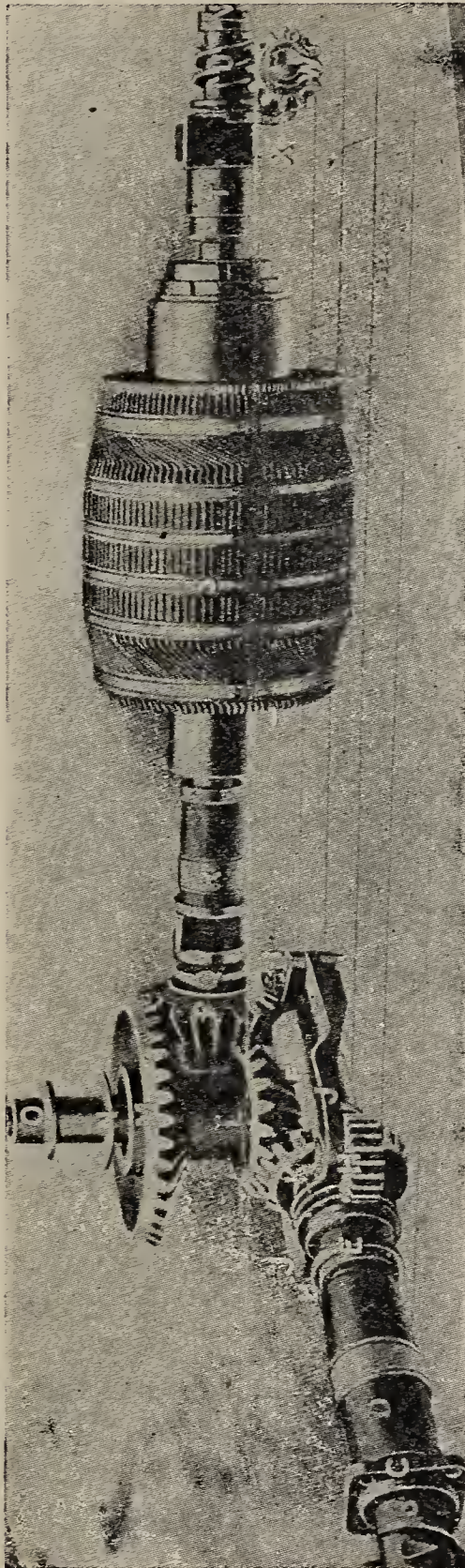


FIG. 137. — Arrangement of gearing in Pick-Quick electric coal-cutter.

- | | | |
|--|--|--|
| A. Cutter picks. | J. Reciprocating rods. | R. Gear-end bearing for armature, stator, or C/shaft. |
| B. Cutter bar, right-hand thread. | K. Cutter bar mitre wheel. | T. Haulage-end bearing for armature, stator, or C/shaft. |
| C. Cutter bar gland. | L. Mitre wheel with long boss. | U. Haulage worm. |
| D. Cutter bar outer bearing. | M. Crown bevel wheel. | V. Shaft-end socket for armature, stator, or C/shaft. |
| E. Cutter bar mitre wheel thrust collar. | N. Crown bevel wheel washers. | W. Haulage worm wheel. |
| F. Cutter bar V washer. | O. Mitre wheel spindle. | X. Haulage shaft bearing. |
| G. Cutter bar inner bearing. | P. Bevel pinion for armature, stator, or C/shaft. | |
| H. Reciprocating wheels. | Q. Bevel pinion washer for armature, stator, or C/shaft. | |
| I. Thrust block brasses. | | |

pneumatic or electric picks. They resemble mechanical rock-drills in that they are only furnished with a single cutting bit, though they differ from them in having no mechanism for rotating nor a screw for giving the feed, while, instead of being fixed at work, they are essentially movable. These machines were first introduced into

American mines about 1880, and since then they have been extensively employed for mechanical cutting in the coal mines of the United States and Canada. In quite a large number of the mines in those countries the under-cutting is entirely performed by percussive cutters, although in some districts they are being gradually displaced by the chain machine. During recent years the percussive type has been largely employed in British mines (over five hundred in 1910), for opening out work, and in pillar and stall working. They take up little space, are easily worked, and can be moved about with facility from place to place. All percussive machines are of one type



FIG. 138.—Cutter pick, half size, for bar machine.

and are worked on a similar principle, though differing somewhat in detail from each other.

They are constructed with the following main parts * :—

- (a) A cylinder 3 to 4 in. diameter and 9 to 12 in. stroke.
- (b) A piston and rod, with the rod extended to make the cutting tool.
- (c) A valve and valve-chest, the valve being thrown by air, or operated by a small independent engine, cushioning arrangements being in all cases a necessity.
- (d) A sleeve or guide attached to the front end of the cylinder, to guide and steady the piston-rod, and also provided with arrangements for preventing the piston and cutting tool from turning.
- (e) A suitable pick or cutter, usually sharpened to a fish-tail shape.
- (f) A pair of plain wheels upon which the machine is carried, the wheels being placed so as to balance the machine.
- (g) A pair of handles at the back for the machineman to take hold of.

* *Engineering Magazine*, August 1903, p. 125.

Method of Working Percussive Machines.—"In operating a pick machine the runner sits on a board or platform (fig. 139), inclined to the face of the coal; one foot of the operator is braced against one wheel of the machine, and with the two handles he directs it against the coal, picking off the coal exactly as a miner would do, except with much more force to each blow. The under-cut made is V-shaped, 12 to 18 in. in height at the face, and tapering back to a feather edge on the floor at the rear of the cut, the depth of the cut being from 3 to 6 ft. deep, according to the thickness of the coal. A helper shovels away the cuttings as the machine, guided by its operator, loosens the coal in the under-cut." *

The work in operating these machines is severe, especially with unskilled machinemen and in hard coal, for the inclined board and



FIG. 139.

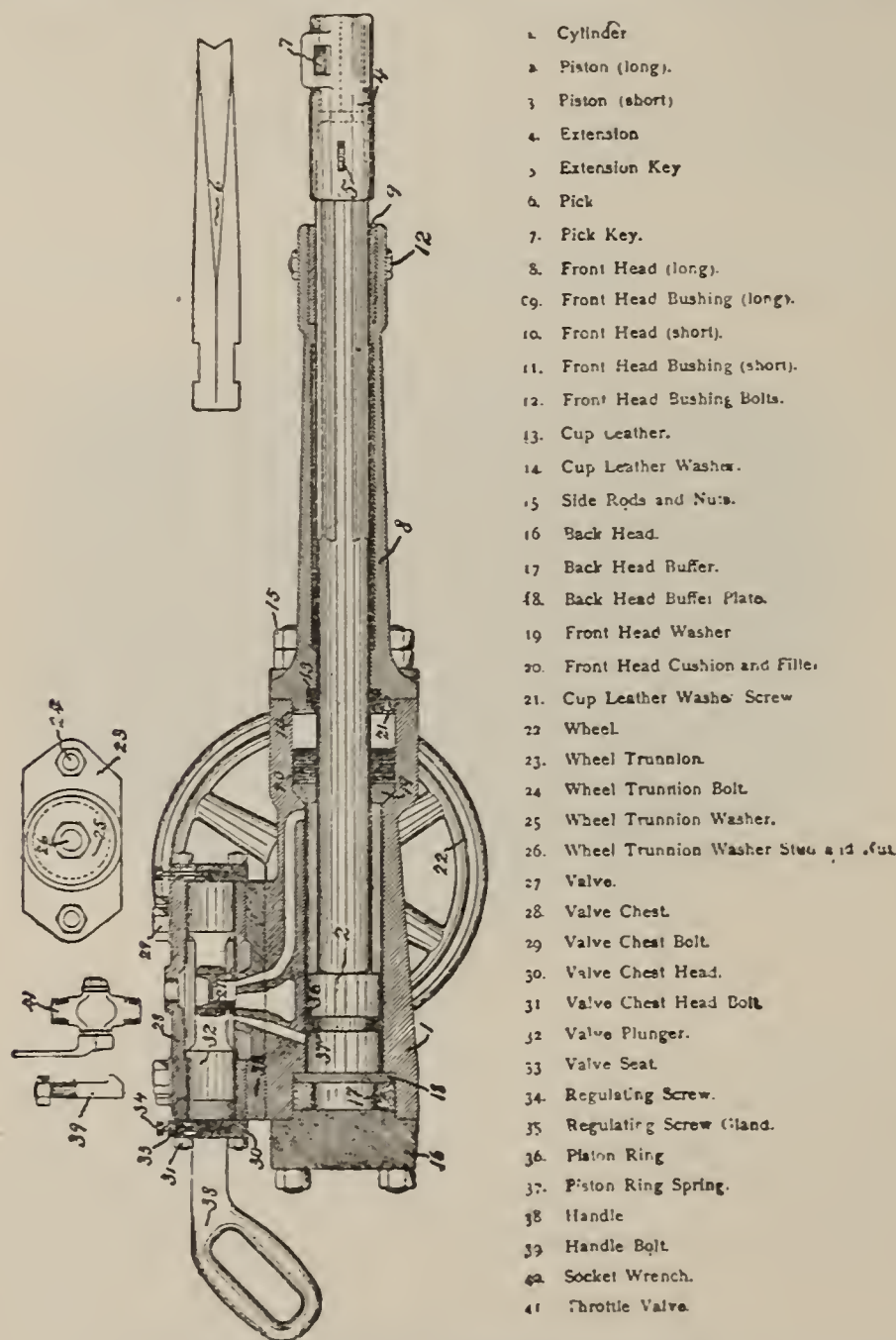
the chock provided by the operator's foot only partially take up the recoil of the machine. In a hard seam, unless the machineman can rest frequently, the labour of keeping the machine up to its work becomes very fatiguing, in fact almost unendurable, and hence, as already stated, they are best suited for cutting in soft and friable seams.

There are now quite a number of these machines in use, but we need only describe one or two, as they are all so much alike.

Ingersoll-Sergeant Machine.—The Ingersoll-Sergeant machine (as also the Harrison machine) is fitted for working in narrow 'rooms' or drifts. It is simply a chisel attached to the continuation of the piston-rod of an air cylinder, and may be called a slotting machine. The machine (fig. 140) is mounted on a pair of wheels to enable the cutting tool to operate in any direction, either under-cutting or cutting in an upward direction, and to facilitate the removal of the machine from point to point. The tool receives a reciprocating

* *Mines and Minerals*, June 1903, p. 510.

motion from a small cylinder of the usual type fixed to the frame of the machine. The piston works through suitable packing at the front end of the cylinder, and is made long enough to project beyond the end of the sleeve in which it works when at the end of its back stroke. At the forward end of the cylinder is placed a buffer of leather of such construction as to form a cushion for the piston.



1. Cylinder
2. Piston (long).
3. Piston (short)
4. Extension
5. Extension Key
6. Pick
7. Pick Key.
8. Front Head (long).
9. Front Head Bushing (long).
10. Front Head (short).
11. Front Head Bushing (short).
12. Front Head Bushing Bolts.
13. Cup Leather.
14. Cup Leather Washer.
15. Side Rods and Nuts.
16. Back Head.
17. Back Head Buffer.
18. Back Head Buffer Plate.
19. Front Head Washer
20. Front Head Cushion and Filler.
21. Cup Leather Washer Screw
22. Wheel.
23. Wheel Trunnion.
24. Wheel Trunnion Bolt.
25. Wheel Trunnion Washer.
26. Wheel Trunnion Washer Stud and Nut.
27. Valve.
28. Valve Chest.
29. Valve Chest Bolt.
30. Valve Chest Head.
31. Valve Chest Head Bolt.
32. Valve Plunger.
33. Valve Seat.
34. Regulating Screw.
35. Regulating Screw (Hand).
36. Piston Ring
37. Piston Ring Spring.
38. Handle
39. Handle Bolt.
40. Socket Wrench.
41. Throttle Valve.

FIG. 140.—Ingersoll-Sergeant machine.

The projecting part of the piston-rod is tapered so as to fit a corresponding taper in an extension piece which is fixed to the piston-rod by a key. The outer end of this extension piece is made to receive the pick or chisel, which is also held in position by another key. The piston-rod is provided with eight straight grooves engaging in similar projections in the end bushing for the purpose of preventing any accidental movement of the piston and pick. The cutting edge of the pick is shaped like a **V**, having a sharp edge and

cutting points. The pick can be easily removed and sharpened. The machine, as already described, is worked from an inclined board of convenient size, while making an open channel under the coal of 4 ft. to 5 ft. of face and 3 ft. to 5 ft. of under-cut. The claims of this machine are that it is small, simple in construction, requires little space, is light—weighing only about 750 lbs.—portable and cheap.

The Harrison machine is very similar to the Ingersoll-Sergeant, but is somewhat heavier and stronger. The air distribution is effected by a small auxiliary motor, which cuts off a half-stroke, and the cylinder head is protected by a leather cushion, which deadens the blow of the piston when the bit or cutter does not strike the coal. With an air pressure of 70 lbs. per square inch this machine gives 190 to 210 blows per minute, with a stroke of 11 in. It makes an under-cut of 6 ft. deep and about 6 in. in height.

The Sullivan machine, which is also driven by compressed air, has a system of variable expansion that permits of graduating the blow, and the cylinder head is protected by an air-cushion. It greatly resembles an ordinary rock drill, is mounted on a carriage running

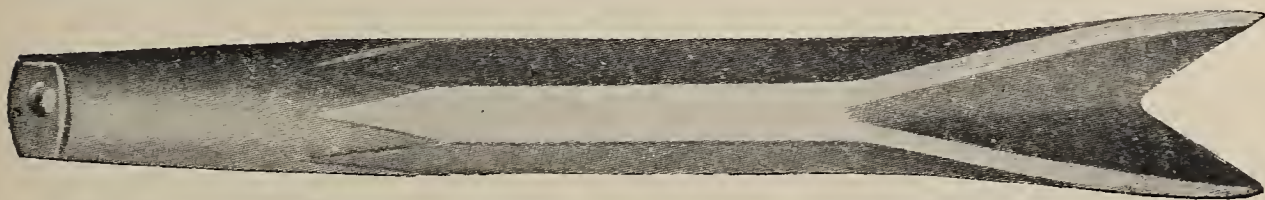


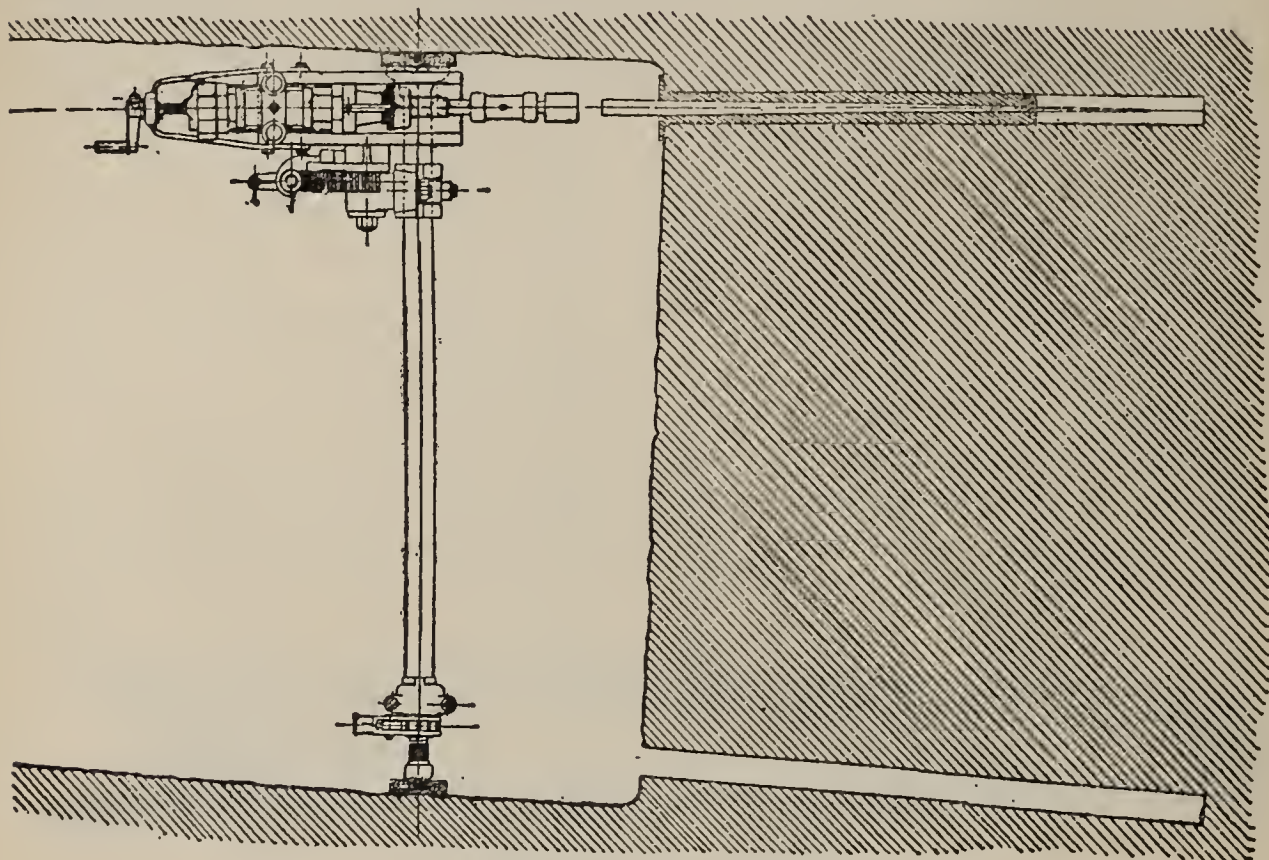
FIG. 141.—Sullivan machine pick.

on rails, and the bit, much longer than usual, is supported by an arm that also acts as a guide. This appliance can be moved in a vertical plane so as to make a cut in the face for its whole height; and the forward feed is given by a small winch on which a chain is wound, so that a cut 6 in. wide can be made to a depth of $7\frac{1}{2}$ ft.

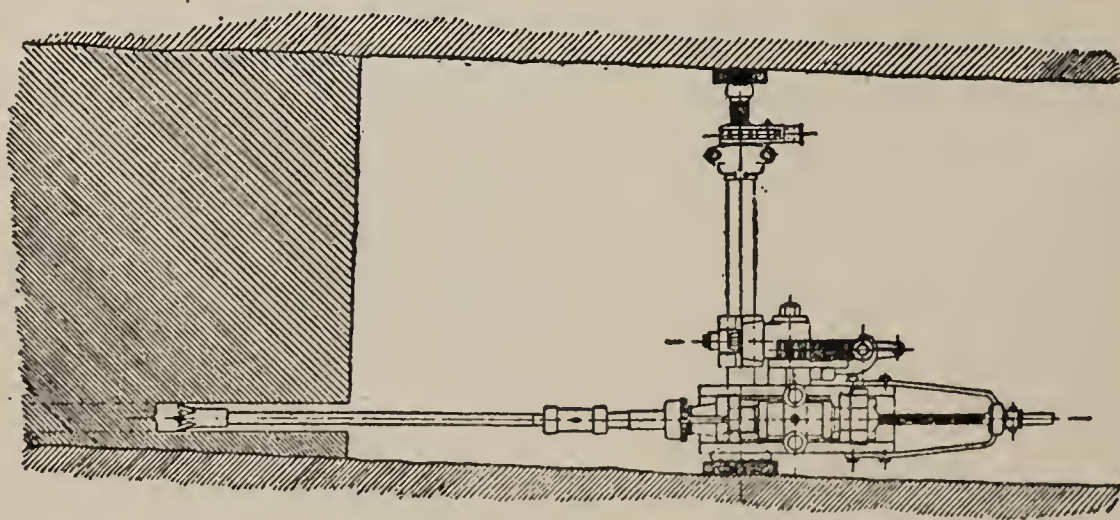
The Morgan-Gardner percussive machine, unlike the three above described, is driven by electricity, and is practically the first of its class which has been successfully worked by electrical power. The construction of the machine is practically similar to those just described, with the exception that an electric motor is substituted for a compressed air cylinder. The rod with the cutter attached is drawn backwards by a cam during half a revolution, and then a strong spring propels it sharply forward, the cam being worked by gear from a continuous current motor with vertical axis carried on the machine. It can give from 175 to 225 blows per minute, and cuts to a depth of $4\frac{1}{2}$ ft. The weight of the machine is about 750 lbs., and it has a total length over all of 7 ft.

Siskol Machine.—The Siskol, formerly called the Champion, machine is also of the percussive type, but differs very considerably in construction from those just described. It is practically a combination of the percussive machine and rock drill. The machine

consists of five essential parts, viz.: (1) The supporting column with clamp (5 ft. long), 156 to 200 lbs.; (2) the segment with accessories, 104 lbs.; (3) the air drill, 226 lbs.; (4) extension rods, five, for cutting up to 7 ft. deep, average total weight 80 lbs.; (5) the bit.



Machine set for wedging.



Under-cutting in thin seams close to floor.

FIGS. 142 and 143.—'Siskol' machine.

The supporting column, of simple construction, is securely fixed about 3 to 3½ ft. from the working face. The column is made in two different constructions, one with a nut and counter-nut, and the other with a ratchet attachment. Each column can be extended 1½ ft., so that a column may be lengthened by 18 in. to 4½ ft. Sliding on this column longitudinally is a sleeve, or hinged clamp,

which can be fastened at any given height of the support by means of a bolt and nut. In a bearing of the sleeve the toothed segment is carried, and may be rocked therein from a horizontal to a vertical plane, or to any intermediate position. Pivoting in the hub of the segment and in its axis is a connecting piece commanded by a worm acting on the teeth of the segment; a handle driving the worm may be attached to either end of its axis. The drill machine is securely fastened to this connecting piece, and can therefore be swung in a plane parallel to the segment by turning the handle of the worm; if, therefore, the segment is attached horizontally, the machine will act as a coal cutter; if vertically, as a coal shearer; if at an angle, the machine will cut into the coal at that angle. It will thus be seen that the machine can make a cut at any given height and at any given angle. It can also be used as an ordinary rock drill, and this simply by not turning the handle commanding the worm. The working of the machine is controlled by one man, who with one hand turning the handle of the worm produces a swinging action of the drill, while with the other hand he regulates the advance of the drill into the coal by turning, as in any ordinary rock drill. When the drill is set to work, the cutting-bit strikes the coal at the rate of about 350 blows per minute. It would penetrate very fast into the coal were the blows directed to one place only; but, while the machine is at work, the cutting-bit is gradually displaced in a plane parallel to the segment, by turning the handle of the worm; and the cutting-bit therefore strikes every blow in close proximity to the preceding one. It describes an arc having for its centre the axis of the segment, and the outcome of this motion is an even cut. When the cutting-bit has reached the end of an arc, the drill is advanced to the extent it has been cutting into the coal by turning the feed screw, and the motion given to the worm is then reversed.

The cutting-bits are provided with either three or five prongs or teeth, the length and design of these varying according to the conditions, the diameter of the cutting edge being about 3 in. The drill consumes about 20 cub. ft. of compressed air, equivalent to about 100 cub. ft. of free air, and the most suitable pressure is 60 to 80 lbs. per sq. in. If a higher pressure is available a smaller diameter of cylinder may be used. In very steep seams it is recommended to use the smallest drill and a pressure of 100 to 120 lbs. per sq. in. The machine can also be utilised to drill holes for inserting compound wedges to wedge down the coal where blasting is not required or allowed. The Siskol is an excellent machine, and is largely employed in coal-mining, and can also be used for drilling shot-holes in stone-work.

'Hardy Puncher' Machine.—This is another machine of the type just described. It consists essentially of a long stroke air-drill, which is fixed in position and held up to its work by means of a

solid drawn steel tube provided with a powerful jack-screw and toothed head and foot. The machine is seated in a cone cup

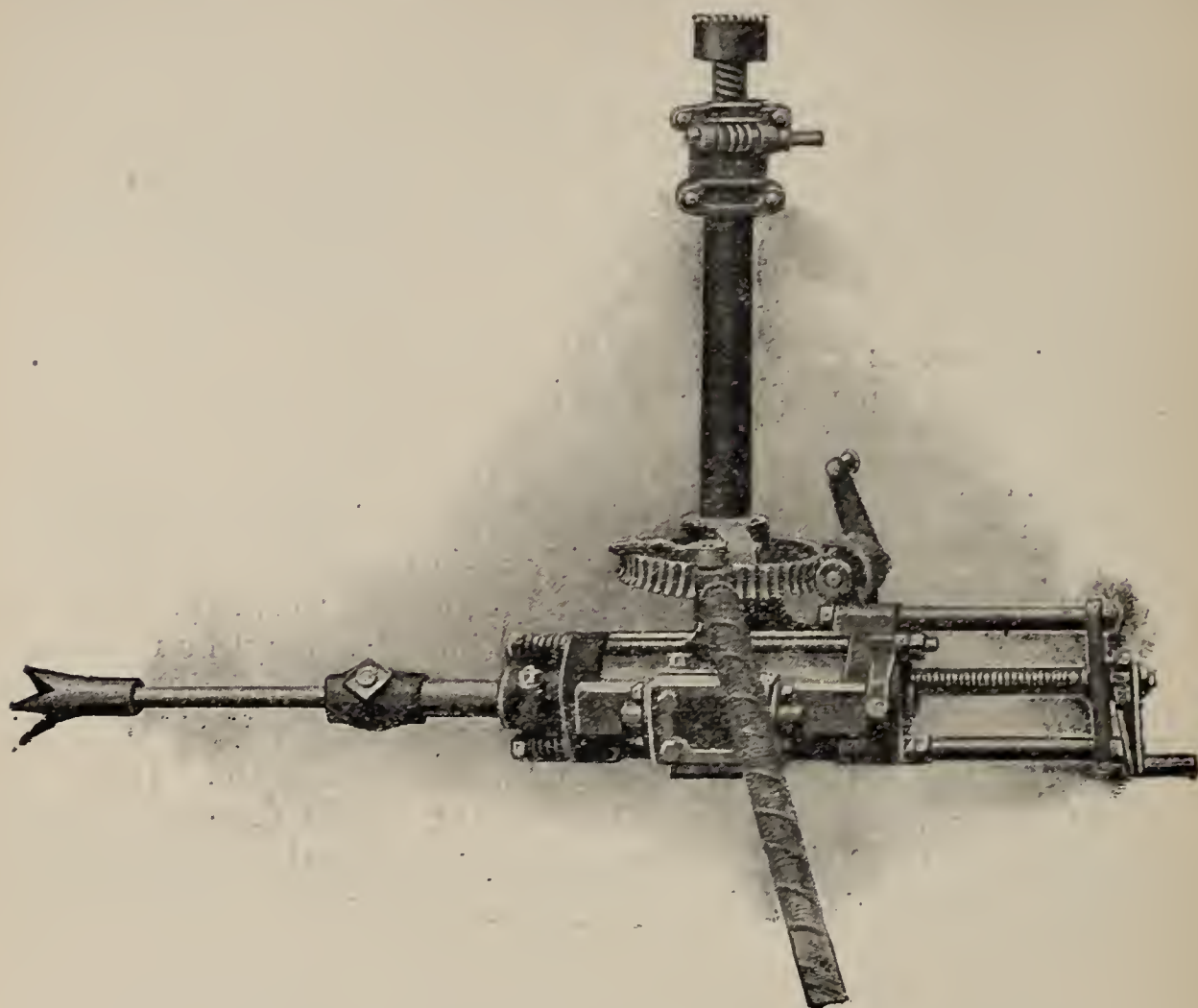


FIG. 144.—‘Hardy Puncher’ fixed for under-cutting. Side view.

forming part of a hinged clamp attached loosely to the column. This clamp carries a worm, the teeth of which mesh with those of a worm wheel bolted rigidly on the column.

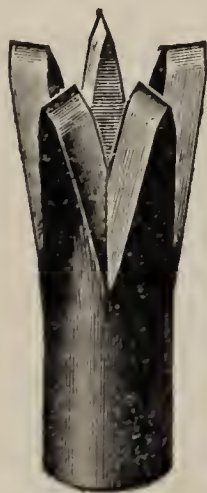
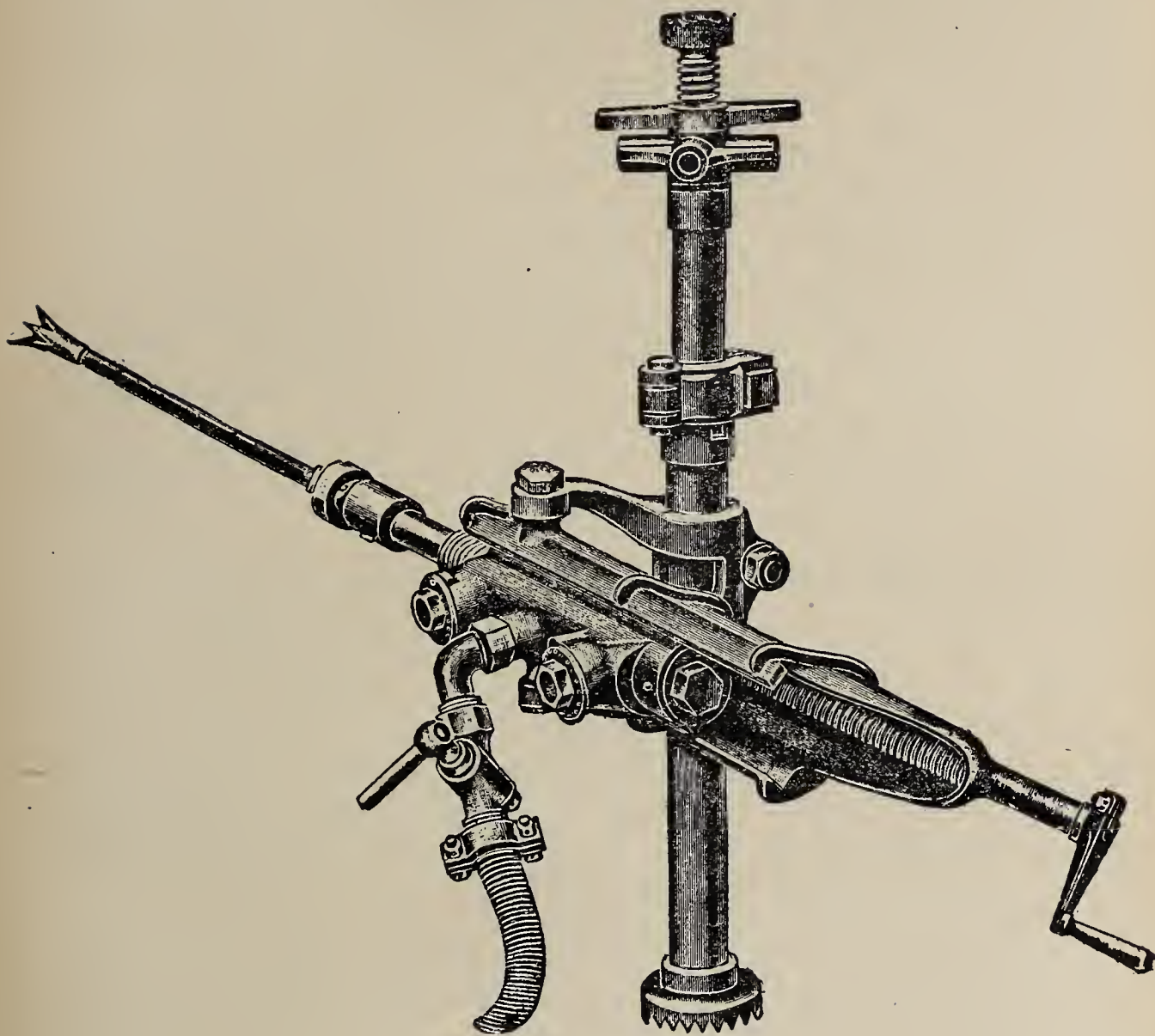


FIG. 145.—Cutter for ‘Hardy Puncher’ coal cutter.

By turning the worm handle the cutting bar is caused to move around the column, whilst the forward movement into the coal is obtained by the feed screw controlled by the handle at the back of the machine (fig. 144). The machine may be fixed to cut at any height between the floor and roof. When air is admitted to the machine the rapid and powerful blows quickly cut away the coal or other material, whilst the machine is swung right and left by the worm gear and fed forward as necessary. An arc-shaped and continuous under-cut is thus made, increasing in width and depth as each successive length of cutter bar is

inserted in the piston of the machine. This machine may also be used to either under-cut or shear the coal.

'Meco' Machine.—This machine is also of the percussive type, and is made by the Mining Engineering Co., Ltd., Sheffield. An advantage claimed for this machine is the suspension fork with which it is provided, which is said to take up all the recoil and vibration, making the handling of the machine simpler than those fitted with a quadrant. The valve gear is also different from other machines, there being two valves at each end of the cylinder, thus



"MECO" MACHINE.

FIG. 146.

avoiding waste air space. They work independently of each other, so that live air cannot enter both ends of the cylinder simultaneously. This arrangement increases the number and power of the blows, without increasing the air consumption. With this type of valve the air can be used expansively. This is accomplished as follows:— Each valve really consists of two cylindrical valves, one above the other, so arranged that when the piston is in a certain position they are closed by the air compressed at the end of the stroke, and are kept closed by means of a connecting port until released and allow-

ing live air again to enter. The valves are constructed in the form of small cylindrical cups. The piston-rod is fixed in a stuffing-box, which is held in position in front of the cylinder by means of a spiral spring. This spring is to prevent breakage, as it takes up any shocks which may occur when the machine is allowed to run free or when roughly handled.

These percussive machines can cut in either narrow work, as in pillar and stall, or in longwall work, and in the latter case are suitable where short lengths of face have to be cut, such as between faults, where it would not be suitable or expedient to employ the regular longwall type. While the American types of machines are made of the punching type working on a sloping platform, the British percussive machines are usually fixed to a rigid column. This arrangement relieves the machine attendant from shock or jar, and also allows the machine being also adapted for shearing or boring holes. There is also the further advantage that the machine can be set to cut at any required level in the seam, and the under-cut is not tapered to the same extent as that made by the American type mounted on wheels, but approximates more closely to the parallel cut obtained by the disc or chain machine. The three machines can under-cut on an average 40 to 50 sq. ft. per hour, the height of the cut being $3\frac{1}{4}$ to $4\frac{1}{2}$ in.

Choice of Machine.—If the owners or manager of a mine determine upon adopting mechanical cutting, the first question which will naturally arise is, What type and class of coal-cutting machine should be used for the work to be done? This is not so simple a question as it looks, and cannot be answered off-hand. The choice of machine will depend upon various circumstances, such as—(1) The mode of working at the mine; (2) the nature of the holing; (3) the position of the holing, *i.e.* whether the holing is to be done at the bottom, centre, or top of the seam; (4) the thickness of the seam; (5) the nature of the floor and roof.

If the mode of working is longwall, which is most usual in this country where coal-cutting is done by machinery, then the selection of a machine must be made from the disc, bar, or chain classes of coal-cutters. We have already stated that by far the larger number of machines used in British mines are of the disc type, and under ordinary conditions this seems to be the type most suitable; but, as already pointed out, there are certain kinds of seams in which it would be possibly better to adopt the bar machine, *e.g.* where the coal is soft and friable, or where the holing has to be done in the centre or at the top of the seam. Hitherto chain machines have been little used in this country, but with the newer types now available they will have a more extended use, both for pillar and stall and longwall work, where the conditions are suitable. Before making choice of a particular machine, the whole circumstances under which it is to work must be carefully studied, and a suitable

type of machine introduced to meet the particular requirements at the colliery, for a machine which gives excellent results in one seam might prove a failure in another. The question of the best machine for any given seam is often only solved by actual trial and experience, as different seams may require different types of machines to get the best results under the prevailing conditions. Previous to selecting a given type or class of machine, it would be well for the interested parties to visit a number of collieries where different kinds of coal-cutters are in use, and to compare carefully the conditions under which they are working with the conditions in the seam or seams which it is proposed to cut by machinery.

Conditions suitable for Coal-cutting Machines.—As we have pointed out, the question of whether a certain seam is suitable for being cut by machinery cannot be answered off-hand. Some seams are more suitable for cutting by machines than others, while others may not be suitable at all to work with coal-cutters; for it must be distinctly understood, as things at present stand in Great Britain, all seams are not suitable for mechanical coal-cutting. Under certain circumstances coal can be got in better condition and at a cheaper rate by machinery than by hand labour. These circumstances may be stated thus:—

1. When the coal or underclay is of a hard nature and not easily holed by hand labour.
2. When the seam is under 3 ft. 6 in. in thickness.
3. When there is a fairly good roof and floor, the latter being fairly level.
4. When the wages of miners are high.

While we have stated that seams under 3 ft. 6 in. may be worked more profitably by machines than by hand labour, we would not say that seams of greater thickness cannot be worked to advantage with coal-cutters, for as a matter of fact seams thicker than $3\frac{1}{2}$ ft. are being now worked at less cost with machines than by hand labour. What we do state is that coal-cutting machines can be used more effectively and more profitably in seams under $3\frac{1}{2}$ ft. thick than in those of greater thickness. Regarding the condition of the roof and floor, there can be no doubt that machines require that both should be fairly good, especially the roof, for with a bad roof a considerable amount of extra labour and expense is involved. While, however, a good roof is a great advantage, it does not follow that with a bad roof it is impossible to cut by machines, for if due care is taken in propping and strapping, the difficulties of a bad roof may, in a large measure, be overcome. The floor should be fairly even, for it is not an easy matter to get the machine to follow the inequalities which are present in some seams. Seams that are much cut up by hitches, faults, and wants are not, as a rule, suitable for coal-cutting machines, unless where machines of the percussive type are used. Then as to the inclination of the seam, there can be no doubt that the best results are attained in seams that are level or with a low

inclination, but if the workings are properly laid out for mechanical cutting, the gradient is not a great difficulty; indeed, machines will be found at work cutting in seams with gradients up to 1 in 3 or 1 in 2. In cutting down such steep gradients the machine will require to be controlled by a back balance or a brake applied to the wheels of the machine.

Timbering at the Machine Face.—Timbering in a machine face is, in most cases, simple enough, especially when the roof is fairly good. The props are usually set up in rows, parallel to the roadway, at regular intervals along the face, a space of $3\frac{1}{2}$ or 4 ft. being left between the front row and the coal to enable the machine to pass along. If the roof is of such a nature that it would be dangerous to leave such a width unsupported, straps are wedged or notched into the coal-face at one end, and the other end supported by a prop (fig. 147). When the machine passes forward, other props are set up to the end of the strap supported by the coal. Sometimes light steel bars are used in preference to the wooden straps.

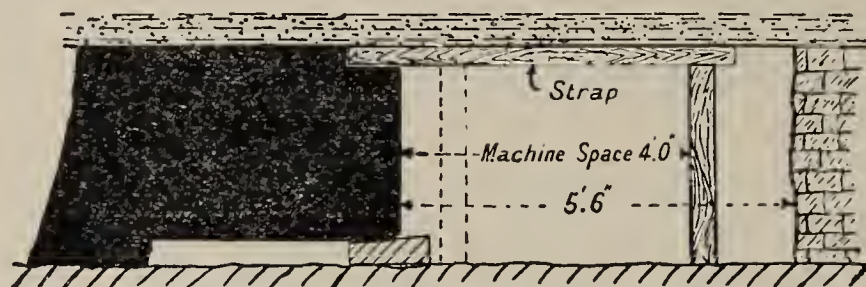


FIG. 147.—Method of supporting roof in machine wall.

When the roof is bad and the coal cannot be relied on to give the necessary support to the end of the cross-straps resting upon it, a longitudinal bearer set underneath and at right angles to the cross-straps, and parallel to the face, will require to be used. This longitudinal bearer may be of planking, or it may be a light steel channel girder. This method has been used at some collieries and found to answer the purpose well. The girders are used in 15 ft. lengths and weigh 25 lbs. per yard.

Props have to be set at regular intervals to these longitudinal bearers before the coal is cut, and as the machine advances they are removed one by one and re-set behind it. By adopting this system mechanical cutting can be carried on under roofs of a very bad nature. Chocks are also used in some workings, in addition to the props, for supporting the roof.

Labour for Machine Cutting.—To work a coal-cutter, usually three men are employed if run on rails, and two if run on skids; one, a principal man, to drive the machine, shovel the small coal out from the cutting wheel, sprag the coal, and securing the roof behind the machine, and also, if rails are used, lift the rails behind and pass them up to front of the machine. This man ought to have

sufficient mechanical knowledge to enable him to do any small repairs in the event of a stoppage or breakdown, and to get a man of this description, sufficient inducement, in the shape of good wages, will require to be offered. Other two men are employed, laying the road, putting in the necessary struts, attending to the bridle and hauling rope, and looking after any other work required during cutting. If the roof is bad, sometimes a fourth man is set specially apart to look after the timbering of the face.

Apart from the machine, another set of men are employed during the daytime. These are distributed as follows:—(a) A set of men for taking out the sprags, bringing down the coal, and backing it out, these being generally called hewers or ‘strippers’; (b) a set of men to get the coal out to the road head (*i.e.* where the tubs are not taken along the face) and filling it into the tubs; (c) a set of drawers who draw the tubs out to horse or mechanical haulage. Where the tubs are lifted from the face by horse haulage this last set of men will, of course, not be required. It will also be found advantageous to employ a specially appointed man to supervise and be responsible for the whole of the work. Regarding the payment of all these sets of men, machinemen included, it will be found, as a rule, most satisfactory to pay them on the piecework system, *i.e.* pay them a tonnage rate for the cutting and getting of the coal. This is generally found to be the most satisfactory method, as it encourages the men to get the maximum length and depth cut in each shift. In some collieries the whole of the work is done by contract, but in most of the districts the miners’ Unions are averse to this system, and give much trouble when it is attempted. It need hardly be said that the best class of men available should be employed for machine cutting, and especially for doing the actual cutting.

Motive Power for Machines.—The motive power employed for driving machines may be either compressed air or electricity. Of a total of 1959 machines employed in British mines in 1910, 1087, or 55·5 per cent., used compressed air as the motive power, and 872, or 44·5 per cent., were electrically driven. But of these numbers 556 were percussive machines, which are essentially compressed air machines and not suited for electric drive. Deducting these, it leaves a total of 1403 which could be operated by either power, and of these approximately 70 per cent. were driven by electricity and 30 per cent. by compressed air. Each of the systems have their advantages and disadvantages.

Mr Garforth* states that “after twenty-eight years’ experience with compressed air motors in underground work, it is the only power that can be safely used in certain deep gaseous mines, where the natural difficulties are quite sufficient without the introduction of artificial dangers. On the other hand, there are many mines

* *Trans. Inst. Min. Engs.*, vol. xxiii. p. 338.

which can be worked safely, and with advantage, by electricity." There can be no doubt that many mining engineers and colliery managers are averse to the use of electrical plant in fiery mines, but we think the dangers connected with the employment of electricity under such circumstances have been greatly exaggerated. Even in what are termed fiery mines it is seldom that an explosive atmosphere exists at the coal face where the machines are at work, and where sparking would be likely to take place. With the newer types of enclosed and gas-tight motors sparking is reduced to a minimum, and seldom occurs. Wherever electricity has been used the consensus of opinion is, so far, in its favour. A very large number of our mines are non-fiery, and in these there is no reason why it should not be used more extensively. In Scotland, where machine mining is largely employed, the total number of machines in 1910 being 581, the electrically driven machines were 425, or 73·2 per cent., and the compressed air machines 256, or 26·8 per cent. With electrical coal-cutters, however, it requires more highly trained men to operate the machines than when they are driven by compressed air.

One great objection to air-driven machines is the amount of noise made by the exhaust and the clouds of dust which it raises in dry and dusty mines. This is due to the fact that with some of the machines, such as the Gillott, and others, the exhaust blows directly on the floor. The noise and dust can be overcome to a certain extent in such cases by fixing a baffle plate under the exhaust. There is also the further disadvantage of low efficiency obtained, 30 to 40 per cent. compared with 50 to 65 per cent. for electricity. Compressed air has the advantage of being used with perfect safety, and any disarrangement can be easily repaired. On the other hand, the higher efficiency, lower cost for extensions, and greater ease with which extensions can be made with electricity, render it particularly suitable for coal-cutting machinery, and there can be little doubt but that, in the near future, it will be more extensively employed for such work, and will gradually displace compressed air as a motive power, especially where there is not much fire-damp to contend with.

Cost of Coal-cutting Installations.—The cost of coal-cutting machines varies according to the type and motive power used. Disc machines driven by compressed air cost from £250 to £300 each; machines driven by electricity cost from £300 to £450 each. The cost of the generating plant on the surface will vary according to the number of machines which it is intended to operate; the cost of a generator for a single machine will be greater in proportion than if a number of machines are at work. When more than three machines are at work the cost of the generating plant may be set down approximately at £1000 per machine if compressed air is the motive power used; electrical plant will average somewhat more than this,

probably £1100 to £1200 per machine. In these estimates the cost of piping or cables is not taken into account.

Power required to drive Coal-cutters.—The power necessary to work coal-cutters will vary greatly according to the class and type of machine, nature of holing, rate of feed, and how machine is handled by attendants. Compressed air disc machines take 15 to 20 horse-power under normal conditions, bar machines 12 to 18 horse-power, and chain machines a little less, but overloads are frequent, *e.g.* cutting extra hard piece of coal or floor, or hard nodules may be met with, and ample power should be provided for such contingencies. Electrically driven machines usually take a little more power, owing to the greater speed of cutting. In the newer types of electrical machines the best makers are now providing them with motors capable of developing 30 to 40 horse-power. Allowing for losses between the generator at the surface and the motor on the machine underground, and other contingencies, it would be wise to calculate for a plant to generate 50 horse-power for one machine, 80 horse-power for two machines, and 110 horse-power for three machines. When more than three machines are employed the horse-power of the power plant need not exceed the total power of the coal-cutter motors, as there is not the same probability of all the machines being operated at the same time, and still less probability of them being subject to overload simultaneously.

Machines on Wheels and Skids.—Until a few years ago all coal-cutting machines were mounted on wheels to enable them to run on rails along the face, but now the greater number of machines are mounted on skids suitable for sliding along the floor of the seam. Both systems have their advocates, and both systems may be equally good, given suitable conditions. If the floor is firm and there is plenty of height, one system is as good as the other, but with a soft floor better results are obtained with rails, unless the seam is too thin to pass the machine effectively. When skid machines are used the labour and expense of laying the rail-track is saved. With disc machines running on rails the depth of cut may be varied to suit conditions, but with skids the machine has a tendency to pull itself into the cut, and there is no easy means of holding it back or to take less than the full depth if desired. Chain and bar machines are better fitted with skids, so as to enable them to cut on the floor level. Skid machines require a little more power than machines on wheels, to overcome the increased resistance, but as the rate of travel along the face is low, about 3 ft. per minute, the amount of extra power required is very small.

Depth of Cut.—This will vary with the thickness and nature of seam, and no definite rule can be laid down which would suit every seam. The depth of cut may vary from a minimum of 3 ft. to a maximum of 6 ft. The usual practice is to cut to a depth of 3 to $3\frac{1}{2}$ ft. in seams $1\frac{1}{2}$ to $2\frac{1}{2}$ or 3 ft. thick; seams $4\frac{1}{2}$ ft. thick, depth of

cut may be 4 to $4\frac{1}{2}$ ft.; 5 ft. thick, under-cut 5 to $5\frac{1}{2}$ ft.; 6 ft. thick, under-cut 6 ft. With seams 3 to 5 ft. thick, an under-cut of $3\frac{3}{4}$ ft. has shown itself to be most profitable; with friable or leafy seams, where the coal is liable to fall, a cut of about 3 ft. is found to be most suitable. In strong seams which require to be blasted down, deeper cutting is usually more successful. In seams with a good parting next the roof, it is better to cut to a depth sufficient to cause the coal to break off with the roof weight rather than resort to blasting. In thin seams, $1\frac{1}{2}$ to 2 ft. thick, little advantage is to be gained by deep cutting, as the coal will sit down, instead of falling over, making it difficult to break up and fill.

Work done by Machines.—This is very variable, and depends on various circumstances, such as the nature of the holing, gradient of seam, power supply, facilities available, and capabilities of the machine attendants. In longwall work a machine ought to cut a minimum of 100 yds., $3\frac{1}{2}$ ft. deep, under unfavourable conditions, and 150 to 200 yds. under favourable conditions, in eight to nine hours. In exceptional cases as much as 240 to 280 yds. have been cut per shift of eight hours, but this is hardly representative of ordinary everyday working. In flat seams the cutting rate is usually higher than in seams with a high gradient, and in the latter it has been found that the rate of cutting is greater going up-hill. In a colliery in Fifeshire a disc machine working in a seam 2 ft. thick, gradient 1 in 3, cut on an average 105 yds. per eight hours, depth of under-cut 3 ft.; colliery in Lanarkshire, seam 2 ft. 3 in. thick, workings almost level, depth of cut 3 ft., length cut varied from 120 to 140 yds. per nine hours; another colliery in Lanarkshire, seam 2 ft. 7 in. thick, fireclay holing, gradient 1 in 15 to 1 in 20, depth of cut $3\frac{1}{2}$ ft., bar machine cut 140 to 150 yds. in twelve to fourteen hours. Stated in tons, the output per machine will vary from 70 to 120 tons per shift, according to thickness of seam.

Cutting by Percussive Machines.—At Neilsland Colliery,* Hamilton, Mr John Blake states that with a ‘Little Hardy’ machine, three places, 11 ft. wide, could be cut to a depth of 3 ft. in an eight-hour shift; and with a ‘Siskol’ machine nine places, 10 ft. wide $\times$ 3 ft. deep, were cut in sixteen hours. Mr Owen Hughes† states that at a colliery under his charge one Ingersoll machine could under-cut on an average four places 10 to 12 ft. wide $\times$ $3\frac{1}{2}$ ft. deep in nine hours, including meal time and ‘flitting’ of machine.

Laying out the Face for Machine Cutting.—There is hardly anything connected with machine cutting of more importance than the laying out of the face in a proper and systematic manner, for unless this is carefully attended to it may mean complete failure from the start. The first desideratum in a machine face is straight-

* *Trans. Scottish Inst. Min. Students*, vol. ii. p. 29.

† *Trans. Manchester Geol. and Min. Soc.*, vol. xxvii. p. 182.

ness. Unless a machine has a straight line of face to work upon, the efficiency of coal-cutters will be greatly reduced, and success almost impossible. Many failures in the introduction of mechanical cutting have been due solely to setting the machine to work on a crooked face, and it is therefore imperative that the face should be got into a straight line before starting the machine on it. It has been amply demonstrated that with a straight line of face the coal breaks more easily and evenly, and falls more readily, while at the same time the roof also subsides more evenly with regular lines of break. With a straight line of face a more systematic method of timbering can be carried out, ensuring greater safety to the workmen. A system which Mr Garforth has adopted for thicker seams at Altofts Colliery, Normanton, is to divide the workings into panels, driving the narrow work by hand labour in the ordinary way, and then cutting the coal back from the boundary homewards by machines. The advantage of this system is, that all the roads are in solid coal, and they are therefore easier and less expensive to keep than if carried through the waste or gob. All collieries, however, cannot be laid out on a stereotyped plan, and each one must be dealt with according to the circumstances under which the machine will have to work, and according to the exigencies of the seam to be won.

Commencing a Cut.—In starting a cut two methods are employed. By one method a narrow stall is cut at the end of the face, the size of the stall being of somewhat greater length than the machine, and of a depth slightly greater than the diameter of the cutter. This stall is cut either by hand labour or by a heading machine, such as the Ingersoll-Sergeant percussive machine. In the other method the machine opens up its own cut, which is accomplished by setting the machine at an angle with the face and gradually working the wheel or cutter into the coal until the requisite depth is attained; or sometimes the face is so laid off that the machine starts at nothing and gradually works into the required depth. The first system is the one that is most generally adopted, and has been found to be most satisfactory, as the line of face can be kept much straighter. The cutting of the stall, when done by hand labour, of course entails additional cost, as the stall has usually to be cut in solid coal, or with at least one side solid. A similar stall should also be made at the other end of the face, so that the machine can finish in the open.

Method of Cutting.—Where cutting is being done on a longwall face, two methods of cutting or opening out the face are practised, viz., continuous cutting and cutting forwards and backwards. In the first system the machine always cuts in one direction along the face. When the end of the cut is reached the machine is removed or 'flitted' back to the point from where it commenced the cutting. In the other system the machine cuts along the face in one direction

till the end of the cut is reached, and then, without removing the machine, cuts back in the opposite direction.

Both methods give good results under suitable conditions. Where there is a long length of face available, say 1800 to 3000 or 4000 ft., the first system is undoubtedly the best, as the flitting of the machine will not require to be so frequent, which is a saving of time and expense, as each flitting of the machine will take three or four men two to four hours, according to length of cut. This system has also the advantage of allowing several machines to be set to work on the same face, the one behind the other.

For thin seams and shorter lengths of face, 300 to 400 ft., the second system is often adopted, and gives good results. This method saves the cost and time of flitting, which, with a machine measuring 7 to 9 ft. in length, and weighing anything between 30 and 50 cwt., is often not easy of accomplishment in low and narrow roads, and more especially if the gradient is steep. The principal drawbacks to the cutting forward and backward system are that in the event of the men not coming to work, or by a want of tubs or waggons, the face is not cleared of cut coal during the day shift, the machine cannot get to work at night, and the following day's work would be lost. The same thing would happen should the machine break down and be prevented from finishing the cut. Where this system is adopted, the length of face to be cut should be well within the capacity of the machine, otherwise much annoyance may ensue and the output be curtailed. In the continuous cutting system the chief disadvantage is the cost and trouble of flitting the machine along the roads from one end of the face to the other.

Cost of Machine Cutting.—This is a very difficult matter to determine, as so many conflicting statements have been published comparing the cost of cutting by machines with hand labour. Some authorities put the saving in cost with machine cutting at 6d. to 1s. per ton compared with hand labour, while others have shown that the saving in cost is little more than 1d. or 2d. per ton; sometimes no saving at all, or an actual loss, may be shown. Both statements may be correct, for where the larger figure of 1s. per ton has been quoted, no account has probably been taken of increased outlay of capital, interest, depreciation of plant, upkeep and repairs, cost of shifting machine, cost of stores, cost of cutting stall for machine, cost of extra timber in some cases, etc.; all of which have to be carefully considered and calculated in order to make a fair comparison between the two methods of cutting. Where the smaller figure of 1d. per ton is given, or an actual loss shown, that only goes to prove the seam was not suitable for machine cutting, and that the whole conditions and details involved in the introduction of coal-cutters had not been sufficiently or carefully thought out. In many cases machines have been introduced where they were not really required, as the coal could be got at an equal cost by hand labour. In

most seams, however, which are suitable for machine cutting, some saving can be generally effected; the reduction in cost being usually greatest in thin seams— $2\frac{1}{2}$ ft. thick and under. There can be no doubt that many thin seams are at present being worked—and more will have to be worked in future—by coal-cutting machines which it would not be possible to work at a profit by hand labour.

The actual cost in the working is usually made up on three items, viz., cutting the coal, stripping and filling, and maintainance. The cost of cutting is very variable, depending on the conditions of the seam and the facilities for cutting. This item may vary from 3d. to 9d. or 1s. per ton, according to the thickness of the seam and under average conditions; for a seam $2\frac{1}{2}$ ft. thick an average rate would be 4d. to 6d. per ton, stripping and filling 10d. to 1s., and maintenance 3d. to 6d.

To give a fair idea of the cost of machine cutting compared with hand labour, all the items should be taken into consideration. The following are the approximate costs per ton of coal got for some of the more important of these items:—Motive power for driving machines, 1d. to 2d.; upkeep and repairs of generating plant, machines, and depreciation, 2d. to 3d.; stores, $\frac{1}{2}$ d. to 1d.; cutting and shearing stall for machine at each end of wall, 1d. to 3d.; flitting machine where continuous cutting is adopted, 1d. to 2d.; extra timber frequently required, 1d. to 2d.; interest on capital outlay, $\frac{1}{2}$ d. to $\frac{3}{4}$ d.

Advantages of Coal-cutting Machines.—The employment of coal-cutters in the majority of cases effects a saving in cost of production, which may arise from various causes. There are also other advantages to be derived from the introduction of coal-cutters, which may be briefly stated as follows:—(1) An increased percentage of large coal is obtained, and the coal is got in a firmer and better condition; (2) a more regular line of face is obtained, which facilitates ventilation and leads to more regular and systematic timbering, and the weight being more regular the roof can be more easily kept up. The greater rapidity of working also tends to keep down the cost of repairs, and causes less damage to overlying seams and the surface, the subsidence being more even; (3) the regular and more systematic working tends to increase the safety of the workmen; (4) seams which, either because of their thinness or hardness, or both, could not be worked at all, or could only be worked at a profit in good times, can be profitably worked by machines; (5) holing is less frequently done in the coal, and when it is, there is much less small made than in the case of holing or under-cutting by hand; (6) the output is increased, and is more regular, and the work is more easily superintended; (7) less explosives are used in getting down the coal—in some cases none. Generally, machine work is less costly than hand work, especially in thin seams. From the point of view of the men the work is safer and easier, and the wages

are better. The importance of lightening the labour of the men will probably be more appreciated as the working places become deeper and the temperature higher.*

In addition to these advantages it may be pointed out that a higher average price is obtained for machine cut coal, owing to the high percentage of large coal; that the upkeep for roadways is less as the output is obtained from a smaller area; that there is a greater possibility of producing the full daily output that the colliery is capable of dealing with, thus keeping down cost charges; and that there is a more rapid development of the mine and earlier return on capital outlay. While these are some of the advantages that may be claimed for coal-cutting by machinery, it must be distinctly understood that they do not invariably apply in every case. The natural conditions in the seam are always important factors in determining whether the use of machinery is advisable or not.

General Remarks.—To be successful with coal-cutting machinery the work must have constant and close supervision, for unless this is done failure is almost sure to be the result. Machinery of the best quality and workmanship should be employed, and the coal-cutter most suitable for the conditions under which it has to work should be selected. A spare machine should be kept in reserve, especially where a number of machines are at work, and as many as possible of the working parts of the machine kept in duplicate, so that repairs can be carried out quickly; otherwise much expense may be incurred owing to the loss of time and work. When it is determined to introduce machines, the face ought to be laid out to suit them, and the cutting reduced to a regular system. The machine-men should be carefully selected, and every encouragement given to suitable men to remain, as a frequent change of men for such work often means decreased efficiency and much worry. Where a number of machines are at work a specially trained man should be appointed to have charge and supervise the working of the machines, and be responsible for the work being carried out in an efficient manner.

The following rules have been suggested for the guidance of those who purpose employing coal-cutting machines † :—

(1) The working face shall be cleared of all coal and dirt at the end of the day shift, and cross-trees or straps shall be needled into the face when the roof requires support nearer than $3\frac{1}{2}$ ft. from the face.

(2) The places at each end of the face shall be left with the pack or building not nearer than 6 ft. from the face, so as to allow the men passage around the machine to get it into its fresh position.

(3) Pack dirt and roadhead buildings must be kept 5 ft. back from the face.

* *Final Report of the Royal Commission on Coal Supplies*, Part I. p. 7.

† *Trans. Inst. Min. Engs.*, vol. xxv. p. 115, and *Report of Committee upon Mechanical Coal-cutting*, *N. of England Min. and Mech. Engs.*, p. 33.

SUGGESTED FORM OF DAILY REPORT.

Date, _____ 19

Coal-Cutter Report, _____ Pit.

Machine No. _____ in _____ seam.

Machineman left bank _____ Commenced cutting _____

Returned to bank _____ Finished cutting _____

Time bank to bank _____ hours. Time cutting _____ hours.

Distance cut _____ yards. Average depth of undercut _____ ft. _____ in.

Position of machine at starting _____

do. do. stopping _____

Work done before commencing to cut _____

Has the face been in good condition for machine? _____

How many sets of cutters were used? _____

Has there been any shortage of cutters, oil, or timber? _____

State duration and reason of any stoppages _____

Work done after finishing cut _____

What length of face is ready for next shift? _____ yards.

What work is required before machine can cut again? _____

What material is required for next cutting shift? _____

Is there anything about cables, machines, or fittings requiring attention? _____

Signed _____

(4) All timber required for spragging coal, packing, and fixing sleepers, and for roof props behind the machine, must be at the roadhead ready for distribution along the faces before the start of the cutting shift.

(5) Machinemen and attendants must at all times pay strict attention to the condition of the roof, and erect timber to ensure their safety.

(6) The machine road must be well laid and properly stayed to keep it from shifting.

(7) Care must be taken that the strain is put on the machine gradually, and that the bearings are well oiled.

(8) The cutting-tools must be changed when blunt, and an inadequate supply must be reported at once.

(9) Bad joints in the hose-couplings and on the pipe-lines, and defective hose or cables, must be reported at once.

CHAPTER VII.

TRANSMISSION OF POWER.

IN the working of collieries some source of motive power must be provided for operating the plant, both on the surface and in the underground workings. No matter what the power may be, it has to be conveyed or transmitted from one point to another. Sometimes these points may be quite close together ; at other times they may be widely separated, and the power required transmitted long distances.

Power may be transmitted in the following ways :—

- (1) Steam in pipes.
- (2) Compressed air in pipes.
- (3) Water under high pressure in pipes.
- (4) Electricity along metal conductors.
- (5) Ropes.
- (6) Rods of wood or iron.

Steam.—Whatever power is employed, steam is the initial power used to generate compressed air, electricity, or to work ropes or rods, and frequently to raise water to the necessary pressure. At one time steam was extensively employed—in fact, was practically the only power used—for operating machinery on the surface and underground, but in modern mining practice electricity is now very largely employed for all kinds of work, and in a lesser degree compressed air for operating certain kinds of plant underground. Where a colliery is comprised of a group of shafts it is becoming the practice to some extent to erect a central generating plant from which electricity is transmitted to the various shafts, and the whole of the work done by this power. Or if the colliery is situated in an area supplied by a power supply company, the current is taken direct from their station instead of generating it, thus saving the initial cost of a steam power plant. For winding engines and for hauling, pumping, and other operations, where the plant is situated on the surface and quite close to the point of generation, steam is probably the cheapest and easiest method of transmitting power. Its simplicity of generation and transmission and convenience are greatly in its favour, but in many instances it has been used, especially underground, where the results obtained could not justify the employment of steam power.

If steam is to be used its proper sphere is the surface, or at most at the bottom of the shaft. The writer would hesitate to recommend the employment of steam power in underground workings under almost any conditions, not because of inefficiency, but because of the disadvantages of transmitting such a power along underground roadways which are frequently cramped and ill-suited for such a purpose.

Loss in Transmission.—The principal losses in conveying steam to underground workings are due to condensation, radiation, and leakage, and to minimise this, high-pressure steam should be used in supply pipes of proper dimensions. At Bockwa Colliery, Germany, the pumping engine is placed at a depth of 590 ft. from the surface, and is designed to raise 3300 gallons per minute. The steam pressure generated at the boilers on the surface is 150 lbs. per sq. in., the steam being conveyed to the engines in two columns of wrought-iron pipes each 4 in. diameter. These pipes are jacketed with a preparation composed of cork, bound round with zinc wire, and coated with gypsum, jute, and asphalt, and an outside sheathing of canvas, painted with Stockholm tar. In this way the loss by condensation is reduced to a minimum.

At Niddrie Colliery, near Edinburgh, steam is carried down a steep gradient, the inclination varying from 55° to 75° , for a distance of 3000 ft. to work winding and hauling engines and pumps placed at various parts of the workings. With a steam pressure at the surface of 45 lbs. per sq. in., the gauges in the mine recorded from 41 to 42 lbs. per sq. in., or a loss of only 3 to 4 lbs., which is an exceedingly small percentage of loss when the distance the steam is carried is taken into account. But while the loss in pressure was small, the loss in other ways was shown to be fairly large, owing, no doubt, to the intermittent working of the haulage and winding engines. There are twelve boilers, one or two of which are, however, always off for repairs, cleaning, etc. When the whole plant is in operation, 3400 gallons of water are evaporated per hour. When all the machinery is idle, but steam is on, and cylinders, steam pipes, and relief-valves open, it requires 1028 gallons of water per hour to maintain the pressure and keep the water in the boilers up to a fixed point. This shows a dead loss of 30 per cent., most of which takes place in the underground piping and machinery.*

The loss in this case can be calculated as follows:—

Length of piping, ...	3000 ft.
Steam pressure at surface, ...	45 lbs. per sq.in.
Water evaporated per hour with machinery all running, ...	3400 galls.
Water evaporated per hour with machinery idle, but with steam on, relief-valves open, and water in boilers maintained at fixed level, ...	1028 galls.

$$\text{Then percentage loss} = \frac{1028 \times 100}{3400} = 32.23.$$

* *Trans. Inst. Min. Engs.*, vol. xv. p. 264.

In this case the loss due to drop of pressure was only 3 to 4 lbs. per sq. in., which is very small, considering the length the steam was carried, but the loss due to condensation, radiation, and leakage was large.

Where steam is used underground the pipes should be carefully fixed in the shaft and properly guided, the pipes being led between two rollers, which are more suitable than rigid guides. Some arrangement should also be made to cut off steam by an automatic runaway valve, in the event of a pipe bursting, as serious danger would result from the escaping of a large volume of steam into the workings before the supply could be shut off from the boilers at the surface. Great care should also be taken in making the connections, as leaking joints are a source of much loss of power and often of danger. Joints made with asbestos rings are preferable to those made with india-rubber.

The disadvantages attending the employment of steam underground are :—

Loss of pressure due to condensation and leaking joints.

The danger of the sudden bursting of a pipe in the workings, or the blowing out of a joint.

The discomfort from the heat due to the great increase in temperature of the air in the mine, particularly in narrow, confined roads.

The bad effects of the moisture, due to the steam, on the roof and timber.

The difficulty of dealing with the exhaust steam, particularly if the engine is at a long distance from the shaft.

The danger of fire when pipes are led in confined places.

There can be no doubt but that the losses in steam generation and transmission could be greatly modified if proper thought and care were taken in selecting the most suitable plant. In the *Final Report of the Royal Commission on Coal Supplies*, the Commissioners state that "it is beyond question that collieries are extremely wasteful in the consumption of coal [for steam generation]; no doubt to a large extent because of the small value of the fuel used, which is generally of inferior quality." The average consumption of coal for steam-raising purposes is 7 or 8 per cent. of the total output, and at some collieries as high as 10 to 12 per cent. If the whole of the plant of the collieries in the United Kingdom were modern plant of the best description, the consumption of coal for steam power should be reduced to one-half. Economies could be made by a more extended use of condensing and expansively worked engines, economisers, automatic stokers, mechanical draught, the super-heating of steam, and heating of the fuel water by the exhaust steam.

Utilisation of Exhaust Steam.—Within recent years the exhaust steam, which had hitherto been allowed to exhaust into the atmosphere, has been utilised to generate power. Professor A. Rateau * says that "the system consists essentially in accumulating

* *Trans. Inst. Min. Engs.*, vol. xxiv. p. 323.

in an appropriate apparatus the non-continuous exhaust steam of intermittently running engines, so as to obtain from that apparatus a regular flow of steam, which may be subsequently utilised in a secondary engine (preferably a turbine) provided with a condenser. This arrangement thus enables intermittent engines to take full advantage of condensing arrangements, and to store up mechanically the considerable stock of energy which these engines have hitherto freely exhausted into the surrounding atmosphere." . . . "To utilise the supplementary energy made available by the application of condensation it was, of course, necessary to transform this energy into mechanical work by means of a suitable engine, and that is the purpose of the steam turbine which is introduced between the accumulator and the condenser. The power developed by the turbine is henceforward obtainable without any additional expenditure of fuel, and may be practically reckoned as all gain for working purposes." These sentences of Professor Rateau explain clearly the idea underlying the utilisation of waste steam. At almost every colliery there are intermittent working engines, such as winding engines, and engines working main and tail-and-tail rope haulage, and in addition there may be steam exhausted from other engines at the surface. The idea, then, is to utilise this exhaust steam to drive a turbine, which in turn is connected, usually direct, to an electric generator, compressed air generator, or hydraulic power generator, from which motive power is taken to drive other plant, either on the surface or underground. The turbine employed may be either of the low-pressure type, such as Parsons' turbine, or mixed-pressure type, such as Zoelly turbine. The latter is the type which has been found most suitable, and is most largely used, for colliery work. In the mixed-pressure type both exhaust and live steam (*i.e.* steam at boiler pressure) may be used, the idea being to use the live steam when the winding engines are not working and no exhaust steam is then available.

For constructional details of turbines we must refer the reader to the various text-books dealing with the subject. There can be no doubt that under certain circumstances, such as where there is a large amount of exhaust steam available, the installation of turbine sets for generating power will result in economy. Some large plants have been already installed at collieries, both in England and Scotland, to generate electric power, but several have also been installed to generate compressed air. A plant of this latter type, with a turbo-compressor, has recently been installed at New Hucknall Colliery, designed to supply 7500 cub. ft. of free air per minute at a pressure of 80 lbs. per sq. in. The speed is 4000 revolutions per minute. Each 1 lb. of low-pressure steam (exhaust) will compress to 80 lbs. and deliver 13 cub. ft. of air per minute, and each 1 lb. of high-pressure steam will likewise deliver 20 cub. ft.

Compressed Air.—As a medium for the transmission of power

to machinery in underground workings, compressed air is very suitable. The ease with which it can be conveyed to distant parts of the workings, the absence of heat in the pipes, and the beneficial effect it exerts on the ventilation, make it invaluable for certain classes of work, such as driving coal-cutting machines, rock drills, hauling engines, and pumps; indeed, for rock drills no other motive power presents anything like the same advantages. Owing, probably, to the large first cost for compressing machinery, and the low efficiency obtainable—25 to 40 per cent.;—it has never been

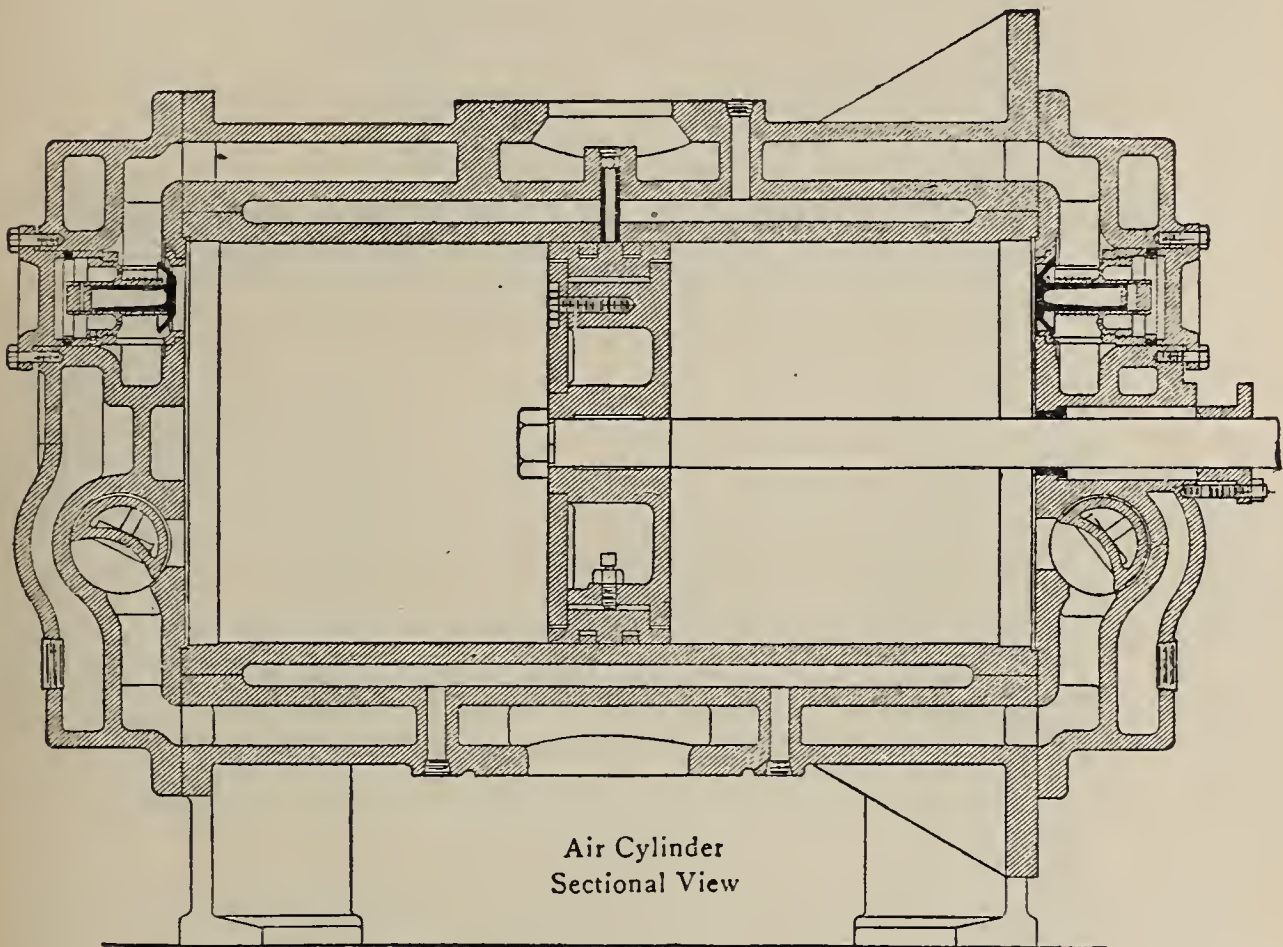


FIG. 148.—Sectional view of air cylinder, Sullivan compressor.

used to the extent it might otherwise have been had the efficiency been higher, because once the plant is put down, the working expenses are not excessive compared with an ordinary steam engine.

In fiery or dusty mines compressed air is undoubtedly the safest method of transmitting power, and at quite a number of collieries where these conditions prevail electricity has been discarded and compressed air installations substituted. With the more stringent regulations of the Coal Mines Act, 1911, there seems to be no doubt but that compressed air will be more largely employed in the future for underground mechanical work.

The cost of a compressed air plant, with coupled horizontal steam cylinders 22 in. diameter and 3 ft. stroke, and air cylinders 24 in. diameter and 3 ft. stroke, with steam boilers, air-receiver, main pipes,

etc., complete, should not be more than £3000 or £3500, which would include its erection and preparing foundations.

Air compressors are simply force pumps in which the air is drawn into the air cylinder by an inlet valve, and is compressed and forced through an outlet valve into an air receiver, from which the supply is drawn for the underground workings. The usual arrangement is to have a pair of steam cylinders placed horizontally and coupled direct, with a fly-wheel in the centre. The piston-rods are continued through the cylinder, and connected to two air cylinders also placed horizontally and in a direct line with the steam cylinders.

Compressors are usually of three classes: (1) Dry compressors; (2) wet compressors; (3) injection or spray compressors.

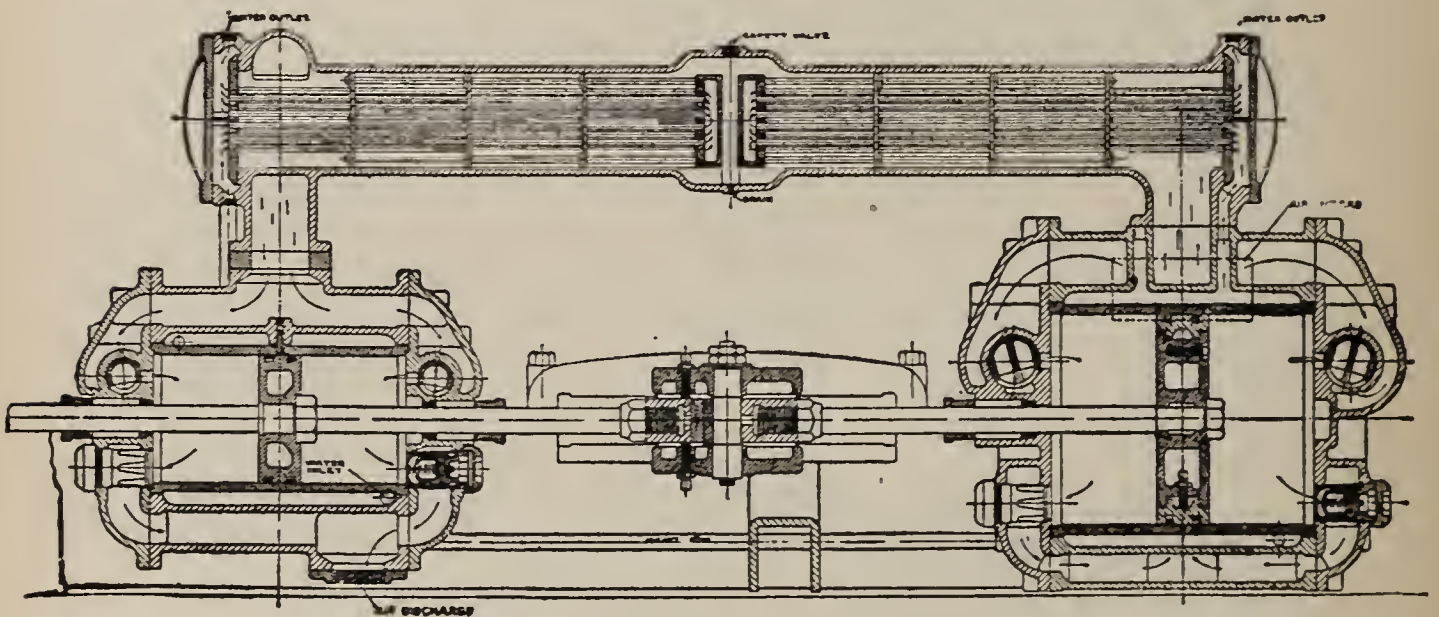


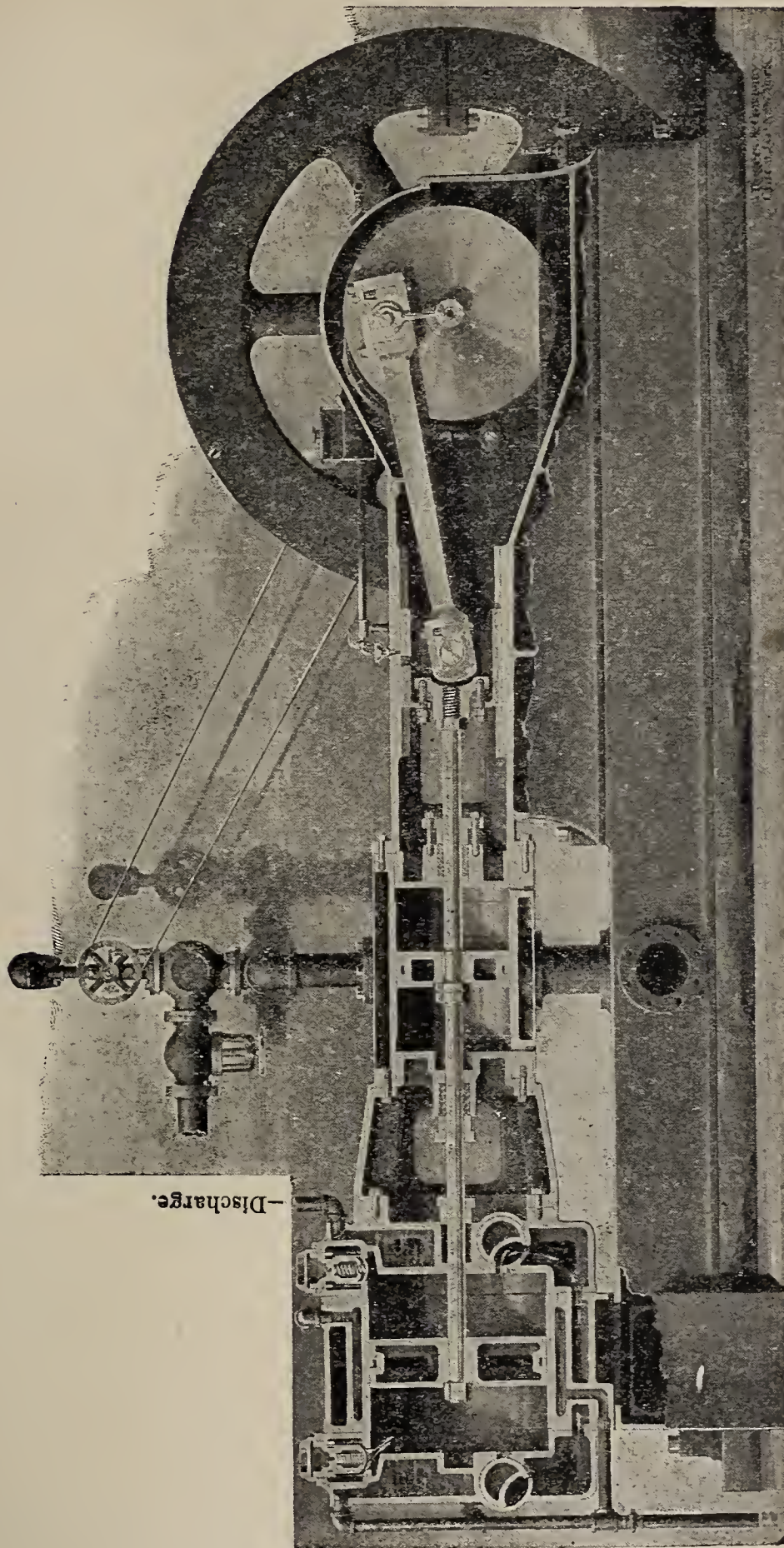
FIG. 149.—Sectional view of air cylinders and intercooler for two-stage compression.

Dry Compressors.—These are extensively used for colliery work, and give fairly good results if fitted with a water jacket and if the air pressure is not too high. A dry air compressor in its simplest form consists of an ordinary cylinder provided with a tight-fitting piston and suitable valves for admitting and delivering the air.

During the out-stroke of the piston, air rushes in and fills the cylinder through the valves; as soon as the piston commences the return stroke, the valves close and the air in the cylinder is compressed until it lifts the delivery valves, when it is forced out of the cylinder into the receiver. It will be seen from this that the action is exactly the same as in an ordinary pump. In Fowler's dry compressor air enters through the inlet valves as the piston moves forward; on the return stroke the air is delivered through the outlet valves, the cylinder being water-jacketed to keep it cool.

Another dry compressor of a quite different design is the Ingersoll-Sergeant compressor. The air does not enter the cylinder in the ordinary way, but first passes through a hollow tail-rod, the inlet

valves being placed inside the piston. The outlet valves are placed



—Inlet.

FIG. 150.—Sectional view of the Sullivan duplex steam-driven air compressor ; duplex steam and two-stage air cylinders.

at the ends of the cylinder, while a water jacket surrounds the cylinder, being kept cool by a continuous flow of cold water.

The advantages claimed for this compressor are :—

The air may be taken from whatever point is most favourable by dryness, reduced temperature, and freedom from dust.

The admission of air being through a single tube, creates a constant and uniform draught in one direction only, thus filling the cylinder at each stroke with air at atmospheric pressure.

The construction of the valves admits of a large area of inlet with but a small throw of valve, allowing the compressor to be run at a high speed.

There being no inlet valves in the ends of the air cylinders, the space otherwise occupied by these valves is filled with cold water, thus presenting a cooling surface to the compressed air near the end of the stroke when the air is hottest.

Wet Compressors.—Wet compressors are used to a considerable extent, but in this type a large volume of water has to be put in motion at each stroke. This absorbs a large amount of power without any recompense, and the engines must also move at a very slow speed, hence large engines involving increased expenditure must be used. Moreover, the column of water in the cylinder, by repeatedly moving backwards and forwards, soon gets hot, and loses the advantages which it is meant to confer, namely, to cool the air in the cylinder during compression. Where large engines are available, and only require to move at a slow speed—not more than 40 or 50 ft. per minute—wet compressors may be used sometimes with advantage.

Injection or Spray Compressors.—Instead of using a solid column of water inside the cylinder to absorb the heat from the air, a supply of water in the form of fine spray is injected into the cylinder. This cools the air, which is now carried forward into the receiver. In the Dubois-François compressor (fig. 151) the spray and wet compressor are combined. It consists of an ordinary pump having large chambers *aa* at each end of the cylinder. The air is admitted through the inlet valves *cc*, and delivered through the outlet valves *ee*. A fine spray of water is injected into the cylinder through the small pipe *d*, and coming in contact with the air, cools it. This type of compressor, like the ordinary wet machine, requires to be run at a low speed, less than 150 ft. per minute, which means that the compressor must be large for a given output of air. Dry compressors are usually run at a speed of 350 to 500 ft. per minute, hence the preference shown for this type of machine. The introduction of water into the cylinder in any form is a very defective method of cooling the air, and often does more harm than good, as it may corrode the walls of the cylinder.

The best method of cooling the air is that of *stage compression* with intermediate cooling. Compressors made on this principle are usually either double-stage or three-stage compressors. By compressing the air in two stages instead of in a single cylinder, an economy of 10 to 15 per cent. is obtained. This economy is due to

the more thorough cooling which it is possible to give the air during compression. In a single cylinder the final pressure, and, therefore, the final temperature, is reached in one cylinder. Water-jacketing of the cylinders and cylinder-heads affords the only means of cooling

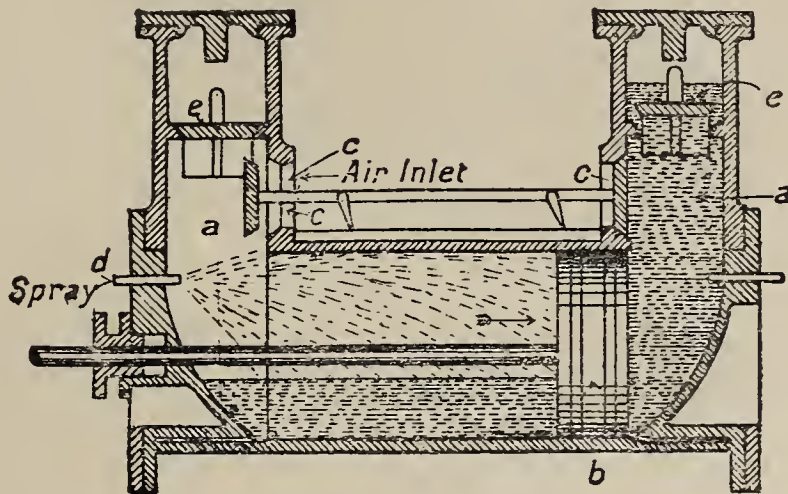


FIG. 151.—Spray compressor.

in this type. This method is of little effect, beyond keeping down the temperature of the cylinders. Two-stage compression divides the work to be done and the heat to be produced into two parts, thus greatly reducing the number of compressions to be made in each cylinder. The cylinder water jackets are thus more effective, and an opportunity is afforded to cool the air between the two stages of

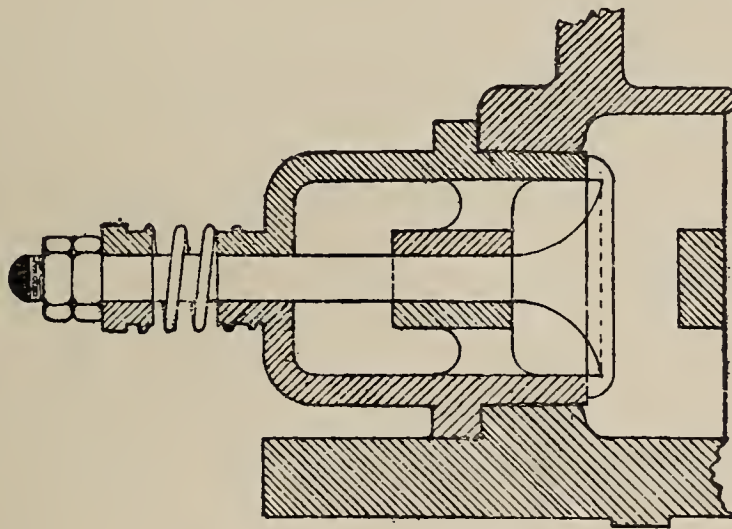


FIG. 152.—Mushroom valve.

compression. In this system the air is first subjected to low pressure in one cylinder, passed through an intermediate cooler, and thence to a second cylinder, where it is then compressed to a higher degree. The initial cost of either double-stage or triple-stage compressors is greater than for ordinary compression machinery, and stage compression is only suited for the production of pressures above 60 lbs. per sq. in. In such circumstances a great saving may be effected,

as 20 to 25 per cent. higher efficiency will be obtained than by ordinary methods of compressing air. In an installation at Paris, Professor Riedler combined stage compressing with spray injectors, which resulted in saving two-thirds to three-fourths of the work expended in uselessly heating the air, the loss due to heating only amounting to 12 per cent. The Riedler two-stage compressor in this case gave a useful effect of 77 per cent., allowing 0·85 as the mechanical efficiency. With stage compression the engines can be worked at a much higher speed, as they are better balanced. In a recent test on a Riedler two-stage installation the following results were obtained :—

	Per cent.
Efficiency of compressor, ...	95
Combined efficiency of engine and compressor, ...	77·8
Percentage loss due to friction of set, ...	9·2
„ „ compression, ...	4·5
„ „ clearance, leakage, and bends, ...	8·5
	100·0

There can be no doubt but that with properly constructed high-class plant, and the air compressed in two or three stages, far higher efficiencies and better results can be obtained than with the older type of single cylinder compressors.

The table given below shows the saving in power between compound and simple air compression :—

HORSE-POWER REQUIRED TO COMPRESS 100 CUB. FT. OF
FREE AIR TO VARIOUS PRESSURES.

Gauge Pressures.	Single Stage.	Two Stages.	Saving, Two-Stage over Single-Stage Compression.	
			Horse-Power.	Per cent.
40	10·25	...	...	...
45	11·10	...	...	...
50	11·87	...	...	...
55	12·60	...	...	...
60	13·30	11·71	1·59	11·95
65	13·97	12·29	1·68	12·03
70	14·61	12·83	1·78	12·18
75	15·22	13·33	1·89	12·42
80	15·81	13·80	2·01	12·71
85	16·38	14·24	2·14	13·06
90	16·93	14·64	2·29	13·53
95	17·46	15·00	2·46	14·09
100	17·99	15·34	2·65	14·73

Valves for Compressors.—A great many different types of valves are used in air cylinders, each claiming some merits of its own.

In the earlier types of compressors ordinary leather flap valves were used, but these did not give the best results, owing to the large amount of leakage they permitted. A common type of valve that is still used to a large extent is the mushroom valve, fitted to a spindle and kept up to its seat by means of springs. Fig. 152 shows this type. These valves are opened automatically by the pressure of the air against the action of the springs, which must be of sufficient strength to close them against currents of air impinging on them. They are often difficult to keep in proper adjustment. If they are heavy, the springs must help to overcome their inertia; the latter are apt to get slack themselves through wear, and the valves then oscillate violently when they are open, which not only restricts the area of opening, but destroys them speedily together with the seats. Again, if the springs are too tight undue resistance is offered to the passage of the air when passing into the compressor, and this resistance is, of course, a dead loss of energy. This loss can be overcome to a certain extent by using valves of large area in order to keep the velocity of the air, while passing through them, as low as possible.

Riedler Valve.—In this valve no springs are used, the valves being operated by mechanical methods, and driven off the steam cylinder. A cam is fastened to the wrist-plate of the steam cylinder, and moves the rod attached to the air-valve gear by means of two steel rollers. Like the valves in the Riedler pump, these air valves are only closed mechanically. By using these valves the compressors may be driven at a very high speed, without injury to the valves, and no violent oscillation takes place, as in the mushroom type.

Losses in Compressing Air.—As already stated, only a very low efficiency is obtained by air-compressing machinery, and the various losses may be accounted for as follows:—

Heating of air during compression and cooling of compressor.

Loss due to air in clearance space in the cylinder.

Leakage at valves and piston.

Resistance of air in passing through inlet and delivery valves.

Loss due to friction in conveying the air from the receiver in pipes to the point of application underground, and also the friction of the air engine itself.

The two largest sources of loss are those due to heat and friction. The first often absorbs about 25 per cent. of the fuel expended, and the second about 20 per cent., the other losses being comparatively trifling compared to these. The loss due to heating will be readily understood from the fact that air, like any other gas, will expand when heated according to Charles's law; so that with the increase of volume due to the rise in temperature there will also be increased resistance to compression. To overcome heating during compression the following remedies, among others, have been suggested:—

To place the air and steam cylinders as widely apart as possible, so as to prevent the compressor being heated by the steam cylinder.

To place the compressing cylinders outside the engine-house, and simply protect them by a shed.

The air, before being admitted to compressor, should be reduced to zero by some freezing mixture.

At the inbye end of the pipe line a receiver should be placed to trap the moisture conveyed in the air. Under certain conditions it is desirable to replace or supplement the receiver at the end of the pipe line with a water trap and separator, for arresting the entrained water and grease. This trap should be situated at the lowest point in the pipe line and as far as possible from the compressor. By means of this device, dry air is ensured for the machines to be operated, and the danger of freezing is greatly reduced or entirely obviated.

Loss by Clearance.—To reduce loss from this source the clearance space should be as small as possible, or mechanical contrivances must be adopted. Thus a 'trick' passage may be made in the slide valve of a slide valve compressor, so that the air imprisoned in front of the piston can escape to the back of the piston and expand freely. The same object is obtained by having a pass-by groove at each end of the cylinder, so arranged that when the piston reaches the end of its stroke the entrapped air will pass to the other side of the piston, and allow the suction valves to open as soon as the piston begins the return stroke. In this way the loss due to clearance is reduced to a minimum, and no danger is incurred of damaging the cylinder covers.

All leaks in compressors, receivers, or pipes should, for the sake of economy, be strictly guarded against. Air leaks cause greater losses than steam leakage, and therefore no leakage should be allowed unless it is required to ventilate some place in the workings. Air at a pressure of 60 lbs. per sq. in. will have a velocity of 500 ft. per second. The great waste of power through leakage is therefore obvious.

All irregularities or quick bends should also be avoided as much as possible, as these materially increase the friction. One great source of loss from friction is that due to the machinery. As already pointed out, it often amounts to 25 per cent. or 30 per cent., but in well-designed plant it may be no greater than 12 or 15 per cent. The only way to reduce this loss is to secure accurate workmanship, well-designed machinery, and efficient methods of lubrication.

High efficiency is often not all that is desirable, for if complex machinery is introduced, and skilled attention required, the benefit arising from increased efficiency may be more than counterbalanced by increased costs. What is wanted for mining purposes is a compression plant, simple in construction, with a low first cost, not easily put out of order, and yielding fairly efficient results.

Motors.—Other sources of loss are due to friction and leakage at the motor. The formation of ice in the latter and in the exhaust

pipes is a difficulty nearly always encountered in using compressed air. The air, which always contains a certain amount of moisture when it reaches the motor, is at about the same temperature as the atmosphere, and when it enters, and is allowed to expand, the temperature will fall to such an extent that the moisture will immediately freeze. To obviate this, the exhaust passages should be made as large and straight as possible. Reheating the air would also overcome this difficulty, but this is not practicable underground.

With plant placed on the surface there is not the same difficulty in reheating the air before it enters the power motors, and this ought to be adopted, as it greatly increases the efficiency. The heater should be placed as near as possible to the point where the air has to be used, and should be well covered with some good non-conducting material, so as to avoid loss of temperature. At Teversal Collieries, Notts,* where the air was reheated close to the cylinders of the motor, the efficiency was increased 28 per cent.

Losses due to Leakage at Valves and Piston.—Losses of compressed air through valves and piston are often obstinate, owing to the difficulty of keeping the valves in proper adjustment, for, as already stated, the inlet valves may be set too tight and cause undue resistance to the air entering the cylinder. To avoid this, larger valves should be used. If they are badly constructed, the pressure of the air in the cylinder is often 9 or 10 lbs. higher than the pressure in the receiver, and, as a consequence, extra work has to be done to deliver the air against this increased pressure.

The leakage at the piston in air cylinders is often a serious loss. The moisture from the steam in a steam cylinder helps to keep joints tight, but with dry air there is no assistance obtained in this way. The only remedy for this defect is to provide the best workmanship in cylinders, and have the pistons finished to fit as truly as possible.

Loss due to Friction in Pipes.—In all liquids or gases delivered through pipes, there is a certain amount of friction between the fluids and the walls of the pipes. The loss owing to this friction will vary according to diameter of the pipes used, the pressure in them, irregularities, and the number of bends present, etc.

For iron or steel pipes the coefficient of friction varies *inversely* as the diameter of the pipe. Professor Unwin states that for pipes of not less than 12 in. diameter the coefficient of friction is 0·003, but increases for smaller sizes. For practical purposes in colliery work it may be taken at 0·004; or it may be obtained by Professor Unwin's formula:—

$$C = 0.0027 \left(1 + \frac{3}{10d} \right),$$

where C = coefficient of friction.
 d = diameter of pipe in ft.

* *Trans. Inst. Min. Engs.*, vol. xxx. p. 527.

The loss of pressure in the pipes can be approximately obtained from Hawksley's formula :—

$$\frac{p}{.036} = \frac{lv^2}{156,800d}$$

p = loss in pressure in lbs. per sq. in.

v = velocity of the air in ft. per second.

l = length of pipe line in ft.

d = diameter of pipes in ft.

Example.—1500 cub. ft. of free air per minute is compressed to four atmospheres, and then delivered to a motor 900 yds. distant. If the velocity of the air at entrance is 35 ft. per second, and assuming a temperature of 60° F., what would be the loss in pressure at the motor, and what diameter of pipe line would be required?

By Boyle's law, $P_1V_1 = P_2V_2$.

$$15 \times 1500 = 60 \times V_2,$$

$$\therefore V_2 = \frac{15 \times 1500}{60} = 375 \text{ cub. ft. after compression.}$$

$$d^2 \times .7854 \times v = \frac{V_2}{60} \times 144,$$

$$d^2 \times .7854 \times 35 = \frac{375}{60} \times 144,$$

$$d^2 = \frac{375 \times 144}{.7854 \times 35 \times 60} = 32.72,$$

$$\text{and } d = \sqrt{32.72} = 5.71 \text{ in.} = 0.51 \text{ ft. approximately.}$$

$$\frac{p}{.036} = \frac{lv^2}{156,800d} = \frac{900 \times 3 \times 35^2}{156,800 \times .51}.$$

The size of pipes used should be as large as can conveniently be adopted, as this will tend to reduce the friction. In a column of pipes 1000 ft. long and 3 in. diameter, if the air has a velocity of 386 cub. ft. per minute, and a pressure of 60 lbs. per sq. in., the loss due to friction would be $3\frac{1}{3}$ lbs. In 6-in. diameter pipes of the same length and with the same pressure, the loss from friction would only amount to about $\frac{1}{9}$ lb.

The advantages of compressed air may be briefly stated as follows :—

Compressed air, being generated at the surface, is under the direct superintendence of the engineman.

It produces no increase in the temperature of the air in the workings, as is the case when steam is used.

There is not the same difficulty of dealing with the exhaust as there is with steam.

The exhaust air assists in the ventilation, and can even be used for ventilating in cases of emergency.

It can be easily applied to almost every kind of underground machinery, such as rock drills, coal-cutting machines, pumping and hauling engines, etc.

It is a safe motive power, and can be conveniently used in wet, dry, or fiery mines.

Water under High Pressure.—Hydraulic transmission is not extensively employed at collieries, and it has become less so since

electricity has come into general use for underground work. The application of water under high pressure at mines has usually been confined solely to the working of underground pumps, and if the distance is not too great this form of transmitting power may be applied very economically under suitable conditions. The pressure must be fairly high, say above 800 lbs. per sq. in., and the speed of the piston low, not more than about 80 ft. per minute. Examples of hydraulic transmission are given in the description of Moore's hydraulic pump and other types in the chapter on pumping.

Ropes and Rods.—Power transmission is largely used for haulages in and about mines. In the case of haulage the wire rope is really only a secondary transmission, as the rope itself is operated by some other form of power, *e.g.* steam, compressed air, or electricity. As the use of rope transmission is fully described in the chapter on haulage, it need not be referred to at further length here. Ropes are sometimes used to operate pumps placed at some distance from the shaft, but this system can only be looked upon as a temporary expedient, especially if the distance is great, or when the power is off the haulage rope, unless the pump to be so actuated can raise all the water given off in the twenty-four hours during the period the haulage is at work. It would be a great waste of power, causing much wear and tear, and consequently expense for upkeep, to keep a long haulage rope in operation merely to work, perhaps, a single pump.

Ropes are also used for transmitting the power from a steam engine to a dynamo, and a rope drive is also frequently used for driving ventilating fans. In these cases usually a number of cotton ropes are used on grooved pulleys. Such a system of driving is preferred by many to a belt drive, as there is less power lost in transmission than by belts. In a belt drive 10 per cent. is usually lost between the engine or motor and the fan or other plant, due to the sag and slip of the belting.

The cost of rope transmission is also less than for either belting or shafting. The speed of rope drive is usually 2000 to 3000 ft. per minute. The horse-power which may be transmitted by a rope drive can be found from the formula, $H.P. = \frac{pV}{33,000k}$,

where H. P. = horse-power in each rope.

p = working stress in lbs.

V = speed of rope in ft. per minute.

k = constant, varying according to portion of circumference of wheel embraced by rope, varies from 1.08 to 1.7; average 1.39.

p also varies according to circumference of rope.

If circumference is

3 in.

$3\frac{1}{2}$ „

4 „

$4\frac{1}{2}$ „

5 „

6 „

p is

90 lbs.

122 „

160 „

202 „

250 „

360 „

Example.—What horse-power would be transmitted by a drive of ten cotton ropes, each rope 1.75 in. diameter, and the speed of rope 3000 ft. per minute?

Here circumference = $1.75 \times 3.1416 = 5.5$ in., and the value of p would be, say, 300 (as per table), then $H.P. = \frac{300 \times 3000}{33,000 \times 1.39} = 19.6$ for each rope, and total for ten ropes = $19.6 \times 10 = 196$ H.P.

The horse-power transmitted by wire ropes may be calculated by the formula, $H.P. = D^2 \times V \times 3.1$.

D^2 = diameter of rope in in.

V = speed of rope in ft. per second.

3.1 = constant.

Example.—What horse-power will be transmitted by a wire rope 1 in. diameter if the speed is 3000 ft. per minute?

$$H.P. = 1^2 \times \frac{3000}{60} \times 3.1 = 155.$$

Power is transmitted by rods of wood or iron, as in shaft pumps operated through quadrants or bell-cranks by means of a steam engine or other power. This method of transmitting power is fully dealt with in the chapter on pumping, to which the reader is referred for details.

Electricity.—During the last decade a great increase has taken place in the application of electricity as a motive power, and it is growing in favour for certain classes of work.

The principal purposes for which it is adopted, and in which it has proved successful, are haulage, pumping, ventilation, coal-cutting, coal-conveying, and lighting, and for such work it undoubtedly has many advantages over other motive agents, such as compressed air or steam. It is also used for shot-firing.

So far as results are obtainable, electricity does not seem to be very much cheaper, if, indeed, in many cases as cheap as steam used direct; but, as already stated, this is only when the plant to be operated is situated near to where the steam is generated. It must also be borne in mind that there are certain dangers connected with the use of electricity which have not yet been overcome, and which still prevent its extensive use in a large number of mines. Electricity is capable of giving rise to a fire, and there is always the danger of an explosion from 'sparking' in a fiery mine.

There are, however, a large number of mines where these conditions do not exist, and where fire-damp is only given off in small quantities which do not constitute any danger; in such mines, electricity may be used with the greatest advantage, economically and otherwise. The danger of explosion in fiery mines arises in two ways, viz. by 'sparking' at the motor, and by what is called 'short-circuiting' of the cables; that is, by faulty insulation, or by the breakage of the cable by a fall from the roof or some other cause.

Where there is danger from fire-damp, especially near to the coal face where coal-cutters or conveyers may be employed, a combination of electricity and compressed air may be employed.

In view of the regulations under the Coal Mines Act, 1911, and the Special Electrical Rules, 1911, this system is likely to be more largely employed in the future in mines where fire-damp is given off

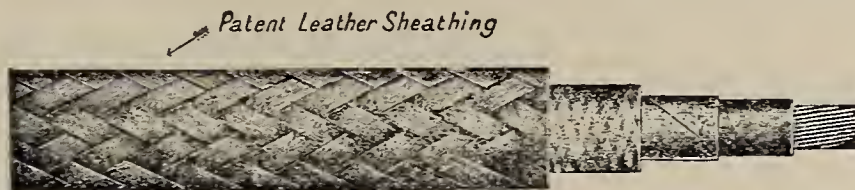


FIG. 153.—Flexible trailing cable with leather sheathing.

in considerable quantity at the working face. One of the best-known compressors for this method of generating power is the 'Reavell' compressor. This compressor consists of four cylinders arranged radially in a circular-shaped casing, and is usually connected direct to an electrically driven motor. The motor and compressor are placed at a suitable place (say in the intake airway) in the workings, and the air generated at this point, and then conveyed to the parts of the workings where power is required.

The motor and compressor take up little space, and it solves a difficulty which could not otherwise be overcome by using electricity alone. The writer saw a plant of this description used in the resinking of a deep shaft in Fife, the compressed air being necessary for the driving of rock drills, and it gave excellent results. The advantage of using this method of generating compressed air is not only its safety, but the much higher efficiency obtained than when the air is generated on the surface, owing to the much lesser distance it has to be conveyed.

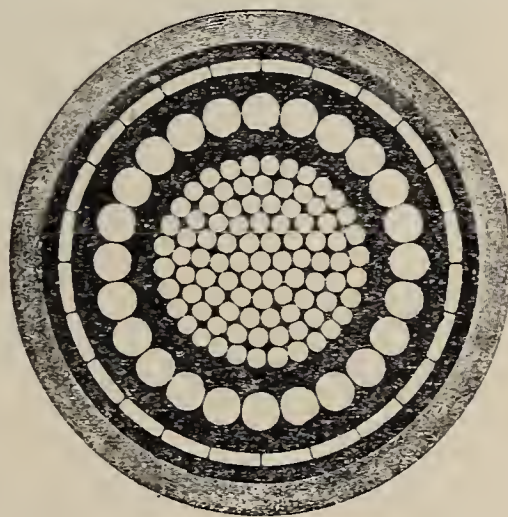


FIG. 154.—Section of diatrine paper-covered leadless concentric cable.

At Ouston Colliery in Durham a combined electricity and compressed air plant was employed for driving percussive heading machines, and an efficiency of 60 per cent. was obtained.*

Electric Mains or Cables.—To conduct an electric current from the place of generation (the dynamo) to where it is to be used (the motor), cables, or metallic conductors, are required. Generally two, or, for three-phase current, three such cables are required for a circuit, one to lead the current in and the other to lead it out, thus resembling a hydraulic motor with its supply and delivery pipes.

* *Trans. Inst. Min. Engs.*, vol. xxxi. p. 256.

The selection of a type of cable will depend upon the conditions under which it has to work. The electric circuit of a colliery may be divided into five portions, each of which introduces specific conditions: (a) surface workings; (b) shafts; (c) main roads; (d) inbye or drawing roads; (e) the coal face.

Each of these portions will require a cable to meet the conditions under which it will have to work. For conveying power to surface works such as screening plant, washeries, engineering shops, etc., a vulcanised rubber cable, heavily braided and compounded, will be found satisfactory. The cable should be of good quality, and flexible for ease in handling. Cables are usually of the following types, viz.

single-core cables, twin-core cables, concentric or double-core cables, and three-core cables.

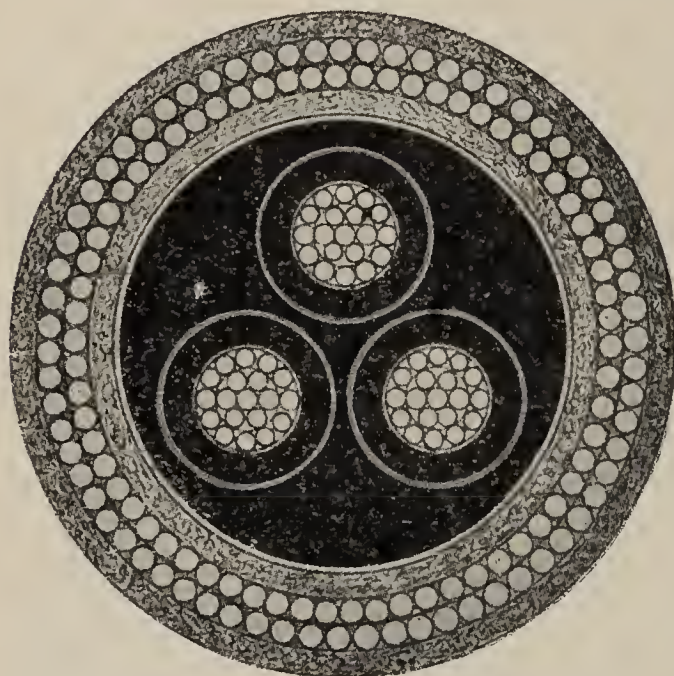


FIG. 155. — Three-core shaft cable, for three-phase working; insulated, wormed, and sheathed and protected by two layers of galvanised iron wires.

Single-core cables have a single conductor placed in each. With this type two cables are required for the circuit. This class of cable is very largely used for colliery work, and is well suited to the rough wear and tear of underground work. They are easy to repair and joint, and are not so apt to 'short-circuit' as the other types, as they can be kept as far apart as the exigencies of the underground roads will allow. Concentric cables are those in which the intake and return

conductors are both encased in a single cable, the conductors being separated from each other by insulating material. When the current has to be carried a long distance or down deep shafts, it is an advantage to use concentric cables, as it reduces the cost and there is less loss of potential. Concentric cables are, however, more troublesome to take branches and joints from, and more difficult to repair when damaged.

Three-core cables, as the name implies, are those in which three conductors are carried under the same covering. They are used for three-phase electric current, and, like concentric cables, are advantageous when the current has to be carried long distances, as they reduce loss of potential and save the carrying of three single parallel cables. All cables used underground should be well insulated to prevent risk of accident from shock by contact with 'live' wires.

Cables for underground work may be insulated with impregnated paper (diatrine or other special compounds being used); then sheathed with bitumen, and protected with tapes and braiding. Vulcanised bitumen is one of the best and most suitable materials for insulating colliery cables; it is not easily injured, and it will be found to give good results and wear well under all ordinary conditions of underground work. Bitumen fibre is also used in conjunction with some of the other materials mentioned, as is also jute compound. India-rubber pure is rarely used by itself, as it is expensive and easily injured. Lead-covered cables should be used with great caution underground, as the lead covering may be chemically acted upon by pit water containing acids, and as a general rule lead-protected cables cannot be recommended for mining work.

In addition to the insulation mentioned above, it is now compulsory to have a metallic outer covering or armouring on all underground cables, unless under certain specified conditions. Unarmoured cables may be used in gate roads if a separate earth wire is provided and there is no danger of igniting coal-dust, and armoured cables are not necessary for conveying to portable apparatus such as rock drills, coal-cutters, etc. This requirement as to armoured cables does not come into force until 1st January 1920 for cables already installed before 1st June 1911, but all new cables put in after that date must be armoured (see Special Rules for the Installation and Use of Electricity in Mines, 1911).

For use at the coal face with coal-cutters, flexibility in the cables is essential, and therefore trailing cables should not be armoured, but, instead, an outer covering of hemp or leather braiding is usually employed. A spiral layer of tarred rope $\frac{1}{4}$ to $\frac{3}{8}$ in. thick makes an excellent outer protection for trailing cables; it lasts well and is easily repaired.

Armoured Cables.—Cables for colliery work are frequently covered by armouring to protect them against mechanical injury from falls of roof, sides, etc. The armouring may consist of—

A single layer of galvanised iron wires, protected by jute compound.

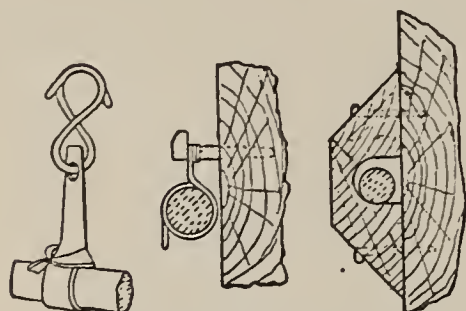
A double layer as above, covered by jute compound.

A double layer of steel tape covered by jute compound.

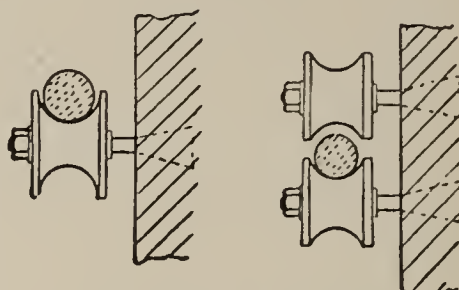
The armouring necessary for cables as specified by the Special Electrical Rules, 1911, are as follows:—For cables below 0·5 in. diameter, galvanised steel wire 0·072 in. diameter; cables from 0·5 to 1 in. diameter, two layers of compounded steel tape, each 0·03 in. thick; cables 1·01 to 2 in. diameter two layers of compounded steel tape, each 0·04 in. thick; and for cables above 2 in. diameter, two layers, each 0·6 in. thick.

Shaft Cables.—For shaft work several types of cables may be used. If continuous or direct current is employed, either two single cables

or a twin-core cable may be used. These cables should be insulated with impregnated paper, then sheathed with bitumen, and protected with tape and braiding; and, unless the cables are completely enclosed in wooden casing, they should also be protected with two layers of galvanised iron wire (armouring) to protect them from injury by falling material. The armouring protects the cables from



FIGS. 156 and 157.



FIGS. 158 and 159.

mechanical injury when fixed, and also allows them to be lowered down the shaft without risk of damage to the insulation, because the copper conductors, especially in large cables, would be liable to stretch or rupture, by reason of their own weight, and the insulation would also be damaged. For alternating three-phase current three single-core cables, or a three-core cable, may be used, similarly

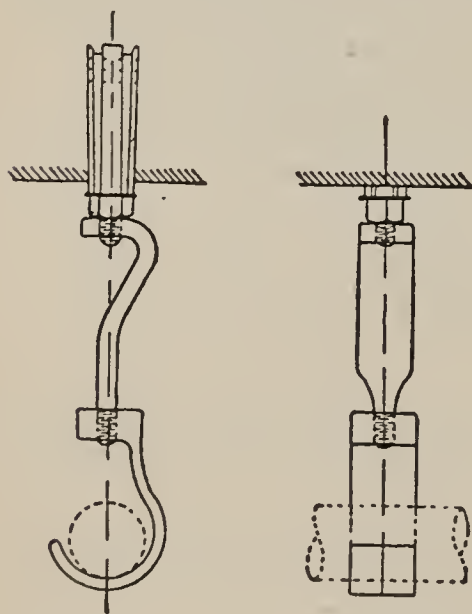


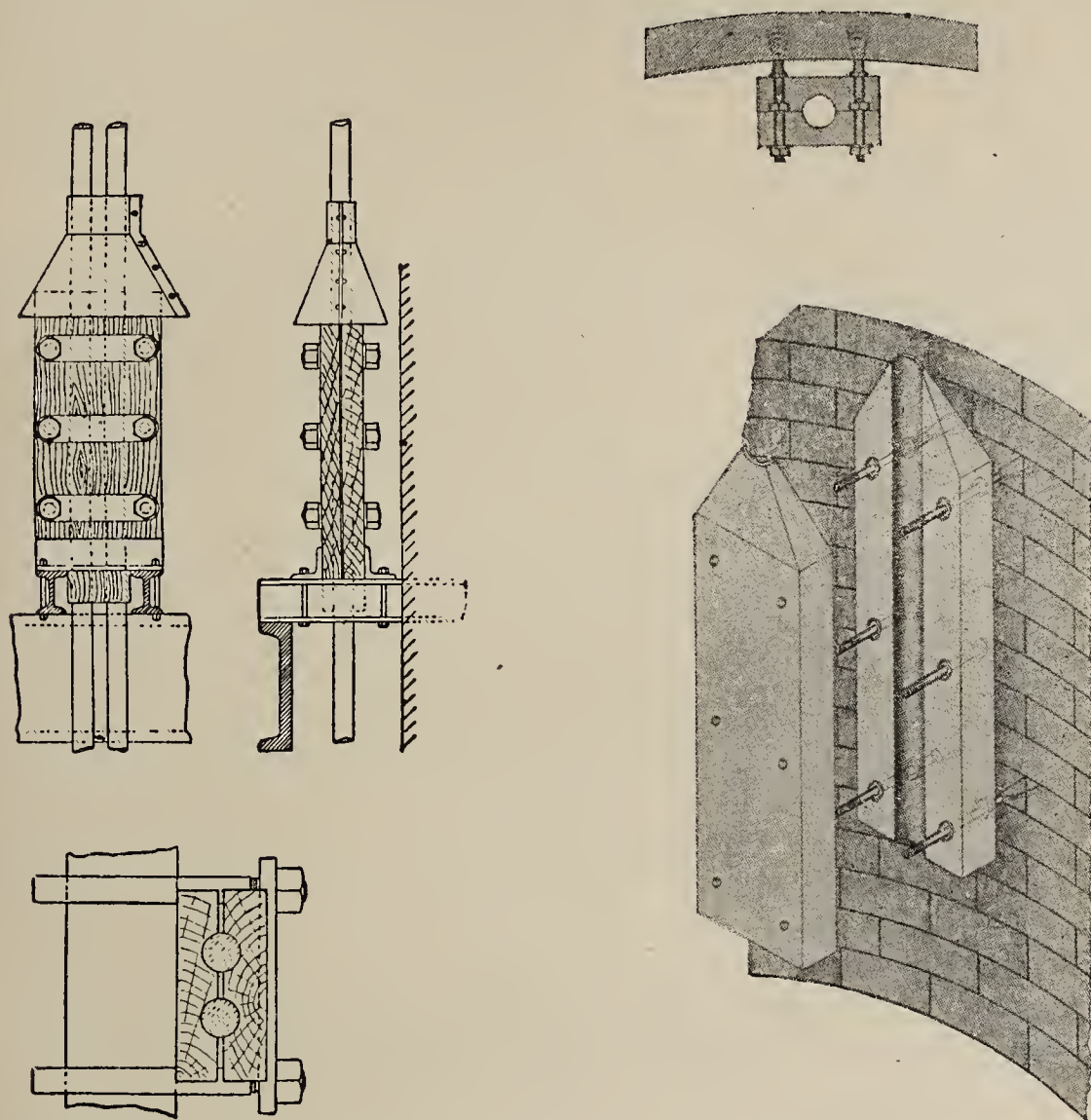
FIG. 160.—Methods of fixing underground cables.

insulated to above and armoured. Unless for very deep shafts the latter type of cable is most frequently employed, but where the depth is great, and a large amount of current requires to be carried, necessitating a cable of large sectional area, the weight of such a cable makes it difficult to carry properly in the shaft, with consequent liability to rupture. There is the further disadvantage that in a deep shaft jointing would have to be resorted to, which would necessitate more frequent testing for earthing and bonding. On the other hand, single-core cables may be made for the deepest shaft without a joint, and they are lighter and more easily handled. In securing the cables in the shaft a variety of

ways may be adopted. A common method is to fix them by wood cleats to the shaft sides or to buntons. Figs. 161 to 165 illustrate methods of fixing cables in shafts.

Fixing Underground Cables.—Where cables have to be taken along main haulage roads or fixed in roadways where there is plenty of room, they may be attached to props or side timbers and supported so that they can be readily taken down for inspection or repair. A

convenient and easy method of fixing cables is by leather strips, 1 in. to $1\frac{1}{2}$ in. wide, secured by flat-headed nails to the timbers every 10 yds. or so. In place of the leather strips some use pieces of tarred twine nailed to the props, and of sufficient strength to carry the cables. Both methods are quite efficient if well looked after, and if a fall of roof does take place the cables and fastening are carried to



FIGS. 161, 162, and 163.—Showing position of cleat (with cable) resting on beam in shaft.

FIG. 164.—Shaft cable cleat. Shown fixed to the wall or shaft by means of lag bolts, ready to receive the cable.

the ground with the débris with less injury than if the fastenings were stronger and of a more permanent character. Sometimes the cables are laid in wooden boxing, the boxing being afterwards run in full with pitch, which has the merit of keeping out damp and preventing mechanical injury to the cable. Wherever possible, cables should be taken along the intake airways, for if taken along the return they are more liable to wear and to damage from damp air and heat.

Junction Boxes.—The cables for underground roads should be

coupled together by junction boxes, and such boxes should be used at all joints and branches. These junction boxes should also be used for testing purposes.

Voltages.—The term voltage is also known as ‘pressure,’ and means the difference of electrical potential between any two conductors, or between a conductor and earth, as read by a hot-wire or electrostatic voltmeter. In the Special Electrical Rules applying to mines, voltages or pressures are divided into four classes, viz. :—

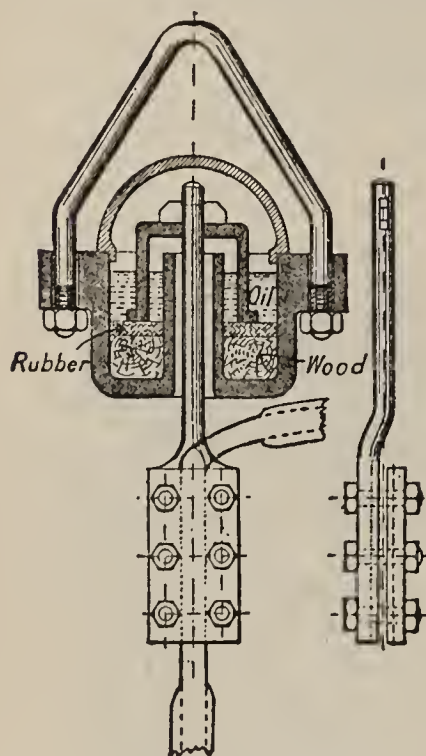


FIG. 165.

- (a) Low pressure, which means a pressure in a system normally not exceeding 250 volts.
- (b) Medium pressure means a pressure in a system normally above 250, but not exceeding 650 volts.
- (c) High pressure means a pressure in a system normally above 650, but not exceeding 3000 volts.
- (d) Extra-high pressure means a pressure normally exceeding 3000 volts.

All these pressures are defined as being measured at the point where electrical energy is used or supplied.

The pressure at which electrical mining plant is run varies greatly according to the design of the machine and the work for which it is intended. For colliery work with continuous current, the voltage used varies between 200 and 600 volts, 400 to 500 volts being a common pressure, and one which is not dangerously high under

ordinary conditions.

Mr W. C. Mountain considers 500 to 600 volts, both with continuous current and three-phase alternating current machinery, a most satisfactory voltage where the motors are placed not more than a mile from the generating plant, and he does not think that even with the three-phase machinery any great saving would be obtained by running at a higher voltage and putting in transformers. For underground motors, a voltage of 500 may be considered a practical working pressure, and is sufficient, provided the power required is not too great.

In Continental mines, especially in Germany and Belgium, the three-phase system is almost universally employed, the voltages being 1000 to 3000, the current being transformed into a lower voltage, 500 to 600, by transformers at some suitable point underground, such as the pit bottom or at station at the end of a main haulage road. This practice is now being largely adopted in British mines for alternating current, which is generated at the surface at 2000 to 3000 volts, and then transformed down at a suitable point underground to 500 or 600 volts, the pressure utilised for driving

the motors. Sometimes the opposite course is adopted, viz. generating at 500 or 600 volts, then transforming up to 2000 or 3000 volts, and again stepping down to the original pressure of 500 or 600 volts at a point underground. This may be done where the distance underground is long over which the current has to be carried.

Generators or Dynamos.—Dynamos have been defined as machines for converting mechanical energy into electrical current or energy. They are usually divided into two classes, viz. :—

(a) Continuous current machines.

(b) Alternating current machines.

Each class comprises many different types, for technical descriptions of which the student should consult one of the many text-books on electrical engineering. Both types of machines are very largely used for colliery work, and both have been found suitable for the varying conditions usually existing. Both systems have their advantages and disadvantages. For intermittent working plant, such as main and tail rope haulage, and coal-cutting where starting and stopping is of frequent occurrence, there is no doubt but that direct current has many advantages. The starting torque is much greater for an alternating current motor, sometimes as much as 100 per cent. over the normal, and therefore, unless there, the motor requires to be of larger capacity than with direct current. There is also a higher first cost for cables in alternating current, and greater danger from accident by shock. The chief disadvantage of direct current is that of sparking at the commutator, but this can be largely eliminated by using an induction motor in conjunction with an auto-starter. The motor can also be totally enclosed. On the other hand, there are many advantages in using three-phase alternating plant. It is more suitable for a variable load, no danger from sparking, and more suitable and economical for transmitting power long distances. Three-phase motors are usually of much better mechanical construction than direct current motors, and therefore breakdowns are less likely. There should also be less liability from interruptions of works, lower cost for repairs and maintenance.

Continuous-current dynamos are divided into three classes according to the method adopted in the winding, the three types being—

Series wound machines.

Shunt wound machines.

Compound wound machines.

Series Wound.—In this system the field-winding, i.e. the armature, the field-coil and the working circuit, are all in series receiving the same current, the current flowing from the positive brush through the field-coil windings, then through the external circuit and back to the negative brush. The whole of the current is sent through a coil consisting of a few windings of comparatively thick wire.

Shunt Wound.—In this type of winding a double path is open to the current. One part goes through the field-coil, which, in this instance, consists of a large number of windings of fine wire, of sufficient resistance and length to give the proper number of ampere-turns to fully excite the magnet, and is connected right across the brushes or poles to get the full pressure. The other part of the current flows through the external circuit, both currents joining at the negative brush before they return to the armature. The magnet coils act as a shunt to the main circuit. This type of dynamo is most suitable where a variable speed is required, such as, for instance, in main and tail rope haulage.

Compound Wound.—When the field-coil of a dynamo is wound with both a shunt and series coil of windings, it is called a compound dynamo. It will be seen that this is a combination of the two just described. The shunt coil consists of a large number of turns of fine wire calculated to give full potential at no load and with the magnet not fully excited, so that when the current increases in the external circuit it passes round the series coils, which are of thick wire, and increases the magnetism, and so raises the pressure to compensate for the drop in potential due to the resistances in the armature circuit. This arrangement of windings enables the dynamo to be self-regulating and give a constant E.M.F. with varying loads.

Motors.—A motor is a machine for converting electrical energy into mechanical energy. If we send a current through the armature of a dynamo whose magnetic field is excited, the armature will be put in motion. With the dynamo armature there will, however, take place, not only a single movement, but a permanent rotation. Owing to the action of the commutator, the current flows through all wires on one half of the armature which are under the influence of the north pole, in one direction, and through all wires which are under the influence of the south pole, in the opposite direction; hence as long as a current is sent through it, the armature will rotate. The machine now absorbs electrical and supplies mechanical energy. In this case the machine is called an *electric motor*, which we may speak of simply as a *motor*, whereas a machine which produces current is called a dynamo or generator. As the construction of the motor is practically the same as the dynamo, we need not further describe it. There is one important difference, however, in the driving of the motor which may be pointed out; the armature always moves in an opposite direction to that of the dynamo armature. If to get a current the dynamo armature requires to turn to the right, then the motor armature will run towards the left.

Electrical Plant Failures.—Electric installations are not more liable to an excessive number of breakdowns than non-electric plant. With reasonable care in design, manufacture, and working, electric machinery can be, and has been, made as reliable, if not more so, than steam, hydraulic, compressed air, or oil-power plants. The

first desideratum in colliery installations is to have ample power in the generating plant and also in the motors. Nearly all the trouble in the past has arisen owing to the plants being made too small for the work they are expected to perform.

The causes of accidents in electric plant are due to constructional defects; bad design and perishing insulation; bad workmanship; overloading; damp and dust, and oil and defective attention.

Electricity has, in England and Scotland, been chiefly applied to the operations of hauling and pumping, coal-cutting machines and rock drills, and is now in quite a number of cases being used for winding.

Compared with compressed air there can be no doubt that electricity gives a very much higher efficiency, often as much as 30 per cent. more. The efficiency of electrical machinery may vary from 40 to 60 per cent. In a paper read by Mr Robertson before the Institution of Civil Engineers, he states that an efficiency of 50 per cent. was obtained from the electrical haulage plant recently erected at Earnock Colliery, Hamilton. Comparing the indicated horse-power of the engine with the power developed on the hauling ropes the losses were—

	Per cent.
In engine, ...	22·00
„ belt and dynamo, ...	8·00
„ cable, ...	12·50
„ motor and gear, ...	7·50
Total loss, ...	50·00

At Haden Hill Colliery the electrical haulage plant gave 59 per cent. efficiency, the losses being—

	Per cent.
In dynamo, ...	9·00
„ cable, ...	8·00
„ motor, ...	9·00
„ gearing, ...	15·00
Total loss, ...	41·00

Another instance may, however, be given, where the tests made on a hauling plant gave an efficiency of 71 per cent.

Tests of Hauling Engines.

Distance of engine to hauling motor, ...	1400 yards
Energy imparted to dynamo, ...	22 H.P.
Work done by dynamo, ...	19·8 „
Loss of energy in dynamo, ...	2·4 „
„ „ cables, ...	2·6 „
Energy imparted to motor, ...	17·2 „
Loss of energy in motor, ...	1·5 „
Work expended in raising coals and tubs, ...	12·1 „
„ „ friction of moving tubs, ...	1·3 „
„ „ haulage, gearing, and ropes, ...	2·3 „

The speed of the rope in this case was three miles per hour, and the total efficiency was 71·3 per cent., which is a very satisfactory result.

Comparing also the first cost of installing compressed air and electricity respectively, the latter is the cheaper. As a general rule an air-compressing plant will cost 20 per cent. to 30 per cent. more than an electrical plant capable of performing the same amount of work, while the cost of running the latter is $7\frac{1}{2}$ per cent. to 10 per cent. less than that of running a compressed air installation of the same power.

Mr J. T. Forgie, in a paper read before the Mining Institute of Scotland, gives the following detailed costs of an electrical haulage plant erected at Dumbreck Colliery, Kilsyth * :—

Generating engine 20 in. diam. by 30 in. stroke complete,	£300	0	0
One dynamo, 50 horse-power,	374	0	0
Two motors, 20 horse-power each,	440	0	0
1750 ft. of 19/17 cable,	93	0	0
Instruments, switches, etc.,	81	10	0
Labour, packing, carriage, etc.,	97	5	0
Two underground haulage arrangements,	400	0	0
Total cost,	£1785	15	0

The cost of a compressed air plant to do the same work may approximately be shown as follows:—

Steam engine, air cylinders, valves, etc.,	£1200	0	0
1750 ft. air pipes 6 in. diameter,	180	16	0
Two air receivers,	70	0	0
Two haulage arrangements,	400	0	0
Labour in fixing pipes, etc.,	100	0	0
Sundries,	200	0	0
Total cost,	£2150	16	0

Showing a difference in first cost of £365, 1s. in favour of the electrical plant, or a saving of approximately 23 per cent.

Pumping.—To no branch of mining has electricity been more successfully applied than that of pumping, and there can be no doubt of its suitability for such purposes, owing to the small space occupied by the driving motor compared with a steam engine, and by cables compared with steam or air pipes, together with the ease with which the former may be carried to any part of the workings.

The cost of the cable will be only about one-half that of pipes.

Electrical pumping installations differ very widely in efficiency, a great deal depending on the suitability of the motor for the class of work required. In a paper read before the Institution of Civil

* *Trans. Inst. Min. Engs.*, vol. vii. p. 129,

Engineers by L. B. and C. W. Atkinson, describing an electrical pumping plant, the gross efficiency was given at 49 per cent., the generating engine being placed 1200 yds. distant from the pumps. Comparing the I.H.P. of the engine, which was 31·75, with the volume of water delivered, the losses were distributed as follows :—

	Per cent.
Loss in generating engine and in belts, ...	31·50
„ dynamo, ...	6·05
„ cable, ...	5·36
„ motor, ...	4·72
„ pump and gear, ...	3·15
Total loss, ...	<u>50·78</u>

Mr T. L. Galloway * in a paper on electrically driven turbine pumps gives the following results :—

Power Used.	Head.	Quantity.	Work Done.	Useful Effect.
Horse- power.	Ft.	Gallons per min.	Horse- power.	Per cent.
50·3	237	440	31·6	62
55·7	242	475	34·8	62
55·1	277	389	32·6	64
43·0	307	294	27·3	63

The pump was of the four-stage turbine type, and was driven by a three-phase induction motor, at a voltage of 500.

In a test made with a triple ram pump 9 in. diameter by 12 in. stroke, pump geared $10\frac{1}{2}$ to 1, the useful effect obtained was 72 per cent. when raising 429 gallons per minute against a statical head of 383 ft. With a head of 171 ft. the useful effect was reduced to 59 per cent.

One of the great advantages claimed for the electrical transmission of power is, that a dynamo or a motor is self-regulating, *i.e.* the dynamo only requires sufficient power to drive it, to enable it to accomplish the work which it is called upon to perform, and a motor only requires current sufficient for the same purpose. It is of the greatest importance that the dynamo or motor should be large enough for the work. The best results are only obtainable with a fair surplus of power in the steam engine. The advantages and disadvantages of electricity as a motive power may be briefly stated thus :—

* *Trans. Inst. Min. Engs.*, vol. xxxvi. p. 82.

Advantages :—

The great facility with which it can be used in any part of the workings, and a motor put down wherever required, for driving a pump or haulage system.

The small amount of space occupied by the motor, while, owing to the high speed at which it works, a large amount of power can be applied from quite a small pulley by belting.

It does away with the danger of ropes or pipes in the shaft, and avoids the complication of pulleys and ropes at the pit-head and pit-bottom.

The cables can be readily fixed and taken round curves ; there are no joints to be affected (as in pipes) by vibrations or shocks, and the space occupied by conductors is very small.

The surface plant can be placed any distance from the shaft and not necessarily in line with the latter.

Higher efficiency can be obtained than with compressed air or steam when used underground.

Disadvantages :—

The danger of fire, either igniting fire-damp or setting fire to screen-cloth or brattice-wood, owing to sparking at the motor or 'short-circuiting' in the cables.

Electric machinery is easily damaged and thrown out of order, and often requires skilled men to repair it.

Its unsuitability in damp and dirty workings, as the cables and beltings soon suffer injury under such conditions.

The first cost is much higher than for ordinary haulage engines driven by steam.

The following table shows the comparison in the cost of transmitting power by the various systems in use * :—

System employed.	330 ft.	1640 ft.	3280 ft.	16,400 ft.	32,810 ft.	65,250 ft.	Motor.
	Pence.	Pence.	Pence.	Pence.	Pence.	Pence.	
Electricity, . . .	1·80	1·84	1·94	2·27	2·84	4·61	} Steam.
Water pressure, . .	2·27	2·36	2·61	4·03	5·79	9·57	
Compressed air, . .	3·17	3·23	3·33	4·05	5·25	6·39	
Ropes, ...	1·26	1·49	1·58	2·88	4·99	12·68	} Hydraulic.
Electricity, . . .	0·46	0·48	0·53	0·62	0·68	1·15	
Water pressure, . .	0·46	0·55	0·62	1·33	3·23	3·54	
Compressed air, . .	0·72	0·80	0·84	1·21	1·86	3·36	
Ropes, ...	0·25	0·27	0·30	0·80	1·42	3·39	

The cost as given above is for the transmission of 100 horse-power for the distance tabulated, from which it would seem that transmission by wire ropes is the cheaper up to 3000 ft., and electricity for longer distances.

Electrical Terms.—The following are the principal terms or definitions used in electrical work :—

* *Trans. Inst. Civil Engrs.*

Ampere = 10^{-1} C.G.S. units of current; or the ampere is the constant electric current which, when passed through a particular solution of nitrate of silver in water, deposits silver at the rate of 0.001118 gramme per second (almost exactly 0.0648 gramme or 1 grain per minute).

Volt = unit of electrical pressure, or = 10^8 C.G.S. units of electromotive force (E.M.F.), or the volt is the electrical pressure which, if steadily applied to a conductor whose resistance is 1 ohm, will produce a current of 1 ampere, and which is represented by $\frac{1000}{1434}$ or 0.6974 part of the electrical pressure

between the poles of a voltaic cell, known as Clark's cell, when that cell is at a temperature of 15° C.; or, simply, it is the electrical pressure required to send unit current of 1 ampere through a conductor whose resistance is 1 ohm.

Ohm = 10^9 C.G.S. units of resistance, or the ohm is the resistance offered to an unvarying electric current by a uniform column of mercury at 0° C., 14.4521 grammes in mass, of a constant cross section (1 sq. mm.), and of a length of 106.3 centimetres. [10 ft. of copper wire, $\frac{1}{100}$ th in. diameter (No. 33 S.W.G.), represents a resistance of 1 ohm, or approximately 60 metres of copper wire 1 sq. mm. in cross section.]

The definitions given are the practical ones in contra-distinction to the legal definitions of the Board of Trade. The ampere, volt, and ohm are the only three legalised electrical units. There are other terms or electrical units in use:—

Coulomb = unit quantity of electricity, or = 10^{-1} C.G.S. unit of quantity = 1 ampere-second. Coulomb is that quantity of electric current that will deposit 0.001118 gramme of silver per second when passed through a solution of nitrate of silver. Coulomb is a term now seldom used, current being always measured in amperes.

Joule = unit of work, and is the work done in 1 second when 1 ampere flows under a pressure of 1 volt = 10^7 C.G.S. units of work = 3.725×10^{-7} horse-power per hour.

Watt = unit of power or rate of doing work = 10^7 C.G.S. units of power. For practical purposes a watt is the power absorbed in a circuit when 1 ampere passes under a potential of 1 volt.

$$\begin{aligned}\text{Thus } 1 \text{ ampere} \times 1 \text{ volt} &= 1 \text{ watt.} \\ 746 \text{ watts} &= 1 \text{ horse-power} = 33,000 \text{ foot-lbs.}\end{aligned}$$

Kilowatt = 1000 watts. The Board of Trade unit of electrical energy is known as the kilowatt-hour, and is the quantity of energy supplied in 1 hour by a current at such a pressure that the product of volts, amperes, and hours = 1000.

$$1 \text{ watt} = 44.25 \text{ foot-lbs., and } 746 \text{ watts} = 33,000 \text{ foot-lbs. } \therefore 1$$

$\text{kilowatt} = \frac{44 \cdot 25 \times 1000}{33,000} = 1 \cdot 34 \text{ H.P., usually taken} = 1\frac{1}{3} \text{ H.P. ; or}$

$1 \text{ B.T.U.} = \frac{1000}{746} = 1\frac{1}{3} \text{ H.P. hours.}$

Example.—A dynamo which supplied current of 125 amperes at a pressure of 600 volts would be supplying energy at the rate of $125 \times 600 = 75,000$ watts. $\therefore \frac{75,000}{100} = 75$ kilowatts, or = 75 B.T.U., and $75 \times 1\frac{1}{3} = 100$ H.P. hours.

The various electrical units may be summarised as below :—

Unit.	Symbol.	Name.	Derivation.
E.M. F.	E	Volt	Ampere $\times$ ohm
Current	C	Ampere	Volt $\div$ ohm
Resistance	R	Ohm	Volt $\div$ ampere
Quantity	Q	Coulomb	Ampere per second
Power	P	Watt	Volt $\times$ ampere
Work	W	Joule	1 watt for 1 second

The value of the watt is $\cdot 7373$ foot-lbs. per second = $44 \cdot 25$ foot-lbs. per minute. $\therefore 746$ watts = $33,000$ foot-lbs. per minute = one horse-power. Kilowatt = 1000 watts. Megohm = $1,000,000$ ohms. Board of Trade unit of work = 1000 watt-hours = $1 \cdot 34$ E.H.P. for one hour.

The difference of pressure of electrical energy at varying points is usually spoken of as electromotive force (written E.M.F.), and is measured in volts.

Let E = electromotive force in volts,
R = resistance in ohms,
C = the current in coulombs,

then $E = R \times C$ or $C = \frac{E}{R}$: (1) $C = \frac{E}{R}$; (2) $R = \frac{E}{C}$; (3) $E = C \times R$;

or power (in watts), $P = \text{electromotive force} \times \text{current strength}$, or $P = E \times C$. If voltage and resistance only be known, to find P, first find the current by the formula :

$C = \frac{E}{R}$; then $P = E \times C = E \times \frac{E}{R} = \frac{E^2}{R}$.

Again, if current and resistance be given, the power (P) can be calculated from the third form of Ohm's law : $E = C \times R$.

By substituting this value of the E.M.F. in $P = E \times C$; i.e. $E = C \times R \therefore P = (C \times R)C = C^2R$, we therefore have the three formulæ :

(1) $P = E \times C$; (2) $P = \frac{E^2}{R}$; (3) $P = C^2 \times R$.

Work done in mechanics is usually measured in foot-lbs.

In electricity, however, the watt is the unit of work (= volt × ampere).

$$\text{H.P. in mechanics} = \frac{\text{foot-lbs.}}{33,000}.$$

$$\text{H.P. in electricity} = \frac{\text{watts}}{746} = \frac{\text{volts} \times \text{amperes}}{746}.$$

Power given out by a current of 1 ampere at a tension of 1 volt =				1 watt
„	„	2 amperes	„	2 volts = 4 watts
„	„	10	„	2 „ = 20 „
„	„	74.6	„	10 „ = 746 „ = 1 H.P.
„	„	7.46	„	100 „ = 746 „ = 1 H.P.

That is, a current of 74.6 amperes at a tension of ten volts, or one of 7.46 amperes at a tension of 100 volts, is equal to one electrical horse-power.

A wire of a given size will permit the flow of a given number of amperes proportional to its area. For instance, a conductor of 19 wires, each wire being 16 B.W.G., is suitable for the flow of about 50 amperes for a distance of 1000 yds., with a loss in the conductor of, roughly, 10 per cent. If the voltage of the above current be 500, then the horse-power will be = $\frac{500 \times 50}{746} = 33.5$.

For three-phase alternating current

$$\text{H.P.} = \frac{\text{volts} \times \text{amps.} \times \text{power factor}}{746}$$

$$\text{or } \frac{\text{volts} \times \text{amps.} \times K}{746}.$$

For large current and small pressure or voltage, large conductors must be used, which are expensive. For small current and high voltage small conductors may, however, be employed, which is more economical, although they increase the danger arising from sparking, short circuits, etc. For underground work voltages exceeding 500 are rarely used, owing to the danger of sparking and of shock to men coming in contact with the cables. For electric lighting with incandescent lamps, a voltage of 50 to 100 volts is usually employed. An incandescent lamp of 20 candle-power at 100 volts takes about 0.6 ampere, *i.e.* $100 \times .6 = 60$ watts, or 3 watts per candle-power.*

Addendum.

To find the Area of Conductor for a given Current.—This depends on current density calculated from loss allowed, due to resistance, efficiency of motor, etc.

Thus for 1000 amps. the loss per sq. in. of sectional area of conductor = 2.45

* For further details of the application of electrical power in mines the reader is referred to *Electrical Practice in Collieries*, by Daniel Burns, M.Inst.M.E., Chas. Griffin & Co., Ltd., London, Third edition.

volts per 100 yds. The loss can, therefore, be readily resolved into a certain density per sq. in.

The area varies *inversely* as the current density, *e.g.* 120 amps. at a density of

$$1000 \text{ per sq. in. gives a section} = \frac{120}{1000} = 0.12 \text{ sq. in.};$$

$$\text{or if density} = 450 \text{ amps. per sq. in.,}$$

$$\text{sectional area} = \frac{120}{450} = 0.26 \text{ sq. in.}$$

To estimate Area of Cable for a given Loss of Power :—

For continuous current :

$$A = \frac{\text{amps.} \times \text{length of (lead + return) in yds.}}{\text{loss in volts} \times 40,000}$$

For three-phase alternating current :

$$A = \frac{\text{amps. per cable} \times \text{length in yds. (one direction)}}{(\text{loss in volts} \div 1.73) \times 40,000}.$$

(This rule only applies to alternating current requiring a section area not greater than 0.4 sq. in.)

Example.—A motor is situated 2000 yds. from the generator, and the power required is 120 H.P. Find area of conductor if loss at motor is 35 volts, for (a) continuous current two-wire; (b) alternating three-phase, three-wire. Power factor $K = 0.8$.

$$120 \text{ H.P.} = 120 \times 746 = 89,520 \text{ watts.}$$

Assuming a voltage of 450 at motor terminals,

$$\text{then amps.} = \frac{89,520 \times 1}{450} = 198.93,$$

$$\text{and sectional area} = \frac{198.93 \times 2000 \times 2}{35 \times 40,000} = 0.57 \text{ sq. in.};$$

for alternating current, three-phase,

$$\text{watts per cable} = \frac{89,540}{3} = 29,840,$$

$$\text{amps. per cable} = \frac{29,840}{\frac{450}{1.73} \times 0.8} = \frac{29,840}{191} = 156.2,$$

$$\text{and sectional area} = \frac{156.2 \times 2000}{\frac{35}{1.73} \times 40,000} = 0.38 \text{ sq. in.}$$

When the power factor $K = 0.8$ the same result can be obtained by adding the areas of the two wires for continuous current and dividing by 3.

$$\text{Thus } 0.57 + 0.57 = 1.14 \text{ sq. in.,}$$

$$\text{and } \frac{1.14}{3} = 0.38 \text{ sq. in., the same as before.}$$

Current in Mains :—

$$\text{Current (C) in main conductors} = \frac{\text{output in watts} \times \text{constant}}{\text{volts between line wires}}.$$

Constant (K) = 1.00 for continuous current of all classes.

Three-phase three-wire—

Efficiency of load =	100	95	90	85	80	75 per cent.
K =	.58	.65	.64	.68	.72	.76 ,,

Example.—What current will be required in main conductors if 150 H.P. requires to be transmitted for power purposes by (a) two-wire continuous current ; (b) three-phase, three-wire alternating current ?

Assume a potential of 450 volts between wire lines and an efficiency of 75 per cent.

$$150 \text{ H.P.} = 150 \times 746 = 111,900 \text{ watts.}$$

$$(a) C = \frac{111,900 \times 1}{450} = 248.66 \text{ amps. for continuous current.}$$

$$(b) C = \frac{111,900}{450 \times 0.76} = 327.2 \text{ amps. for alternating current.}$$

Thus two-wire continuous current will require cables of a capacity to carry 248.66 amps., and the three-phase alternating system requires to carry 327.2 amps., but three conductors are required instead of two as in the continuous system.

Loss in Conductors.—The fall in potential may be calculated from Ohm's law when the resistance and current are known. In making such calculations the total length of cable (lead and return) must be taken into account.

Example.—Find the loss in volts in a $\frac{1}{8}$ cable, where the total length is $1\frac{1}{2}$ miles and the current 120 amps.

The resistance of a $\frac{1}{8}$ cable is 1.243 ohms per mile.

$$\therefore 1.5 \times 1.243 = 1.864 \text{ ohms,}$$

$$\text{and } E (\text{drop in potential}) = 1.864 \times 120 = 221.74 \text{ volts.}$$

If cable is $\frac{1}{16}$ the resistance is 0.1744 ohm per mile.

$$\therefore 1.5 \times 0.1744 = 0.2616 \text{ ohm,}$$

$$\text{and } E = 0.2616 \times 120 = 31.39 \text{ volts.}$$

The resistance of a conductor is *directly* proportional to its length, and also to specific resistance of material ; and *inversely* proportional to area of cross section.

Resistance (R) of a Conductor.—This may be found from the formula

$$R = \frac{l}{a} \times k \quad \text{or} \quad \frac{E}{C} = \frac{l}{a} \times k,$$

where R = resistance in ohms.

l = length of conductor in inches.

a = cross sectional area in inches.

k = specific resistance (the value of k for different materials may be obtained from tables).

Example.—What resistance has a copper conductor 1000 ft. long, having a sectional area of 1.5 sq. in. ?

$$\text{Value of } k \text{ for copper} = \frac{0.63}{1,000,000},$$

$$\therefore R = \frac{0.63}{1,000,000} \times \frac{1000 \times 12}{1.5} = 0.005 \text{ ohm.}$$

HORSE-POWER OF ENGINES, DYNAMOS, AND MOTORS.

Symbol.	Name.	Derivation.
H.P.	Horse-power	33,000 foot-lbs. per minute.
I.H.P.	Indicated H.P.	33,000 foot-lbs. per minute developed in cylinder of engine.
B.H.P.	Brake H.P.	33,000 foot-lbs. per minute mechanical work.
E.H.P.	Electrical H.P.	746 watts in electrical circuit.

An engine develops I.H.P. in cylinder and delivers B.H.P.

Engine efficiency = $\frac{\text{B.H.P.}}{\text{I.H.P.}}$

A dynamo receives B.H.P. and delivers E.H.P.

Dynamo efficiency = $\frac{\text{E.H.P.}}{\text{B.H.P.}}$

Cables receive E.H.P. and deliver E.H.P.

Efficiency of transmission = $\frac{\text{E.H.P. output.}}{\text{E.H.P. input.}}$

A motor receives E.H.P. and delivers B.H.P.

Motor efficiency = $\frac{\text{B.H.P.}}{\text{E.H.P.}}$

LOSSES IN TRANSFORMATION AND TRANSMISSION OF POWER.

The proportion of loss depends upon the size of the plant, but the following may be taken as typical for 100 B.H.P. delivered at a distance of half-a-mile ; the E.M.F. of the motor being 450 volts :—				
Horse-power delivered to	Horse-power delivered from	Efficiency per cent.	Loss per cent. of Total.	Total Loss.
Dynamo, 138 B.H.P.	Dynamo, 127 E.H.P.	92	8	} 27 per cent.
Cable, 127 E.H.P.	Cable, 114 E.H.P.	90	8·7	
Motors, 114 E.H.P.	Motors, 100 B.H.P.	88	10·3	

CURRENT-CARRYING CAPACITY OF CONDUCTING CABLES.

Number of Strands and Standard Gauge.	Sectional Area of Copper. Sq. in.	Resistance per 100 yds. of Single Cable.	Maximum Current Permissible.	Current at 4 Volts. Loss per 100 yds. Distance. (Length × 2.)
7/18	·01254	·1918	21 amps.	10 amps.
7/16	·02227	·108	33 "	20 "
19/18	·0339	·0707	47 "	30 "
19/16	·06039	·0398	75 "	50 "
19/14	·09442	·0254	108 "	80 "
19/13	·125	·0194	136 "	100 "
19/12	·1595	·01507	166 "	130 "

CHAPTER VIII.

MODES OF WORKING.

Choice of Methods.—The main object in any system of working is to extract as much of the coal as possible, with the maximum of economy and safety.

The two principal methods of working are pillar and stall (stoop and room in Scotland), and longwall, all other systems being simply modifications or combinations of these two.

The method of working any seam naturally depends on local circumstances in each individual colliery, and, speaking generally, varies according to the thickness of the coal. No definite rule can be laid down on this basis, as the conditions vary so much in the different coal-fields, and even in adjoining collieries. Seams of 4 ft. and upwards may be considered suitable for working by pillar and stall, while seams having a thickness of less than 4 ft. are more suitable for working by longwall; but, as stated above, no hard and fast rule can be fixed, for so much depends on other circumstances besides the mere thickness of the seam. The writer has seen seams of 3 ft. thick worked by the pillar and stall system, and seams 5 and 6 ft. thick worked longwall. In the Fifeshire coal-field it is not uncommon to find seams 8 to 14 ft. thick being worked longwall, but these are exceptional cases.

Besides the thickness of the seam, the mode of working will depend on other circumstances, such as—

- The inclination of the strata and the nature of roof and pavement.
- The depth of the seam from the surface.
- The chemical and physical properties of the coal.
- The natural cleavage of the coal and that of the rocks forming the roof.
- The presence or absence of water.
- The vicinity of other seams or of other workings which should not be interfered with.
- The number of dykes and dislocations in the field to be worked.

Longwall.—In this system the whole of the mineral is usually wholly extracted in one operation, and the excavated area is either wholly or partly stowed or packed with the débris made in the getting of the seam and the ripping down of the roof in the road-

ways. In some cases a modification of the system is adopted, and pillars are left along the main haulage roads to help to maintain them, but it is very doubtful if this is an advantage, as it generally entails the loss of a large percentage of coal, which might otherwise have been got, and, as far as the security of the road is concerned, it would seem to do more harm than good.

To work a seam to the best advantage by this method, there must be—

A fairly good roof and pavement.

The roof and pavement should be free from water.

The seam must be neither too thick nor too highly inclined.

The coal itself should neither be too soft nor too friable.

Sufficient débris for proper stowage.

A very soft, friable seam with a hard roof and pavement is usually unsuitable for longwall, no matter what its thickness may be.

Thick seams, worked on the longwall system, are generally more dangerous, from falls of roof and sides, owing to the roads being imperfectly built for want of proper stowage; they are also more expensive as regards timber, but, as a rule, a larger percentage of round coal is got than by pillar and stall system. Longwall may be practised in two different ways, viz.: (a) working direct from the shaft pillar to the limits or boundary of the area to be worked, excavating the whole of the seam. This is known as the advance system. (b) Driving out narrow roadways to the boundary, and then working the seam back towards the shaft; this being called the retreating system. The first—the advance system—is almost universally employed, and on the whole gives the best results. Under certain conditions it might be more suitable to adopt the retreating method, such as in seams liable to spontaneous combustion, or if the roof and pavement are bad and the roadways difficult to keep open.

Laying Out the Workings.—The workings are generally laid out in a regular manner, after the seam has been opened out, the main haulage or drawing roads being set out to the full dip or rise, with side or branch roads at right angles to them. Fig. 166 shows a method of laying out the workings in which the main roads are carried to the full dip and rise.

An important factor in determining the direction of the roads is the jointing or cleavage of the coal. These joints or cleats cross one another at right angles, but there is always one direction along which the coal yields more easily than in any other, known as the main cleat or back. As a general rule, it is better to drive roads at right angles to the line of main cleat ('on plane'), and not to set off a distance in advance, extending beyond the next main cleat. Sometimes, however, if the seam is soft and friable, it is better to work the roads parallel to the main cleat, and it also suits the inclination of the roads to work sometimes along the bedding. If the cleats are

good and the coal soft, it is often best to drive the places or walls 'half on end' (i.e. at an angle of 45° to the line of main cleat). Joints in the roof sometimes coincide with the cleats in the coal, when, if the walls are driven parallel to them, the roof becomes bad and dangerous, and repair of the roads difficult and expensive.

Length of Walls.—The length of wall depends on the thickness of the seam and the amount of material at disposal for 'packs,' and may vary from 10 to 60 or 80 yds. It is difficult to state any definite length of wall which would be suitable for the same thick-

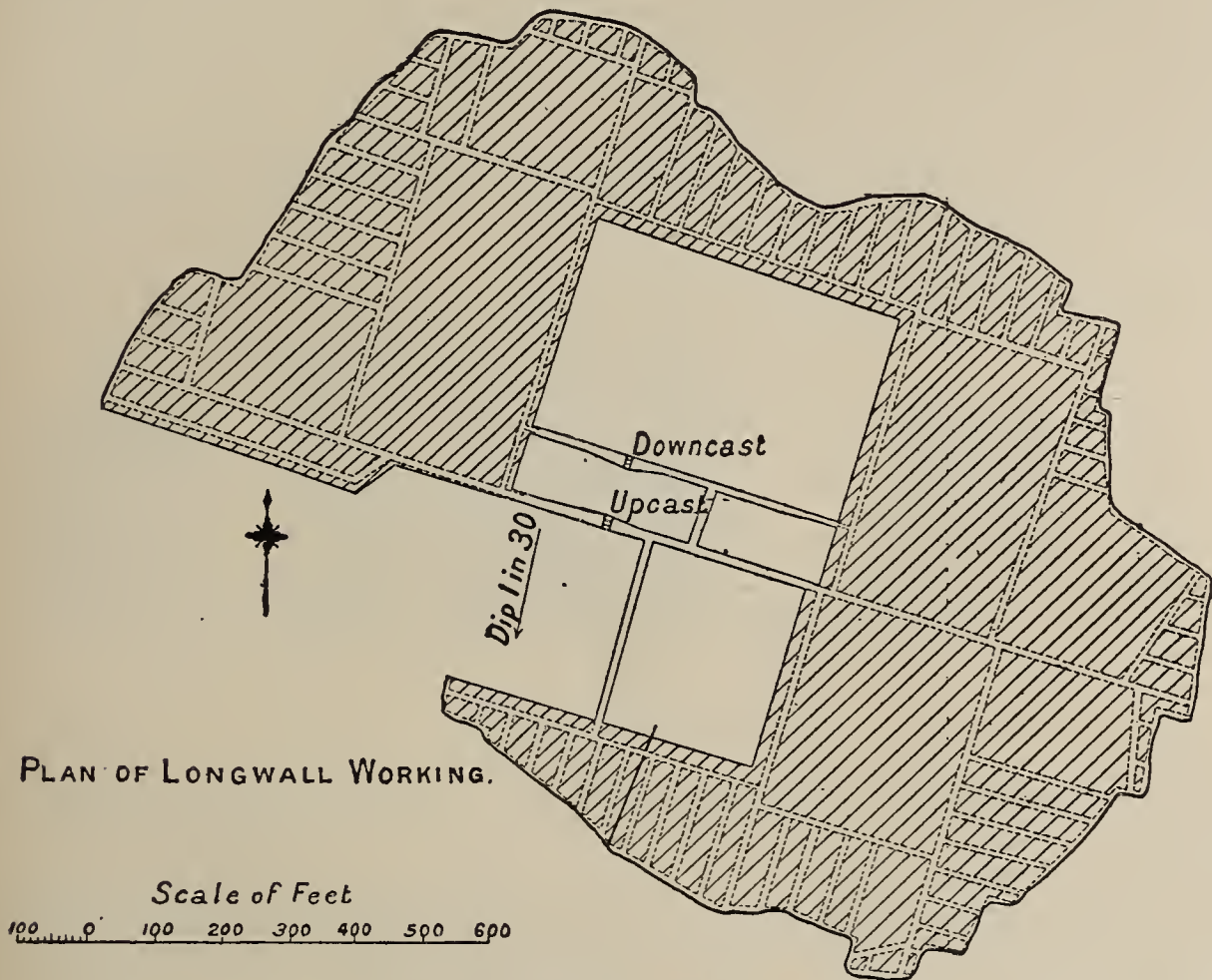


FIG. 166.—Longwall working.

ness of seam under varying conditions. For a seam 4 ft. thick, with a good roof, 12 to 15 yds. is quite long enough; for a $3\frac{1}{2}$ ft. seam, 15 to 20 yds. is sufficient; and for seams $1\frac{1}{2}$ ft. to 2 ft. thick, the wall may be 20 to 25 yds. in length. In some districts with moderately thick seams, 4 to 5 ft., the walls are often as much as 40 to 80 yds. in length, and a tram road laid along the face.

In thin seams the walls ought to be long enough to hold all the débris, and the longer the walls the less will be the cost for ripping. If, however, the walls are too long (in thin seams) the coal is much injured by breakage, in the process of throwing it two or three times along the wall to the road-head or gate-road.

Width of Roads.—The width of roads varies from 6 to 10 ft., according to the depth, nature of roof and floor, and thickness of

seam. They should be at least 7 ft. wide and $5\frac{1}{2}$ ft. high, as narrow roads give much trouble, owing to the packs getting squeezed out and catching on the tubs. For seams 3 to 4 ft. thick it is quite a common practice to make the roads 9 to 12 ft. wide, to enable sufficient débris being got to put in a substantial packwall along the face.

Size of 'Buildings' or Packs.—The size of packs along the face varies from 6 to 12 ft., according to depth, but no pack ought to be less than 6 ft. along the face. The size naturally depends largely on the amount of rubbish available in the workings, but a good-sized pack for a 4-ft. seam at a depth of 120 fms. would be 12 ft. along the face. The size of pack will depend largely on the amount of refuse got in the working of the seam, and in its thickness. In seams where there is a large amount of rubbish got in working and in ripping the roads, the whole of the space from which the coal has been extracted may be completely stowed or packed.

The side or stall roads should not be kept too long in use, as they

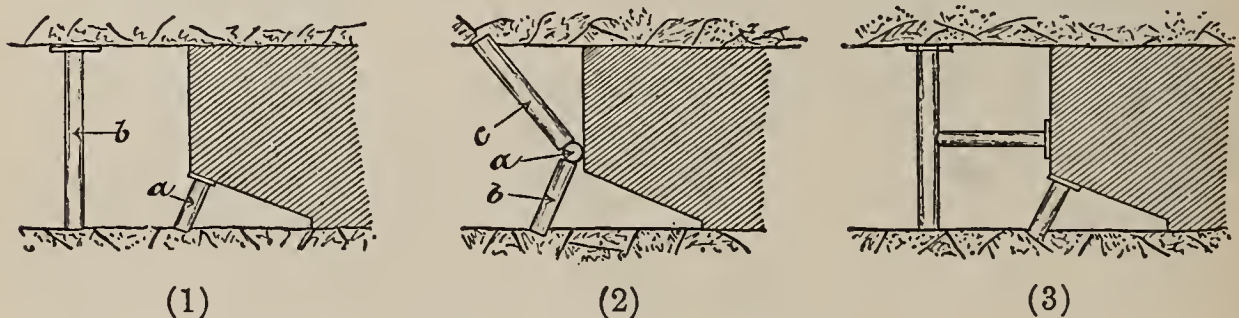


FIG. 167.—Spragging.

often become dangerous and require frequent repairs to keep them open.

Level or slope roads may be set away every 50 fms., or, where the roof is bad, every 35 or 40 fms. The cost of upkeep may be lessened by ripping anew every tenth or twelfth side road, after complete subsidence, and converting them into main drawing roads. By this means rails and sleepers are economised and ventilation is facilitated. The re-arrangement certainly entails additional expense, as it will cost 6s. to 7s. 6d. per fathom for back ripping these roads, but the gain will in other ways more than compensate for this outlay. The cost for ripping in ordinary workings varies from 3d. to 6d. per ton of coal produced, but, as a general rule, it should never exceed 6d. per ton.

The working places ought to be carefully propped and no more timber left in the waste (goaf) than can be avoided, as it keeps the roof from subsiding properly, and needlessly increases the cost. The coal ought also to be properly spragged with 'holing props' to get the full benefit of the 'weight' when it comes on the coal.

Fig. 167 shows the methods of using the holing props. When the coal is soft and difficult to keep up while it is being holed, the method

shown in fig. 167 (2) or (3) is adopted, while, in ordinary circumstances, the method shown in fig. 167 (1) is used. This simply consists of setting up a series of short props or sprags *a*, either with or without a lid, along the face at the edge of the holing. When the coal is soft a horizontal prop (fig. 167 (2), *a*) is placed parallel to and against the coal, a short prop, *b*, supports it from the floor, while another prop, *c*, set at an angle to the roof, helps to keep it in position. The same result may be attained by using the method shown in fig. 167 (3). The holing props should be set up at intervals of not more than 6 ft. apart, and this rule ought to be strictly enforced, as a great many accidents from falls of coal are due to the neglect of using holing props (see Coal Mines Act as to rules for spragging the coal). By properly spragging the coal a larger amount of round is obtained and a smaller percentage of dross than when this precaution is neglected.

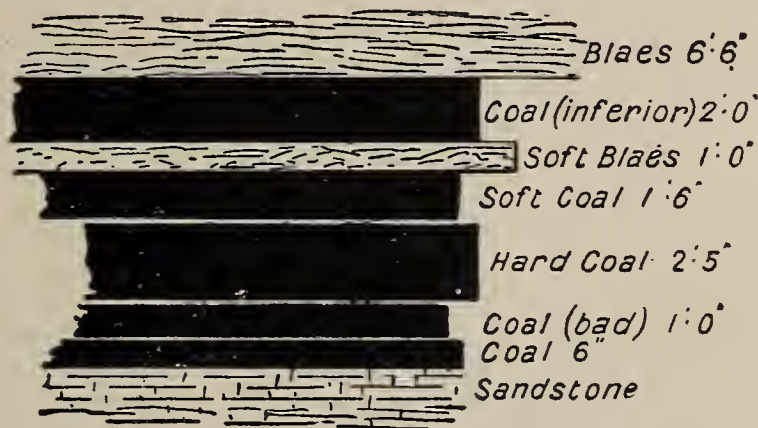


FIG. 168.—Nottingham 'Top Hard' seam.

In some districts where longwall is practised, the walls are made as long as possible, 25 to 60 or even 70 yds., and a road laid along the face to take in the tubs. This is a decided advantage where a large area of coal can be opened out, as it saves expenditure for ripping, and few roads require to be kept open for a given length of face.

In the 'Top Hard' seam at Nottingham, of which a section is given, fig. 168, the method of working is longwall, the walls being 25 to 30 yds. in length. The 'holing' is done in the soft blaes,* and sprags are put in every 6 ft. The places are worked on 'end,' and no blasting is required; when it is worked on 'plane' the coal is more crushed. The rubbish from the walls and ripping is sufficient to pack the whole of the waste, and a pack is put in when there is 4 ft. of ripping available. Two rows of wood, 5 in. diameter, are kept between the pack and the coal, the back row props being placed at a distance of not more than 6 ft. apart. The tram road is laid between these two rows of props (fig. 169). The coal having been holed, the whole length of the wall and the rails are lifted, a 'cut' is taken

* Argillaceous shale.

simultaneously right and left along the wall, the rails being laid down anew from either side at the same time, and a new pack of 5 ft. is put in (at X).

The props of the back row are drawn and set up at the face again. The branch roads are cut off every 50 fms. by new slopes, and the top coal is taken down by these roads. In the main roads the $6\frac{1}{2}$ ft.

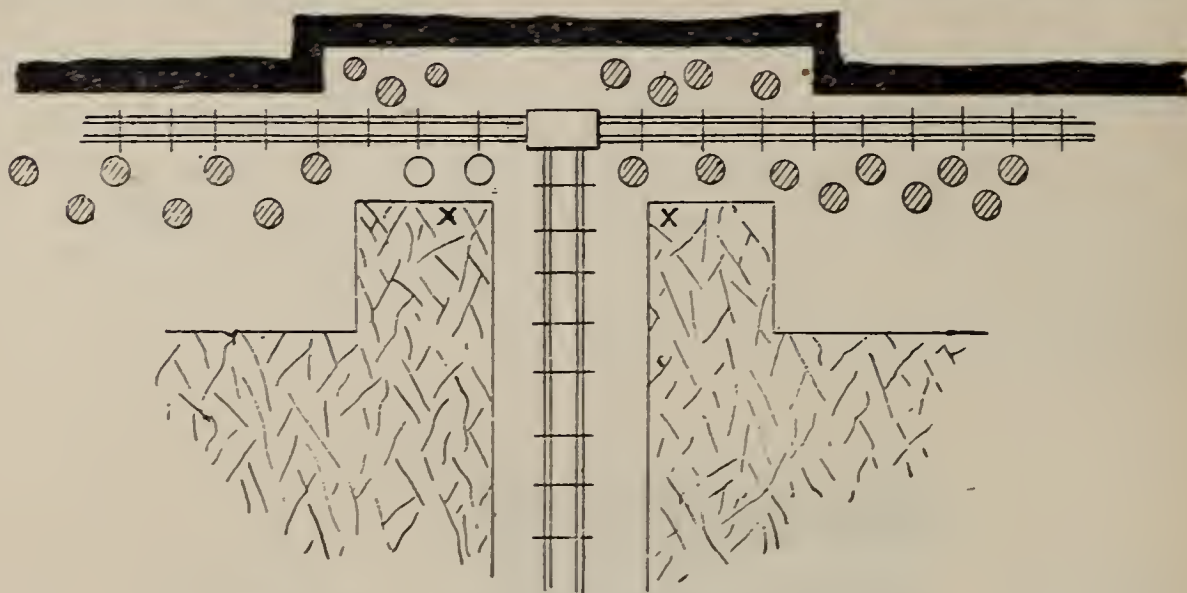


FIG. 169.—Plan of wall showing tram road.

of shale is also taken down, a good high-road resulting. The output per man averages four tons per day.

In the Main seam at the same colliery the working is similar, but the walls here are 60 yds. long, and the seam is worked on 'end' as before, no blasting being required. The under coal and fireclay are holed to a depth of 5 or 6 ft., and sprags put in.

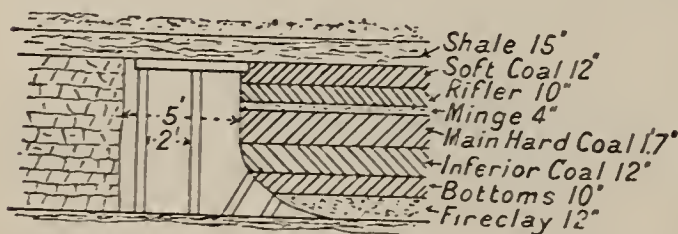


FIG. 170.

The rubbish got from ripping and holing is not sufficient to fill the whole of the goaf, and packs 6 ft. thick and 9 ft. apart are put in at intervals along the wall, while two, and sometimes three, rows of props of 6 in. or 8 in. diameter wood are set up, 6 ft. apart, with a

distance of 3 ft. between the rows; the back props being drawn and shifted forward as packing proceeds. The tubs are taken along the face between the inner row of props and the coal. Where the roof is tender, straps composed of old rails are put up between the coal and the props in the nearest row, the inner end being wedged into the coal.

At High Park Colliery, Langley Mills, Notts, the Top Hard seam has a total thickness of 5 ft. 2 in., with 12 in. of underclay. Fig. 170 shows a section of the seam. The gradient is 1 in 22, and the roadways are worked to the rise, and are cut off by levels every 400

yds. (fig. 171). The roads are set away at 50 yds. centres and made 12 ft. wide. Packwalls 6 ft. wide are put in at intervals of 10 ft. along the line of face, the packs being kept within 4 ft. of the face when holing is begun, and there is a row of props 8 in. diameter set $4\frac{1}{2}$ ft. apart along the face. The props are set up with lids or caps 12 in. $\times$ 6 in. $\times$ 4 in., made from broken props and put in by the stallmen.

The place being in this condition the holers begin at the centre and go on holing to both sides of the roadway, putting in sprags 2 ft. long and 6 ft. apart. Where necessary, short sprags 15 in. long are put in underneath the coal. The holing is carried in to a depth of 4 to 5 ft. While the coal is standing on sprags a shearing or vertical cut is made at the road-head, 2 ft. wide at the outside and tapering to 3 in. about 5 ft. in. After this is done the stallman withdraws three or four sprags, and allows the coal they supported to fall. This fallen coal is then filled into tubs and the tram road laid

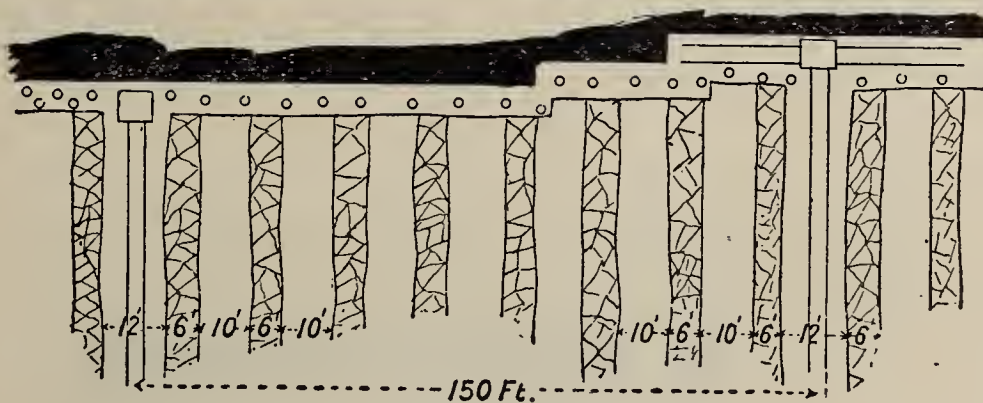


FIG. 171.

along the face, props and cross-timbers or crowns being set up over the rails about every $4\frac{1}{2}$ ft. The one end of the crown is let into the coal for 3 to 4 in., and supported at the other end by a prop. When this is done other sprags are withdrawn, the coal filled, and the process repeated until the end of the stall on both sides of the roadway. There are two rows of props 5 ft. apart behind the sets of cross-timbers before the packs are shifted forward. In each stall there are three stallmen, and they employ loaders and holers, and do all the other work in the place, ripping and building in 3 ft. of ripping, and the coal is drawn to the nearest pass-bye, which is never more than 50 yds. from the face.

The Warren seam in Yorkshire is about 8 ft. thick, a section of which is shown in fig. 172. The system of working is longwall, the roadways being put in at 80 ft. centres, and driven to the rise and parallel with the planes. The gate-roads are cut off by levels 80 yds. apart. Packwalls, 9 ft. wide, are put in along the gate-roads, and smaller packs, $4\frac{1}{2}$ ft. wide, are put along the face at intervals of 15 ft. (see fig. 173). The centre portion is taken out first to a depth of 4 ft., and vertical sprags set between the bottom and middle coal.

Other sprags, set up on angle, are also set to the middle coal to support it. When a length of centre portion is taken out the two sets of sprags are withdrawn, and the middle coal brought down. After this has been filled away the bottom portion is lifted and filled. Owing to the shale above the seam being of a soft nature, the top coal, 2 ft. thick, is left on for a roof. This top coal is afterwards dropped between the packs, the props being withdrawn to allow it to fall. The portion of top coal over the packwalls is not recovered.

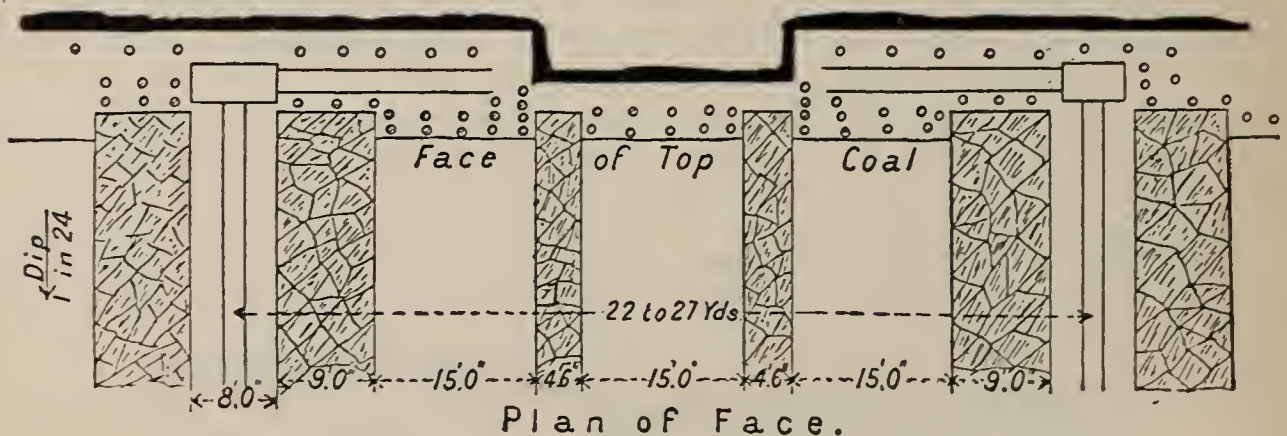


FIG. 172.

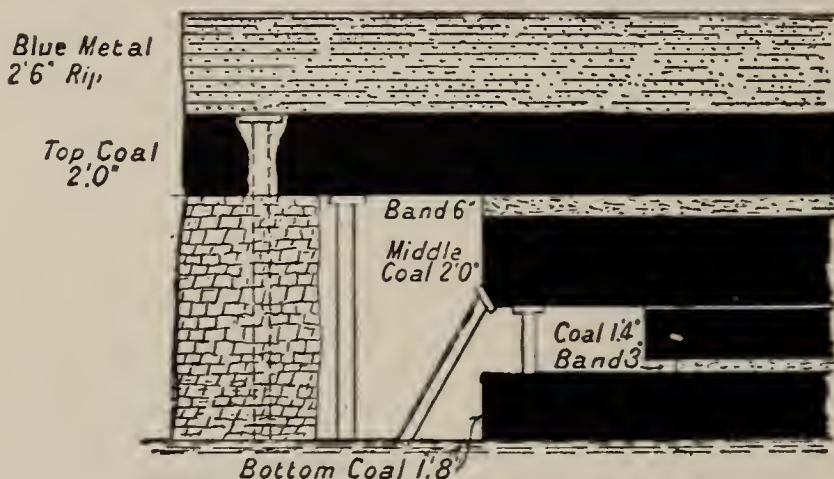


FIG. 173.

The 'Drumgray' seam in Lanarkshire varies in thickness from $1\frac{1}{2}$ ft. to $2\frac{1}{2}$ ft., and is worked longwall. The seam in some districts lies very level, and the general mode of working is to set off roads in the direction of the rise and drive branches, right and left, the walls being 15 to 20 yds. long. The coal is sometimes hauled by boys along the face in small bogies to the 'road-head,' and there filled into the tubs. The roads are ripped to a height of 5 or $5\frac{1}{2}$ ft., and made $7\frac{1}{2}$ ft. wide, the rubbish got from the ripping and holing being sufficient to pack the whole of the goaf. The ripping of the roads costs 7s. 6d. to 10s. 6d. per fathom, and frequent repairs are necessary to keep them in order.

At Westrigg Colliery, near Bathgate, the Drumgray seam is only

14 to 17 in. thick, with a hard rock roof. It lies flat and does not contain much water, although the other seams at this colliery are very wet. Owing to the roof being strong, very little timber is required, but the roads soon get very low. The holing is done in the coal, the floor being hard shale. Each man keeps his own place in order, and does the ripping of his own road, the price paid in 1898 being about 4s. per ton of coal got. This price includes the ripping of roads and the drawing of the coal. The output per working place is about three tons per shift of nine hours. To work so thin a seam would be almost an impossibility with a bad roof, as the cost of upkeep would be too great; but, in this instance, very little extra expenditure is required, the fireman alone being generally able to examine and keep fifteen to twenty places going without much trouble.

To secure success in working longwall it is necessary that—

The places or walls should be kept going regularly, and fully equipped with the full complement of men, otherwise some of the places will fall behind and will cause trouble with the ventilation, and in other ways.

The line of face should be kept as even as possible, and unless the seam is very highly inclined, not worked zigzag, as the point or 'nose' is sure to get crushed, the 'rib-side' will not have sufficient weight, and very often the wall will get closed at this point altogether, hindering ventilation and causing trouble and expense in 'winning-out' again.

No portion of the face should be in advance of the rest further than a single 'cut,' as this makes the coal more difficult to get, and more dross ('slack') will be made by the increased 'shearing' and 'holing.'

The places should be carefully propped, while no timber should be left in the waste that can be avoided, as it keeps the roof from subsiding properly, besides increasing timber costs.

The holed coal ought to be carefully spragged by holing props or 'gibs,' to get the full benefit of the weight when it comes on.

The ripping in each road should always be kept well forward and the buildings (packs) well and tightly laid. If they are loose, much trouble is often caused by their being pushed out into the roads when the weight comes on them.

Advantages of Longwall.—The advantages to be derived from working any seam by longwall may be briefly stated thus:—

The coal is generally extracted with only 5 per cent. to $7\frac{1}{2}$ per cent. loss, resulting from places 'closing' and requiring ribs of coal left in, and from bottom pillars required.

As the 'shearing' is confined to one or two places or main roads, there is a considerable saving in that part of the work, and, therefore, better coal at lower cost is obtainable than where much shearing is required.

The coal is easier to work, and the working price is cheaper, as a rule, than in bord and pillar.

The 'weight' often reduces the labour of getting the coal, for, if properly taken advantage of, it helps to bring down the coal after being 'holed,' thus saving expense in blasting and giving from 10 to 15 per cent. more round coal than pillar and stall working.

Ease in ventilation, small cost for bratticing, and comparatively short distance for haulage for a given output.

More men can be employed in a given area, and, therefore, a larger output can be got than in pillar and stall working.

Fewer roads require to be kept open for a given output, and there is a consequent saving in rails, sleepers, timber, etc.

With thin seams and where the conditions are suitable, coal-cutting machines can be used.

The disadvantages are :—

The roads are more difficult to keep open than in pillar and stall, especially if the roof and floor are wet, and the latter tends to creep.*

Unless the work proceeds regularly, roads and faces are difficult to keep open, and the ventilation is hindered.

Dykes and dislocations are more difficult to deal with than in pillar and stall working, and cause much trouble with the roof and sides.

Longwall working is unsuitable for thick seams with little rubbish available for packs in the wall.

It would seem from the Inspectors of Mines' Reports that there is practically no difference between longwall and pillar and stall, so far as safety and the number of accidents are concerned.

Pillar and Stall, Bord and Pillar, or Stoop and Room.—This mode of working, with its numerous modifications, may be said to be the only other method of working a seam unsuitable for longwall. Seams from 4 ft. thick and upwards may be worked by pillar and stall, and some even thinner than 3 ft., where the pavement is soft fireclay, a portion being lifted, as in some districts of Cumberland. If a seam situated below another working containing water has to be worked, it is often the best method to adopt, no matter what thickness the seam may be. The system, however, is best suited for thick or moderately thick seams, with no available débris. In seams of a soft or friable nature with a rock roof it is likewise, as a rule, the best method of working.

Pillar and stall working is divided into two distinct operations :

Driving places in the solid coal, and dividing the area of coal to be worked into square or rectangular blocks or pillars by narrow places called 'stalls,' 'rooms,' or 'drifts.'

Extracting the pillars and allowing the roof to fall in and fill up the space left. This is usually the most difficult and important part of the work.

The accompanying illustration (fig. 174) shows the pillars or stoops, and the stalls, bords, or rooms.

Size of Bottom Pillar.—The first important point to be considered is the size of the pillar to be left at the bottom of the shaft to protect the surface buildings and the shaft itself from damage. It has been suggested that the whole of the coal might be extracted and no shaft pillar left, but it is only in the case of a thin and valuable seam, probably at no great depth, that it could be treated in this way. Few

* The word 'creep' is usually confined to the slow rising of floors, which sometimes, owing to the pressure of the walls on either side, become more and more convex, and sometimes even block up the road or render it impassable.

engineers would care to take the risk of leaving no pillar to support the shaft, which is sunk to a considerable depth to win a fairly thick seam. The usual practice of leaving a sufficiently large pillar to protect the shaft is, we think, likely to prevail for many years to come. The size of shaft pillars is given differently by different authorities, but none of the rules given can be said to be more than approximately correct, as so much depends on the individual circumstances of each colliery. A larger pillar will be necessary if the overlying strata is of a soft or broken nature, compared with the size

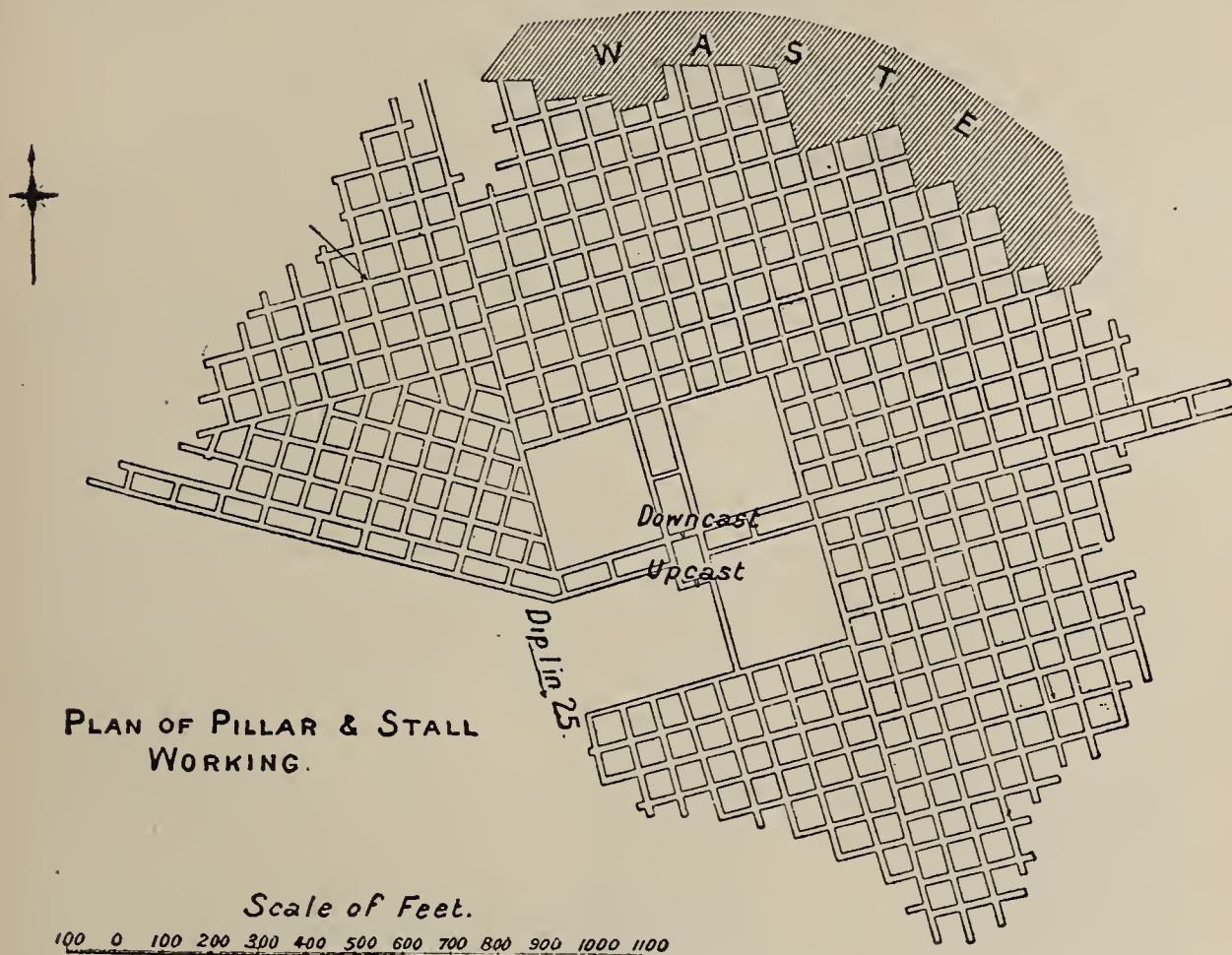


FIG. 174.

required for hard and firm strata. A good-sized pillar is one 40 yds. square for a depth of 50 fms., and the size should increase by 5 yds. for every 10 fms. increase in depth of shaft; *i.e.* the size of pillar for 60 fms. would be 45 yds. square, and for 70 fms. deep 50 yds. square, etc.

Another plan, which is sometimes adopted in Scotland, is to draw a line enclosing all the surface buildings, such as engine-houses, fans, screens, etc., that it is necessary to protect, and make the shaft pillar of such a size that solid coal will be left in all round this line for a distance equal to one-third of the depth to the seam. This rule is more frequently used for determining the size of pillar necessary to support houses or buildings situated on the royalty to be worked.

The size of bottom pillar may also be calculated from the formula $R = \frac{\sqrt{3d} \times \sqrt[3]{t}}{0.8}$, where R = radius of pillar in yards, d = depth from surface in yards, t = thickness of seam in feet.

André gives the following sizes as suitable for shaft pillars:—

Up to 150 yds. deep,	pillar 35 yds. square.
„ 175 „ „	40 „ „
„ 200 „ „	45 „ „

the size increasing 5 yds. for every 25 yds. increase in depth.

A rule frequently used in the steam coal seams of the North of England is to make the radius of the pillar equal to one-fourth the depth of the shaft. Thus for a shaft 1200 ft. deep the radius of the pillar $R = 1200 \times \frac{1}{4} = 300$ ft., and the diameter or side of pillar $S = 300 \times 2 = 600$ ft. = 200 yds.; that is, the shaft pillar would have a diameter of 200 yds., or be 200 yds. square.

The writer's own experience leads him to believe that safety is secured by making the shaft pillar area equal in length in yards to the depth of the shaft in fathoms. For a depth of 100 fms. the shaft pillar should be 100 yds. square; for 120 fms., 120 yds. square, etc. The size of pillar, however, must depend on the nature of the coal, the roof and the floor, as well as the inclination of the seam and the extent and nature of the surface buildings. It must also be remembered that bottom pillars should have an excess rather than a deficiency of coal, as they are frequently cut up later for new haulage roads, etc. In steeply inclined workings at least $\frac{2}{3}$ of the pillar should be left on the rise side of the shaft, as the 'weight' always tends downhill.

The size of pillars in the ordinary working varies very much in different localities and in different seams. It would be difficult to lay down a definite rule for determining the size of pillar to suit every condition, and, therefore, each colliery and each seam must be dealt with on its own merits and conditions. The main idea to be kept in view is to make the pillars of such a size that they will not only efficiently support the roof and keep the roadways open in the first working, *i.e.* the narrow work, or during the time the pillars are being formed, but also prevent thrust and creep in the roof and floor during the second working, *i.e.* the extraction of the pillars. In the Hamilton district of Lanarkshire, in the Ell coal, which averages about 5 ft. thick and is at a depth of 100 fms. from the surface, a common size is 30 yds. $\times$ 20 yds., while for the Splint coal in the same district, which is usually about 6 ft. thick and lies about 20 fms. or so deeper than Ell, the pillar is often 30 yds. $\times$ 30 yds. or 40 yds. $\times$ 30 yds. The pillars left in the first working are much larger now than was formerly the case; often in the first working only 8 to 10 per cent. being taken out, and 90 to 92 per cent. left in the pillars, according to circumstances. At Milnwood Colliery, Bells-

hill, in the Splint coal, only $8\frac{1}{2}$ per cent. was taken out and $91\frac{1}{2}$ per cent. left in the pillars.

For determining the size of pillars the following rule may be used:—Allow 6 sq. yds. of pillar per fathom depth for a 4-ft. seam, 8 sq. yds. per fathom for a 5-ft. seam, and 10 sq. yds. per fathom for a 6-ft. seam, etc.; multiply this allowance by the number of fathoms in depth, and extract the square root of the result. In the case of a 4-ft. seam at a depth of 120 fms., we have $120 \times 6 = 720$; $\sqrt{720} = 26.8$ yds. square. Or the following approximate sizes may be given:—

At a depth of	50 fathoms	leave pillars	20 yds. square.
„	80	„	23 „
„	100	„	25 „
„	150	„	30 „

and so on, increasing size of pillar by 1 yd. square for every increase in depth of 10 fms. These sizes are based on the understanding that the roof and floor are of average hardness.

The size of pillar may also be ascertained approximately by the formula, $S = \sqrt{\frac{D}{50}} + 22$, where S = size of pillar in yards, D = depth of seam in yards. Taking all things into consideration, pillars as large as possible should be left in during the first working.

If roof and floor are both soft and only small pillars are left, 'creep' will occur* (fig. 175), and a large amount of the coal may be entirely lost. 'Robbing' the pillars in an irregular manner may also bring on 'creep,' and large areas of valuable coal have been lost in this way. Pillars sufficient to prevent creep in the first working may be quite incapable of doing so when the work of extracting has commenced. Again,

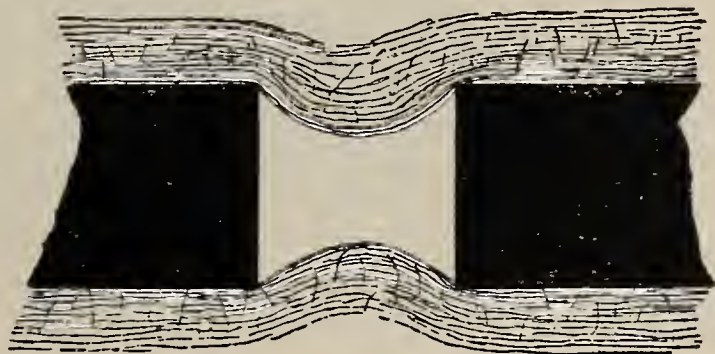


FIG. 175.—Showing creep and crush.

if the roof is bad, the floor hard, and small pillars only have been left, then when a crush comes on the roof will 'spill' or 'ride' over the pillars, necessitating a large amount of 'redding' when stooping is going on, and much of the coal will be crushed into small, of much less value.

If both roof and pavement are hard and small pillars are left, the coal will be much crushed, and a large percentage of dross will be got. When this takes place it is called 'thrust' or crush.

Another point to be taken into consideration in fixing the size of the pillars is the amount of output required daily. Generally speak-

* See note, page 218.

ing, small square pillars are most favourable for a large output, as the stalls are numerous, and help to win out new places speedily.

Direction of Pillars.—Pillars should, as a general rule, be worked lengthwise to the rise of the seam, unless the latter becomes very steep, when it is found more economical to make the long side of the pillars at right angles to the inclination, and so have a larger amount of coal to work from on the level course (fig. 176). The most important point is to have the pillars running parallel with the main cleat, as this secures the best coal at the least cost.

Width of Stalls or Rooms.—This will depend almost entirely on the nature of the roof, but if the latter is good, the stalls should be driven as wide as possible, consistent with the safety of the men.

Wide stalls give a much larger percentage extraction of round coal than narrow ones, which is a very important point, except where the coal is to be coked, in which case it does not so much matter about small coal being produced. With a stall 6 ft. wide there will



FIG. 176.—Direction of pillars.

be an average of 25 per cent. dross (slack); with a stall in the same seam 15 ft. wide, the average would be only about 15 per cent.

In Scotland, the 'rooms' and 'ends' are usually the same width, varying from 9 to 12 ft.; in the Splint coal of Lanarkshire, which has a rather soft roof, the stalls are usually 8 or 9 ft. wide, but in other seams where stoop and room is practised 12 ft. is a common width. In the North of England the 'bords' or stalls are narrow, being only 8 to 12 ft., while the 'throughers,' or places at right angles to the 'bords,' are often as many yards wide. Again, in other districts, it is the practice to open out both 'bords' and 'throughers' narrow (6 to 9 ft. wide for a short distance), and then to increase the width to 8 or 9 yds. No hard and fast rule can be laid down for the width of openings, the main requirement being to keep the stalls of such a width that the minimum of dross or slack will be obtained with the maximum of safety.

Extracting the Pillars.—This is the most important and dangerous part of the work in pillar and stall working, and great care should be exercised in carrying it out. The chief point to aim at is to get out the coal as quickly as possible, without endangering the men or

losing a whole 'lift' ('judd') of timber for the sake of a tub or two of coal. On the other hand, no coal ought to be left in that can possibly be extracted. To satisfy both conditions, the pillars ought to be worked out in a regular manner, beginning next the goaf or boundary, and working forward in regular order (fig. 177).

Too many pillars should not be removed at once. As soon as a section of solid coal has been driven through, it ought to be extracted as quickly as possible, otherwise much loss will take place, and the percentage of round coal will decrease, as pillars deteriorate when left standing long. With the greatest care possible, there is always

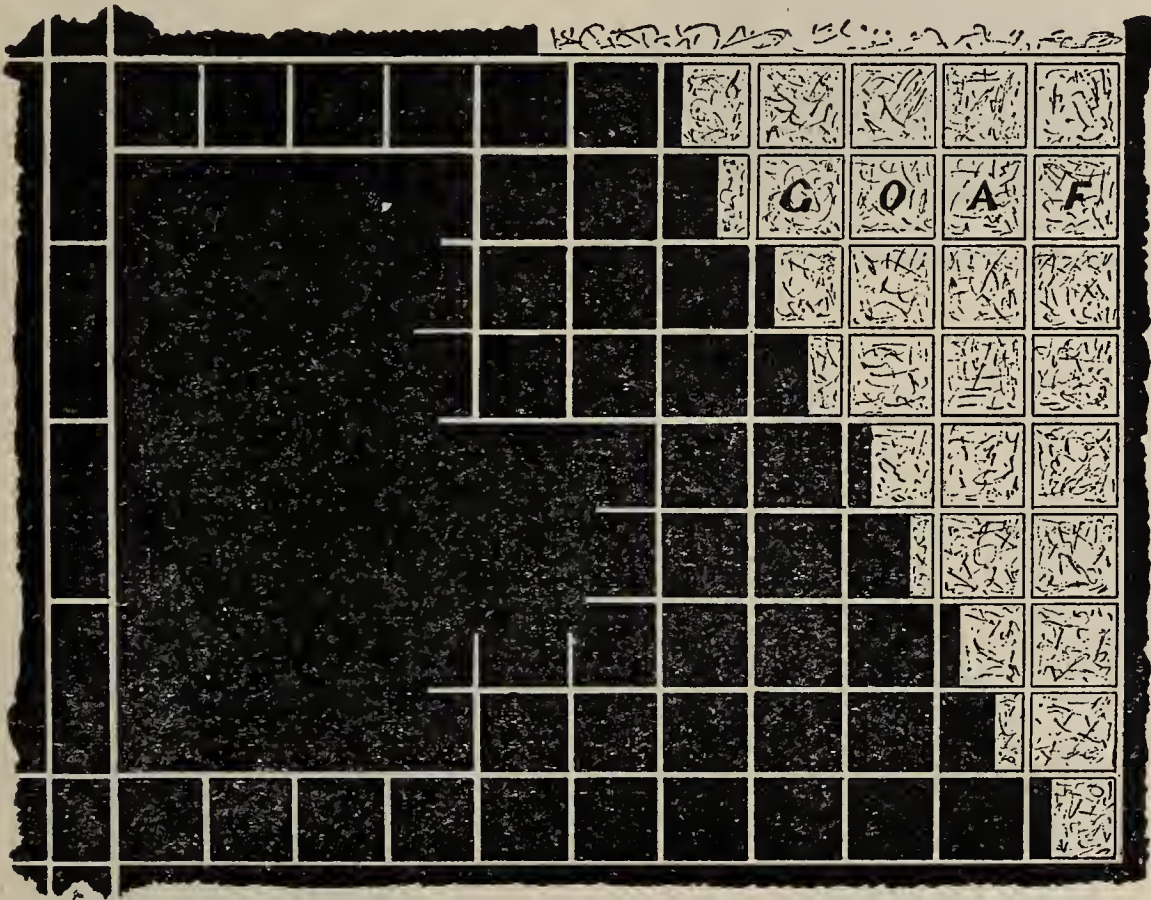


FIG. 177.—Extracting pillars.

some loss, varying from 7 per cent. to 12 per cent., in removing the pillars, and sometimes considerably more is lost through careless working and not setting sufficient props.

The pillars themselves are extracted by taking slices, lifts, or judds off them, varying from 12 to 20 ft. in width,—15 ft. being a very common lift. In this way they are reduced in size till a small pillar about 8 ft. square is left, this small pillar being then extracted as rapidly as possible. Fig. 178 shows one of the methods of reducing the pillars by taking a 'lift' off each side simultaneously.

If the pillars left are large and square, they are often split into two by driving a road through their centre, and then extracting the remainder by taking 'lifts' right and left as shown in fig. 179.

In this way the length of lift is shortened, which in the removal

of pillars is a considerable advantage. Figs. 180 and 181 show other methods of extracting the pillars which are sometimes adopted.

In some districts, instead of the pillars being taken out in lifts,

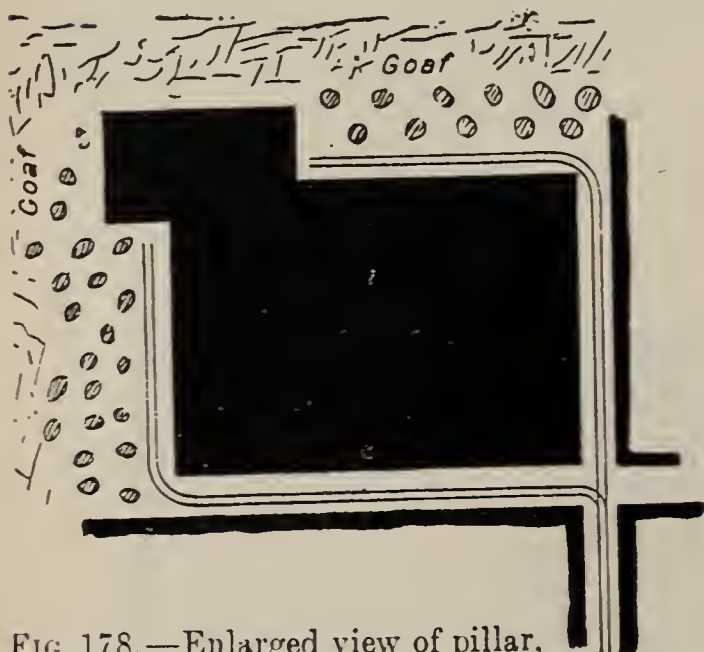


FIG. 178.—Enlarged view of pillar, showing method of extraction.

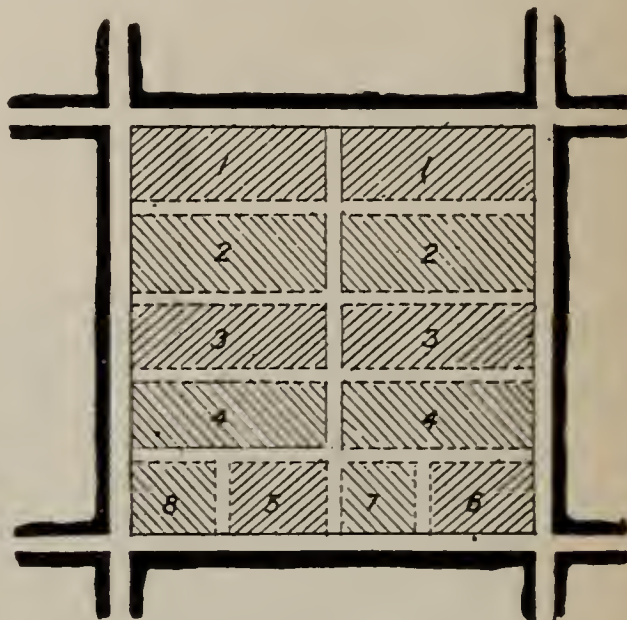
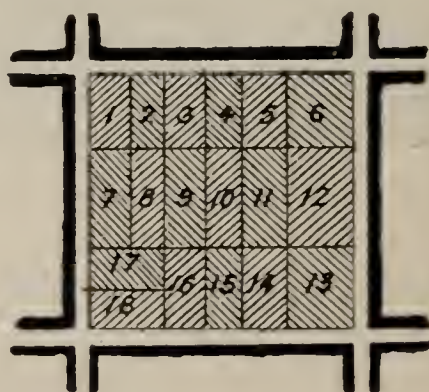
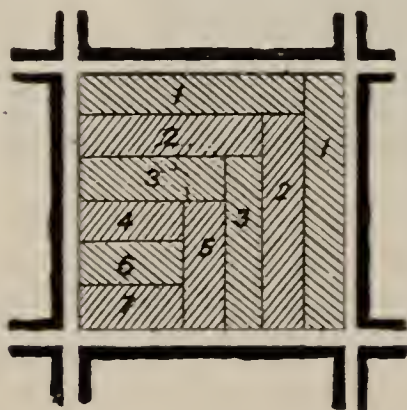


FIG. 179.—Extracting pillars.

the whole pillar is extracted in one operation, and the waste left packed with rubbish as in ordinary longwall working. It is not, however, always convenient or even safe to attempt this method.

When the roof is bad, and fire-damp is given off freely, extracting the pillars becomes a still more dangerous operation. To be success-



FIGS. 180 and 181.—Extracting pillars.

ful in this part of the work, the following rules ought to be carefully attended to:—

No naked lights ought to be used in withdrawing timber, whether gas has been found or not, and the timber should be withdrawn when only a few men are in the pit.

The lifts should be made as short as possible and not too wide.

They should proceed regularly and as speedily as possible.

Two 'lifts' should not meet each other; one should be finished and the timber withdrawn before the other comes forward.

The timber ought to be withdrawn as soon as practical after a 'lift' is completed.

If plenty of timber be used (about one prop per square foot) less will be lost, and a plentiful supply of timber should be kept as near as possible to the working faces.

Panel System.—In the old method of working, the whole royalty to be worked was first cut into pillars right to the boundary line before any of the pillars were extracted; this entailed a large amount of loss, due to the length of time the stoops or pillars stood before

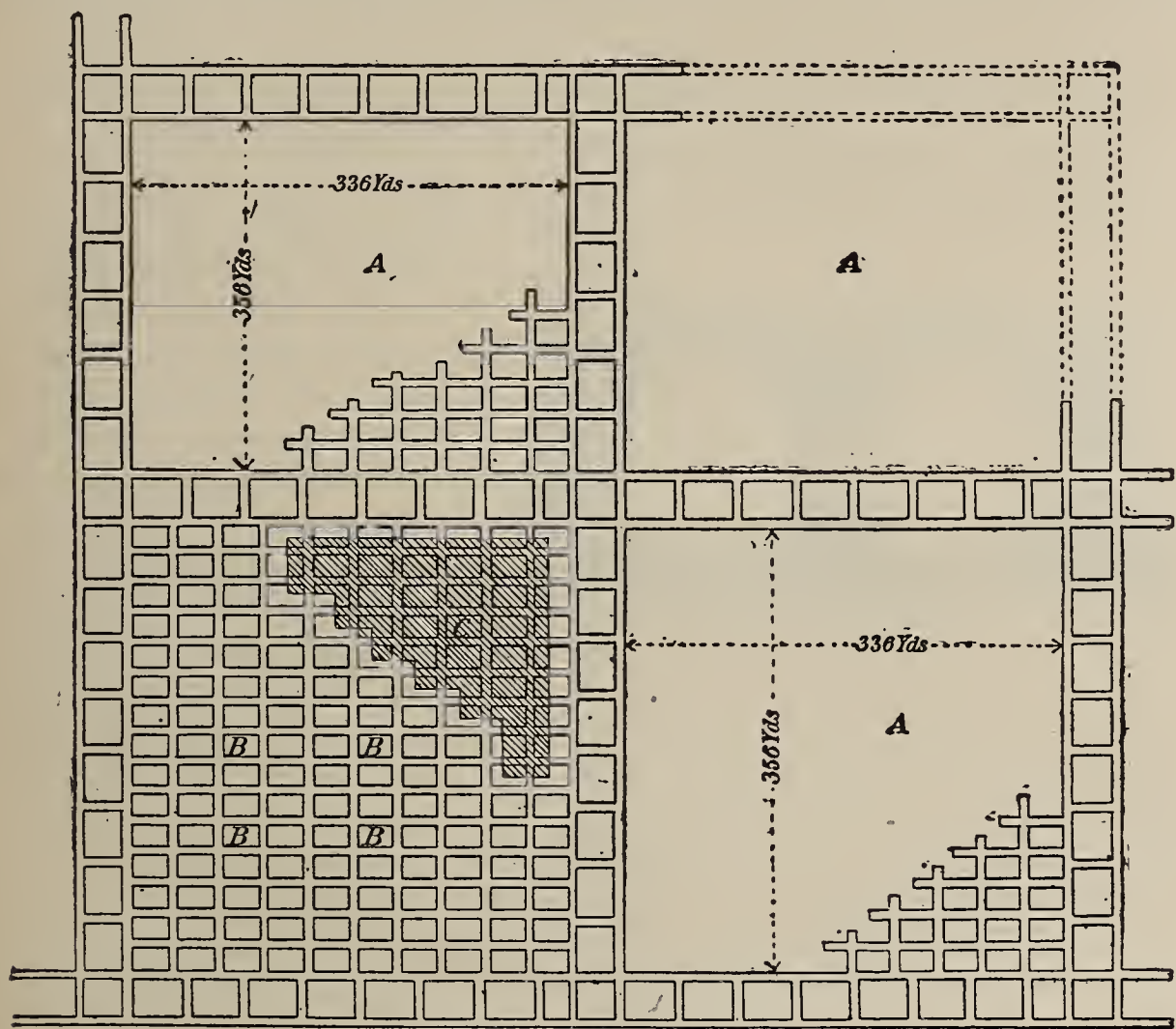


FIG. 182.—Panel system.

AA = Large pillars or panels. *BB* = Small pillars. *CC* = Pillars being extracted.

being taken out, for where the roof was bad or the coal tender, they would be much crushed. To overcome this difficulty, workings are now often laid out in sections or panels, and as soon as one section is turned into pillars, these are at once extracted without being allowed to stand any length of time. This method is found both cheaper and to result in the production of better coal. At Hamilton Palace Colliery this system of working was adopted, the whole field being first divided into large blocks or panels, 336 × 356 yds., by pairs of headings or levels driven close to each other (fig. 182) and 'throughers' driven for ventilation. The levels and headings were

driven by Stanley coal-heading machines, two machines working a level each with only a rib of coal 1 ft. thick left between them (fig. 183). This rib of coal obviated the use of brattice, and served to preserve the roof; the rib was subsequently taken out and the two drivages converted into one level 11 ft. wide, which was quite sufficient for a double road being laid down for haulage purposes. As the large blocks are formed, men are immediately set to work to form smaller pillars, 30 yds. by 20 yds., and as soon as these are driven a third set of men proceed to extract them. The great advantage of this method of working, as compared with that of forming pillars over large areas, is that they only stand for a short time after being formed, better coal being got, while a larger number of men can be employed, and hence a larger output is obtained.

At Clifton Hall Colliery, in the Manchester district, the Doe coal seam is worked on a similar principle. Pairs of levels are driven to

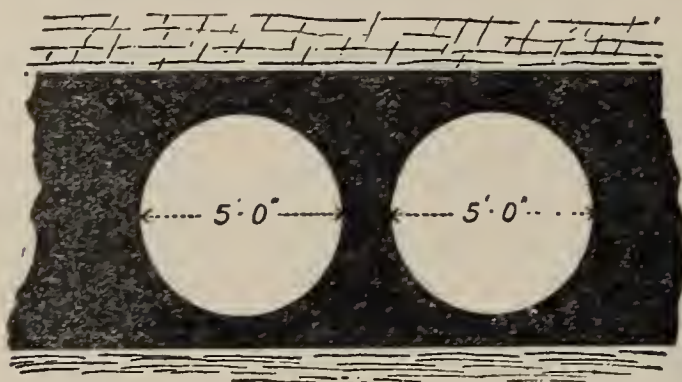


FIG. 183.—Machine levels or headings.

the boundary, with 40 yds. between the openings, and 200 yds. above these another pair of levels is driven parallel. When the boundary is reached, a pair of headings is set away to connect them, thus dividing the coal into large blocks 200 yds. square. These blocks are then sub-divided into pillars 30 yds. square, and when a

block has been turned into pillars the latter are at once extracted, beginning at the boundary and working backwards. Lifts of 12 yds. wide are taken off the pillars, a packing 9 ft. wide being carried up alongside the road—the material for which is inferior coal.

The seam is altogether 9 ft. thick, but the top $3\frac{1}{2}$ ft. is left on for a roof; the holing is done in the 7 in. of fireclay, which is holed for 3 ft. The top coal is then taken down and another 3 ft. holed, until 9 ft. of the bottom coal is bared—this part of bottom coal being obtained by blasting. The lifts off the pillars are always taken to the full rise of the seam.

Sand or Water Pack.—In British mines where seams are worked on the pillar and stall system, it is not usual to pack the waste ground when the pillars are extracted, the timber being usually withdrawn and the roof allowed to collapse. If the seam is thick and situated at no great depth, much damage may be done to the surface when the pillars are extracted, especially in the vicinity of towns, where the buildings may be partially wrecked, of which many instances have occurred throughout Britain. To obviate this the goaf may be packed with sand or fine débris run in with water through a series of bore-holes from the surface. This process was first introduced in America in connection with the working of the

thick anthracite seams, which in many places lie at no great depth from the surface. The practice there is to utilise the fine refuse and sludge from the screening and coal-washing plant, which is sluiced with water into the waste workings.

The system has also been successfully applied in Germany, where in some districts, such as Silesia, thick seams, 20 to 30 ft., have to be worked, and where previous to the introduction of the sand-pack a large quantity of the coal was lost in the removal of the pillars, and much damage done to the surface. The chief material used for this purpose in the Silesian mines is sand, although other material, such as coal-dust, cinders, clay, refuse from coal washeries, ground stones and bricks, are also used. The fine débris is carried down the shafts in pipes, and thence to the required position in the workings, by a stream of water, the water being allowed to filter through the pack and then pumped to the surface. "At the space to be packed underground, the delivery pipe is raised as close to the roof as practicable at the upper end of the open space, and the openings, except that for the delivery pipe, are closed by dams. Water alone is then allowed to flow through the pipes for some time so as to clear them thoroughly, and afterwards the tipping of the material into them, which is done through a funnel, is commenced. The water, after depositing its load, filters away through a dam placed at the bottom end of the pillar, eventually reaching clearing tanks, whence it is pumped to the surface. Meanwhile the material is gradually depositing and being tightened by its own weight, until only the small space round the mouth of the delivery pipe remains unfilled; the flow is then turned off in this direction, and the same operation repeated with it by means of branch pipes somewhere else. It is advisable to have telephonic communication between the tipping and delivery ends of the pipes. After giving the packing a day or two to dry, the dams are removed, and the coal in the neighbourhood can be worked: the packing remains quite tight and firm, and resembles a natural stratified bed."* The cost at one colliery in Germany for packing the waste in this way was about 6d. per ton of coal won, this including the cost of pumping, pipes, interest on capital—everything, in fact, except the sand and water, which were got free. There are few districts in Britain where sand could be had free in sufficient quantities to carry out this system successfully, but at many collieries there are large heaps of refuse which might be utilised for the purpose. The principal advantages claimed for this method of packing are: (a) a larger quantity of coal can be won; (b) little damage is done to the surface; (c) less timber is required.

Working Contiguous Seams.—When two or more seams are close together, with little strata between them, it is often difficult to determine the method of working best adapted for getting all the coal out in good condition, particularly when the strata between the

* *Trans. Inst. Min. Engs.*, vol. xxviii. p. 325.

coal seams are soft and friable, and when the seams themselves are largely intermixed with bands of dirt.

All the circumstances in each separate case must be closely studied before adopting any system, and no hard and fast rule can be laid down as to what would be the most suitable method under varying conditions, as so much depends on the nature of the coal, the strata separating the seams, and also the roof and floor; also the amount of water given off and the amount of fire-damp met with in working. What may suit one set of conditions may not suit a different set of conditions at all.

When two seams are separated by 6 or 10 fms. of strata, it is, as a general rule, best to work the lower seam out first, particularly if much water is given off, and to let the upper one stand for two or three years. By that time the roof will have settled down gradually

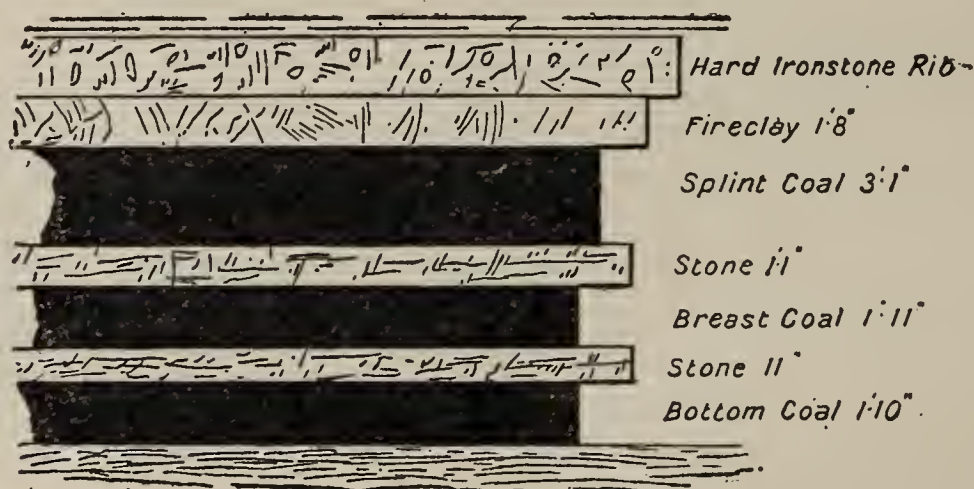


FIG. 184.—Section of seam.

and evenly. This plan may cost more than if the upper seam were worked first, but far better results are obtained as a rule. Of course, there are exceptions; for instance, if the top seam had a bad roof, it might be better to work it first, or there may be a stipulation in the lease that the upper seam be first exhausted. When two seams are separated from one another by only a small thickness of rock, and if this stratum be hard and firm, it is better to work the top seam first, allow the roof to settle down, and then work the lower seam. If the intervening stratum be loose and friable, the bottom seam should be worked first, as otherwise it may be found impossible to work it at all. Then, again, sometimes a number of seams occur together, separated only by thin bands of dirt or inferior coal. In a seam having a section, such as in fig. 184, the bottom portion is worked first up to the bottom of the Splint coal, leaving this on for a roof, while the stone and holings pack the waste, the working being carried on in the regular longwall way until the boundary is reached. Then the top part (Splint coal) is either wrought back from the boundary to the shaft or worked inwards in the usual way, the old roads serving for this working. The main object to be

kept in view in working two seams close together is to get as much of both as possible, and not to damage either by working the other.

In Lanarkshire, the Splint and Virgin and the Kiltongue seams are often divided by bands of dirt varying from 1 ft. to 6 ft. in thickness, which renders them difficult to work, especially when the dirt between the two portions is 4 ft. to 6 ft. thick.

In the Blantyre district, the Splint and Virgin seams are separated by a band of dirt only 1 ft. to 3 ft. thick, and in some of the collieries these seams are worked in two portions. The Virgin seam is worked in the ordinary long-wall method, the walls being 14 or 16 yds. long. The coal consists of the Virgin seam and the ply of stone up to the bottom of the Splint coal; but to give sufficient height in the

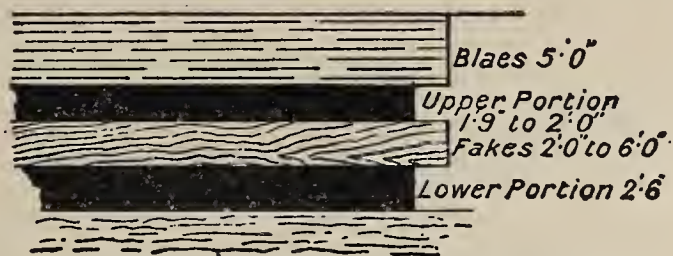


FIG. 185.—Section of Kiltongue seam.

roads a portion of the latter is also ripped down. The walls are packed with the intervening stone, and with any stone resulting from falls of roof on the roads. The Virgin seam is worked in for a considerable distance in this manner, and then the Splint coal is worked back, what was the packing for the lower portion being now the holing for the top part. The one set of roads serves for both seams. By this method of working, round coal is economically got, but the main roads are extremely difficult to keep in repair, owing to the insufficient packing of the walls and the weight from the roof.

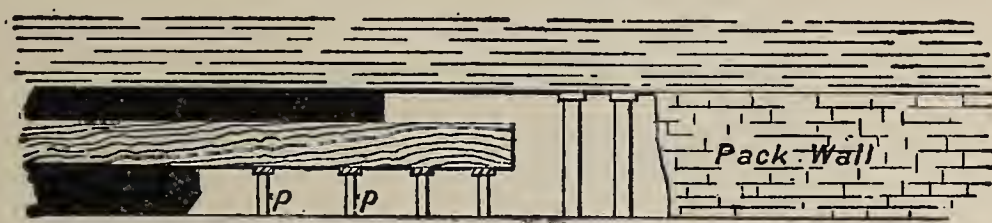


FIG. 186.—Method of working Kiltongue seam.

Fig. 185 represents a section of the Kiltongue seam as it is found in the Coatbridge district. A common method of working it is as follows:—The bottom portion is worked from 9 to 12 ft. in advance of the upper portion, and short props *p p* are put up along the wall to support the intervening strata (fig. 186). When the coal has been taken out in the lower seam, the short props are then drawn, and the fakes allowed to fall, a pack being put in along the face until all the loose material is used up. The top portion is then either blasted or wedged down in the usual way, care being taken to support the roof properly immediately the top coal is removed. This method works very well when the thickness of the strata between the two seams does not exceed 3 ft.; when it is greater it

is dangerous to work by this system, owing to the difficulty of getting out the short props, and the danger arising from large slabs of the fakes breaking off and falling over at the face. Any open spaces along the face should be supported by wood pillars. Another method of working this is to keep both portions going as separate workings, and when the road is ripped to put in a short pack in both seams at the same time.

When the dirt separating the two seams is 5 or 6 ft. thick, the following method of working is often adopted:—The bottom seam is worked first by ordinary longwall. Main levels and headings are set away every 40 or 50 fms., and the drawing roads cut off on one

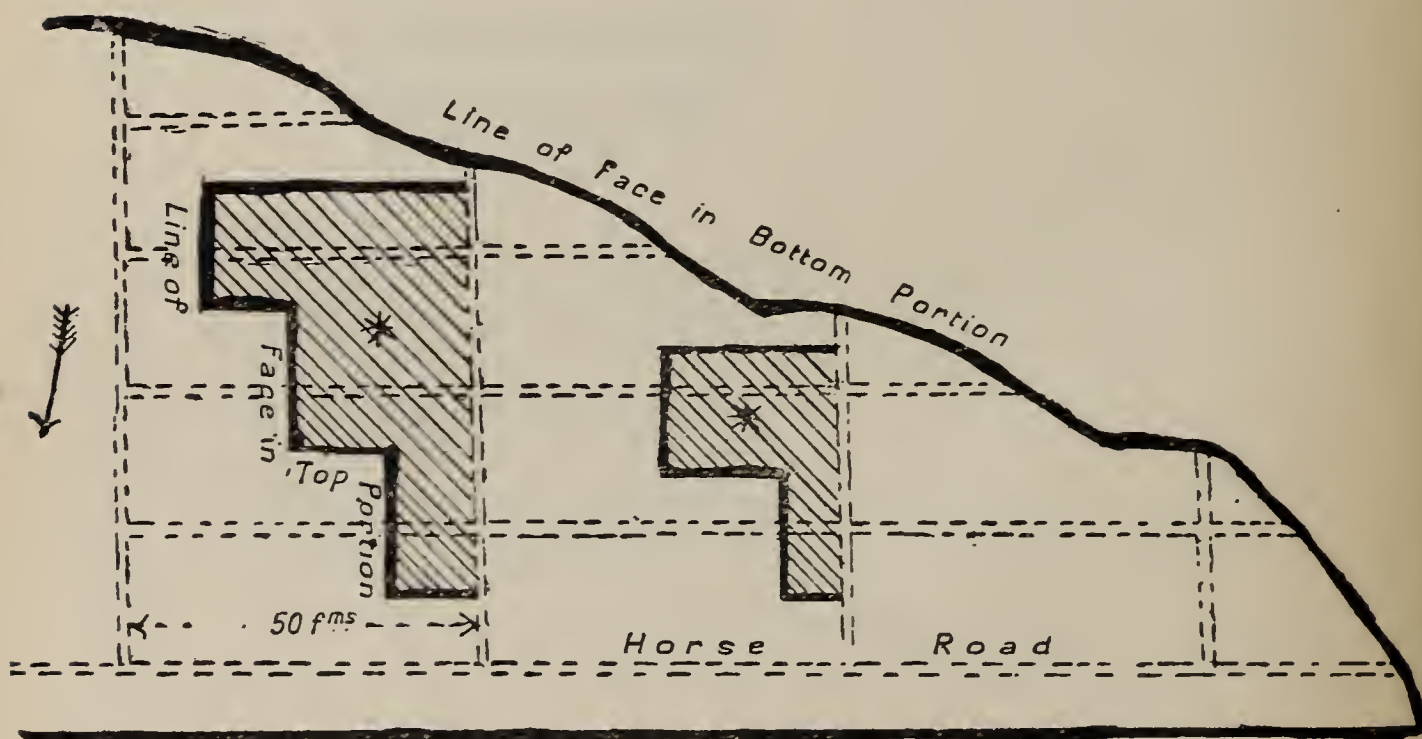


FIG. 187.—Working special seams.

side, and new branch roads are opened on the opposite side of the heading. As the branches are thus cut off in a section, the upper seam is opened up in the old road-heads and the seam worked back towards the main levels (fig. 187), beginning at the level itself, which would require to be worked during the night shift.

By cutting up on the 'low' side of the first branch road and the 'rise' side of the level, means are afforded for an air-current to circulate. In working by this method, the walls and roadways in the lower seam should be built very carefully, otherwise great difficulty is sometimes experienced in reaching the top part of the seam.

The advantages claimed for this method are :

It is safe to work.

Little ripping is required, and keeping roads open is dispensed with.

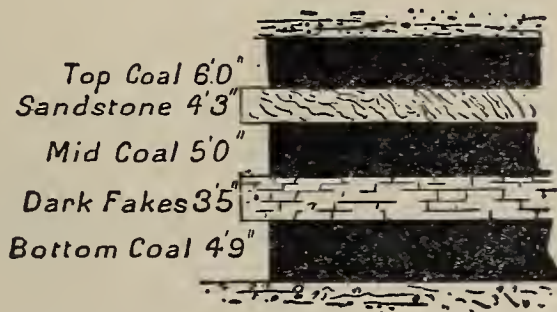
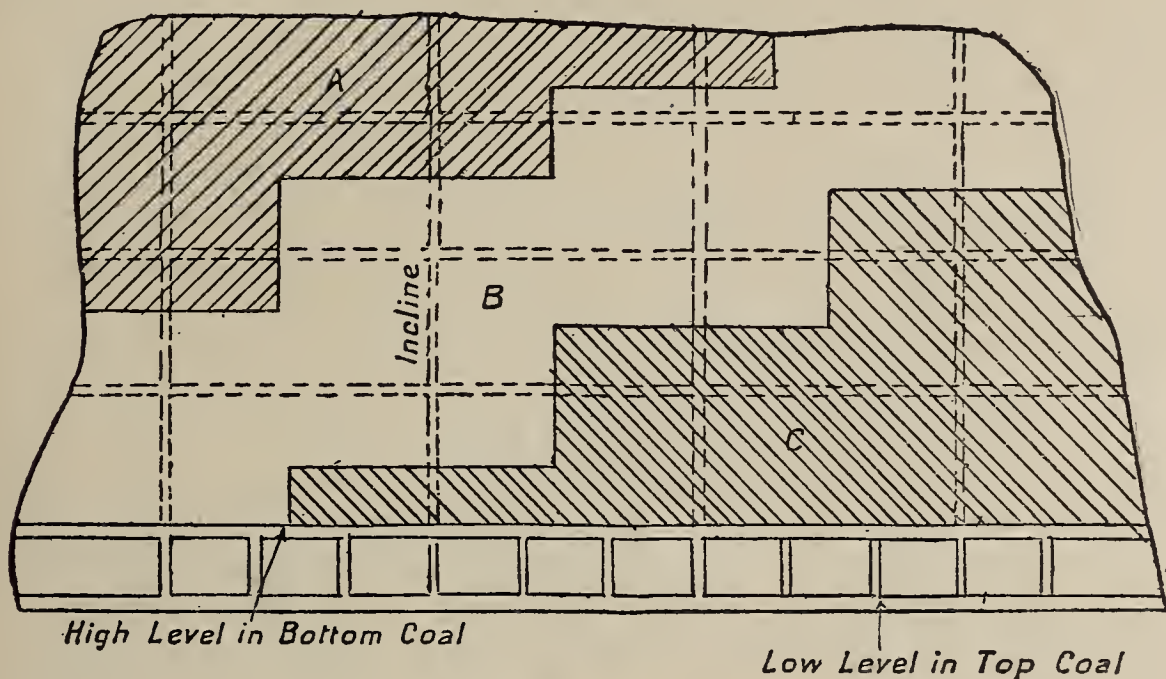
Less expense is incurred for timber.

A larger output is got by working both seams practically at the same time.

Sometimes three seams are found within a few feet of each other, as in some parts of the Ayrshire coal-field. This greatly increases the difficulty of working. The section here given (fig. 188) represents one that occurs at Dalmellington.*

The method of working adopted is very similar to that just described for two seams occurring close together.

In this working two levels are driven—the low or main level in the top coal and the high level in the bottom coal. Sometimes, when the roof of the top portion is not very good, the main levels



FIGS. 188 and 189.—System of working adopted at Dalmellington in Ayrshire.

are driven in the bottom coal. Level cross-cuts are driven at convenient distances—usually 40 to 50 fms.—apart, from the top coal in the low level to the bottom coal in the high level. In this coal longwall workings are opened out, a barrier being left between the two levels, and branch roads 12 yds. apart are set off to the rise at right angles to the level (fig. 188). In the roads of this working the dark fakes are ripped down to the bottom of the mid coal, the roads being packed 12 ft. wide. When the branch roads are worked up 40 or 50 fms. they are cut off by a new level; only those in a line with the cross-cuts are kept open as main

* *Trans. Min. Inst. Scot.*, vol. xiv. pp. 113–114.

roads, being afterwards fitted up as self-acting inclines to convey the coal to the low level.

When a certain area has been exhausted in the bottom coal, a new set of operations is begun at the *cousie* * level, by piercing up into the mid coal and opening out longwall workings in that seam, the roads of the lower seam being used. A pillar is left to protect the cross level, should this still be in use when it is reached. The roads are finally ripped up to the bottom of the top coal, and a pack put in on the top of the dark fakes, which was the roof of the bottom coal, but is now the floor of the mid coal. When the mid coal has reached its limit, the top coal is pierced up along the faces and worked backwards longwall, there being no ripping in this working. Stones from the roadsides and old timber are used for pillars on either side of the road-head (gateway), the walls being propped in the usual way. By this method very little coal is lost, and the percentage of round coal got is larger than when the seams were worked by stoop and room.

Spontaneous Combustion.—In many districts spontaneous combustion occurs when the coal is being worked. In the South Staffordshire thick coal underground fires are of frequent occurrence, and in the Dysart thick seam in Fifeshire the coal very readily takes fire on being worked. These underground fires are more or less due to the presence of quantities of dross or inferior coal in the waste. The theory, or rather theories—for there are many—of spontaneous combustion in underground workings need not be discussed here, and only the methods for dealing with such occurrences will be dealt with. To overcome the danger arising from underground fires, various means have been adopted, but none of them can be said to be wholly successful; in fact, it would be difficult to invent any method which would prevent underground fires under the varying circumstances met with in working different seams. This is amply borne out by the terrible accidents that take place from time to time from spontaneous combustion. Where the seam is worked on the longwall principle, continuous walls of clay have been employed to prevent the air from entering the waste; banks of sand have also been used for the same purpose, but neither method seems to be very practicable. When a ‘crush’ came on, a clay wall would, in all probability, be pushed out into the roadway, while such walls would certainly entail a large amount of extra labour and expense in construction.

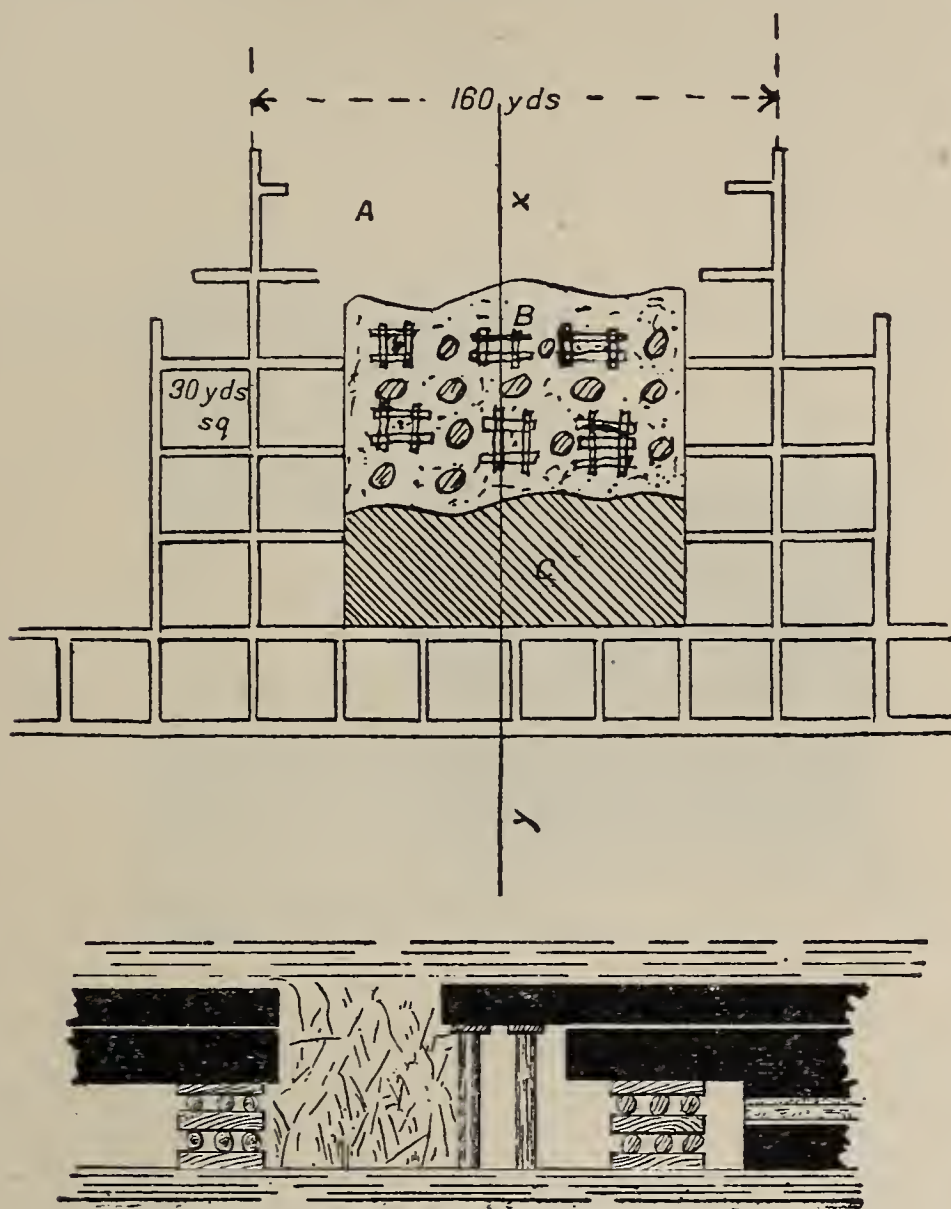
Sand walls have also been employed, but unless they can be kept in position by some means they have the same disadvantages as clay.

Where these underground fires are of frequent occurrence, it would seem best to work the seam on the ‘retreating’ system, *i.e.* driving the principal roads out to the boundary, and keeping them dipping

* Self-acting incline.

from the shaft if possible; the coal could then be worked uphill towards the shaft, and the waste allowed to fill with water up to within a convenient distance from the face.

To overcome the danger from underground fires in Fifeshire, the coal is sometimes worked on the panel system. The coal is thick, and is generally worked in two portions (figs. 190, 191, 192). Dur-



FIGS. 190 and 191.—Dysart thick seam working.

ing the first working extraction extends to the 'cherry' coal, the levels being driven 8 ft. wide, and pillars 30 yds. square left on the 'rise' side. The cross 'throughers' are also driven 8 ft. wide to allow double roads to be laid, as the latter are worked as self-acting inclines. The panels are generally formed 140 to 160 yds. square; and in forming the panel a main heading is driven 8 ft. wide, and pillars formed on each side 30 yds. square. The heading is driven up 80 yds., thus forming two pillars and part of a third one. The regular working is now begun. The stalls are driven 15 ft. wide, and small coal pillars, 6 to 8 ft. square, left to support the roof.

No dross is filled in this working. Wood pillars are put in, generally three in each place, this being sufficient to allow the roof to subside gradually. This method is continued until the 30-yd. level above is reached; it is then cut off and the coal brought out on the upper road. While working thus, the abandoned workings left behind will subside from 5 ft. at the commencement to a height of 2 ft. 6 in. A margin of 10 ft. is left next the upper level for protection, and the timber is taken out and the dross filled, the small 8-ft. pillars being now crushed to dust. The coal is then worked back, and every place of ingress built up, to prevent air travelling through the abandoned workings. When the boundary has been reached, the large pillars forming the barriers are worked back.

The advantages claimed for this system are that practically all the coal is extracted, and in the event of any fire occurring it can

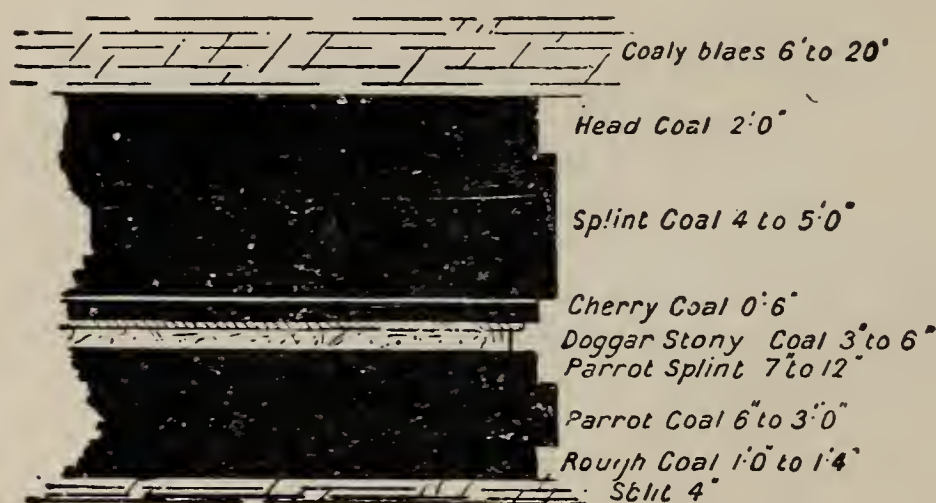


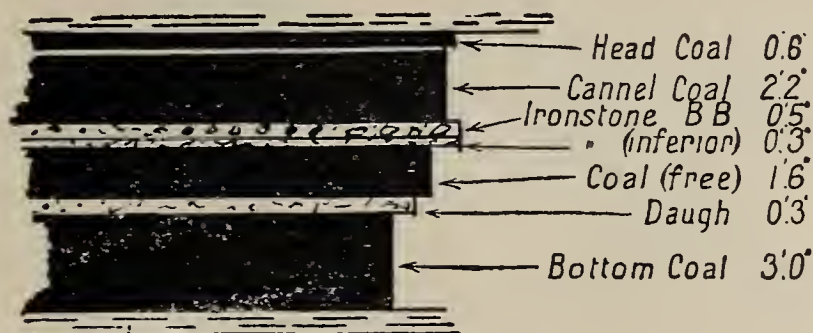
FIG. 192.—Section of Dysart thick seam.

be easily shut off by driving a level higher up and opening out anew. By this method of working, however, a large proportion of the coal, and particularly of the pillars, is so badly crushed as to be of relatively little commercial value. The seams at most of the collieries in Fife are now worked on the two-fold system. If the seam is worked by longwall, the best preventives are to have the waste as thoroughly stowed as possible, and to leave behind the minimum of small coal.

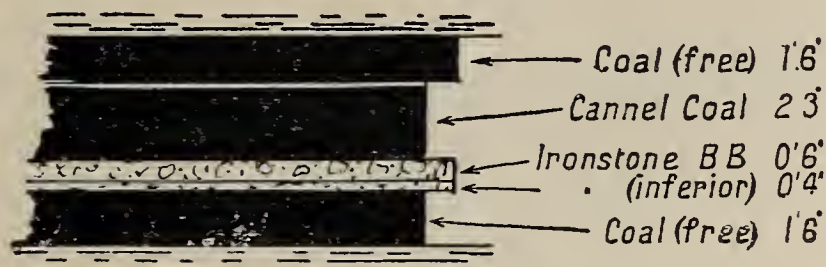
Working Highly Inclined Seams.—In working seams that are highly inclined, from 70° to 90° from the horizontal, it is found, as a general rule, that whether the seams are moderately thick or very thin, the best system of working is by longwall or some modification of it; especially is this the case when the seams reach a certain depth, say 120 to 150 fms. from the surface, when the pressure in highly inclined seams becomes very great. Until they reach such depths, they may, however, be worked perfectly well by 'bord and pillar' in the ordinary way.

At Niddrie Collieries, near Edinburgh, where the seams are inclined at an angle of 65° to 90° from the horizontal, longwall is now the only system that is practised, although formerly the seams were worked by 'stoop and room.'* Figs. 193, 194 give a section of the two seams most largely worked. The coal is won by both inclined and vertical shafts, and is worked in lifts of from 60 to 80 fms. in depth, divided into panels of about 200 fms. in length.

In each panel, and preferably as near to its centre as possible, a 'brake' incline is formed, by means of which the coal is lowered from any intermediate roads to the bottom level, along which it is conveyed to the winding incline or shaft.



Section of Great Seam



Section of Stairhead Seam

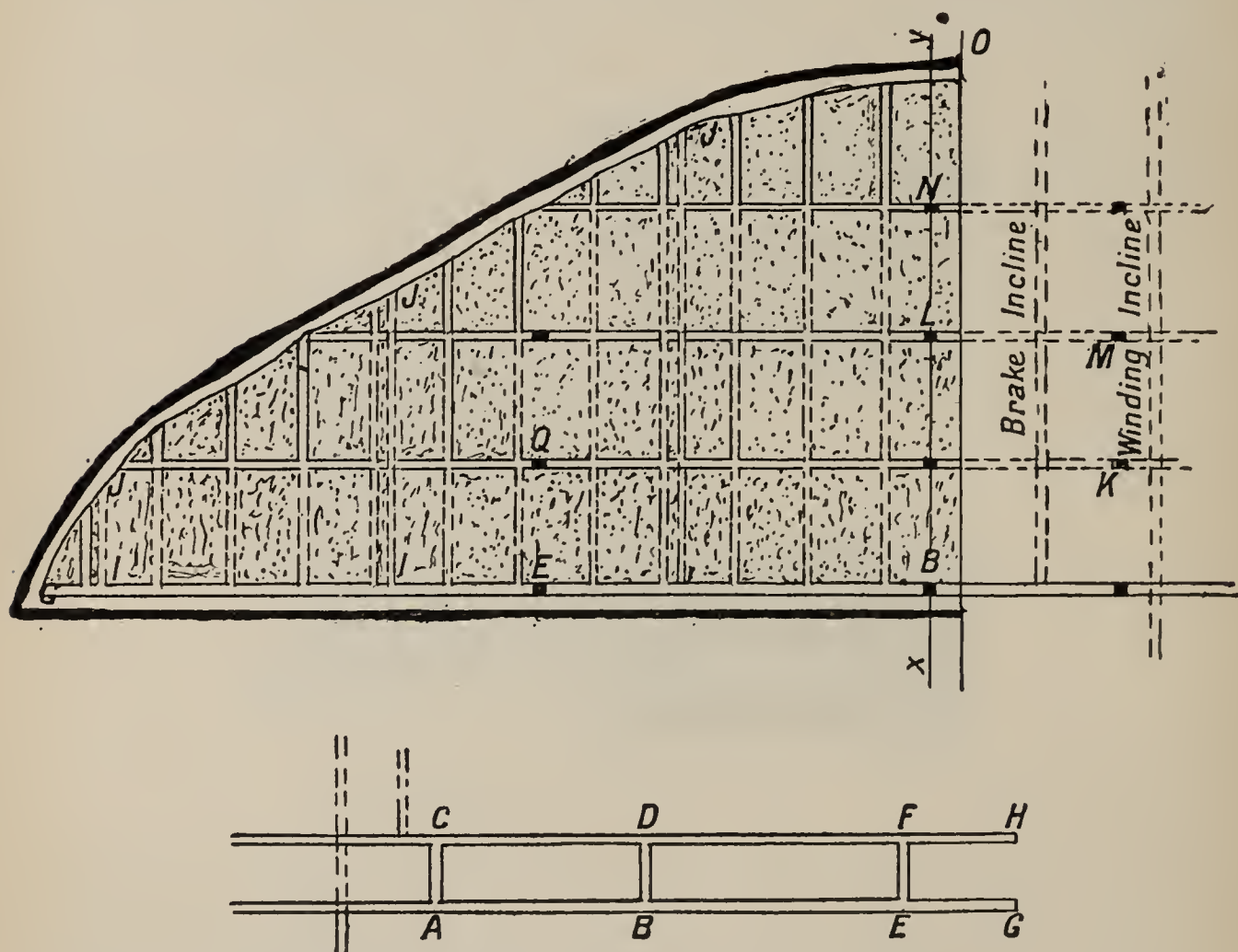
FIGS. 193 and 194.—Sections at Niddrie.

When the dip does not exceed 70° , the method of working is as follows:—Close levels (A, B, C, D, figs. 195, 196) are driven in both seams from the winding incline at the depth fixed upon as the bottom of the lift, and when a sufficient distance has been reached they are connected by a cross-cut (B D, fig. 196), and longwall operations are then commenced. A level about 8 yds. wide (B, E, G, fig. 196) is set away, leaving 6 ft. of stowage under the rails; the rise side of the place being continuously timbered with pillars 3 ft. thick and built alternately draught-board fashion, the open spaces being filled with dross. 'Spouts' or 'shoots,' 3 to 4 ft. wide, built and causewayed with pieces of ironstone, are branched off straight to the rise 12 to 16 yds. apart from centre to centre. The goaf is stowed with the daugh or fireclay, dross, rough coal, and any

* Mr Hugh Johnstone, *Trans. Min. Inst. Scot.*, vol. x. p. 204.

ironstone not required for packings. For convenience in working, the walls are so arranged that each has a long 'rise' side and a very short 'dip' side. The 'cannel' coal, which is the part for which the seam is principally worked, is dropped down the spouts (I J, fig. 195), at the bottom of which it is filled by a 'drawer.' At intervals of about 70 yds. travelling roads (L B, fig. 195) are formed to afford convenient access to the working places at different points. These are built similar to the spouts, and are furnished with ladders.

While the longwall is progressing, roads (K, L, M, N, fig. 195)



FIGS. 195 and 196.—Longwall system at Niddrie.

are driven in the Stairhead seam at intervals of 40 yds., and cross-cut mines (G N, fig. 195) are driven from thence to the Great seam, so as to strike the latter before the longwall headings reach their level. From these mines, intermediate levels (L N, fig. 195) are carried across the working faces as they come up, cutting off the 'spouts,' or what would be, in ordinary longwall, branch roads. The rails for these intermediate levels are laid upon the stowage, and the rise side of the road is timbered similarly to the levels. The close level in the Stairhead seam (D H, fig. 196) is carried in advance of the level in the Great seam, and from it cross-cuts (F E, fig. 196) are driven, connecting the two seams at intervals

of 60 to 80 fms., for the purpose of cutting off the out-bye portion of the Great seam level as soon as the 'spouts' on it have been cut off by the intermediate level above. The same system is followed

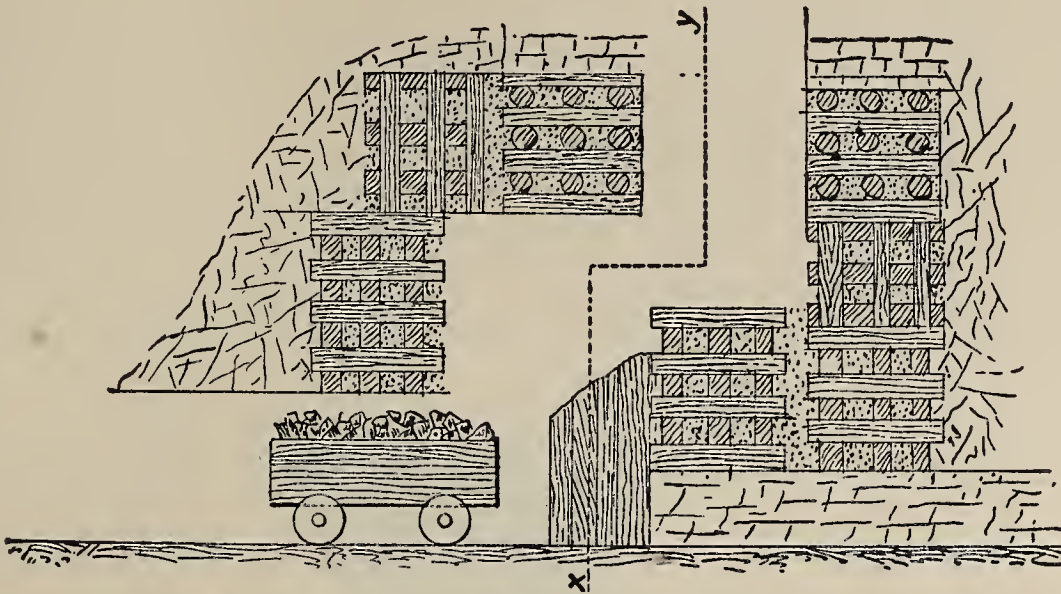


FIG. 197.—Method of supporting 'shoots.'

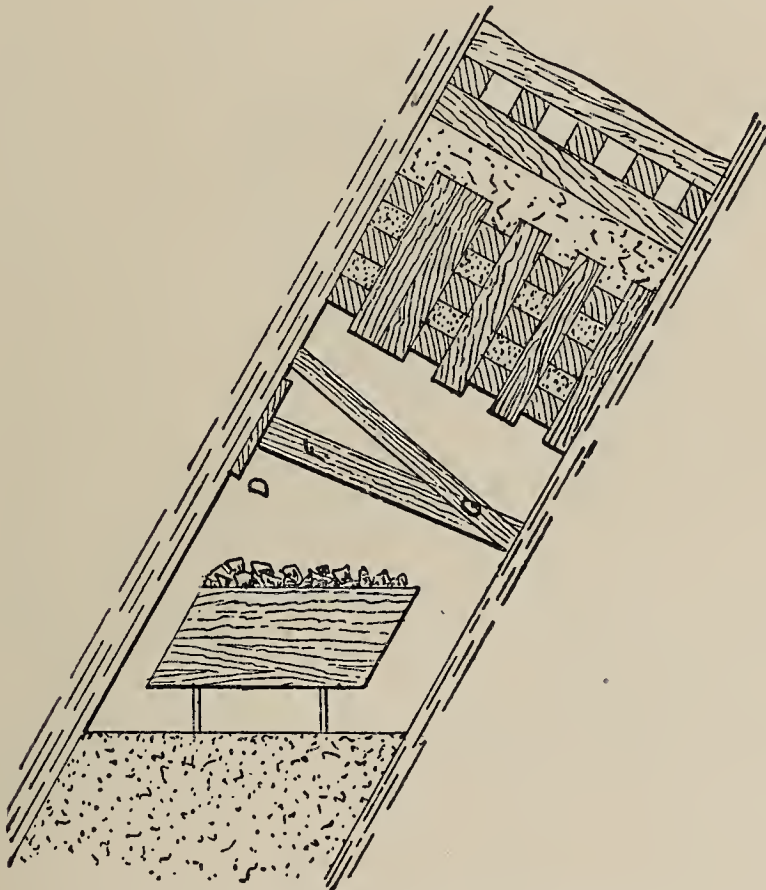


FIG. 198.—Method of supporting level.

with the upper levels, the object being to 'shorten the life' of the roads in the Great seam and to keep the horse or engine haulage as near to the working faces as possible. The method of building the levels and spouts is shown in figs. 197, 198, 199. The stowage

on the dip side of the levels tends to prevent the roof breaking and bursting out on the roads. The roof is supported by 'half-rounds' (D, fig. 198), 8 in. $\times$ 4 in., placed 4 ft. apart and supported at each end by props 5 in. diameter.

The walls are either 12 or 16 yds. long; in the first case two men usually work in each, and three men when they are 16 yds. in length. Very few props are used at the face, but the coal is kept closely 'spragged,' the distance between the 'sprags' or 'holing props' being not more than 3 or 4 ft. When the seam lies at an angle of 90° or in a vertical position, a rather different method is adopted. The plan shown in fig. 200 will explain the system.

The brake incline and haulage roads are made in the Stairhead

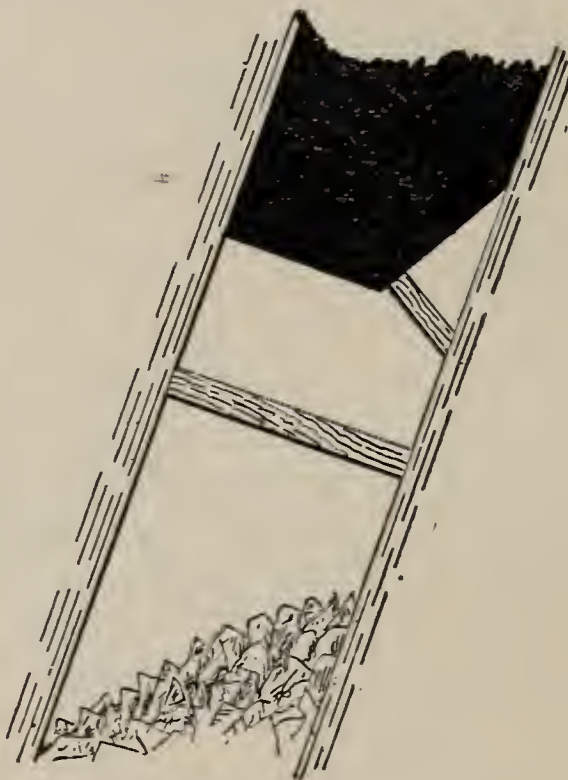


FIG. 199.—Face of working.

seam as before, and close levels are branched off this incline at distances 18 yds. apart, and from each level a cross-cut mine is driven to the Great seam.

In the bottom level (A B, fig. 200), 6 ft. of stowage is kept below the rails, and the rise side of road is well timbered. The height of the roads is $5\frac{1}{2}$ ft. clear. The 'rise' side of the place is kept trailing so as to form an angle of 45° with the road. As soon as this level has been opened up sufficiently to let the rise side reach the level of the cross-cut mine (C), roads from this mine are laid on the top of the stowage, their rise sides being timbered, and the working is again extended to the cross-cut above (D), and so on to the top of the brake incline.

The roof in this working forms one side of the road, and is supported by half-round crowns, 8 in. $\times$ 8 in. $\times$ 4 in. and 4 ft. apart,

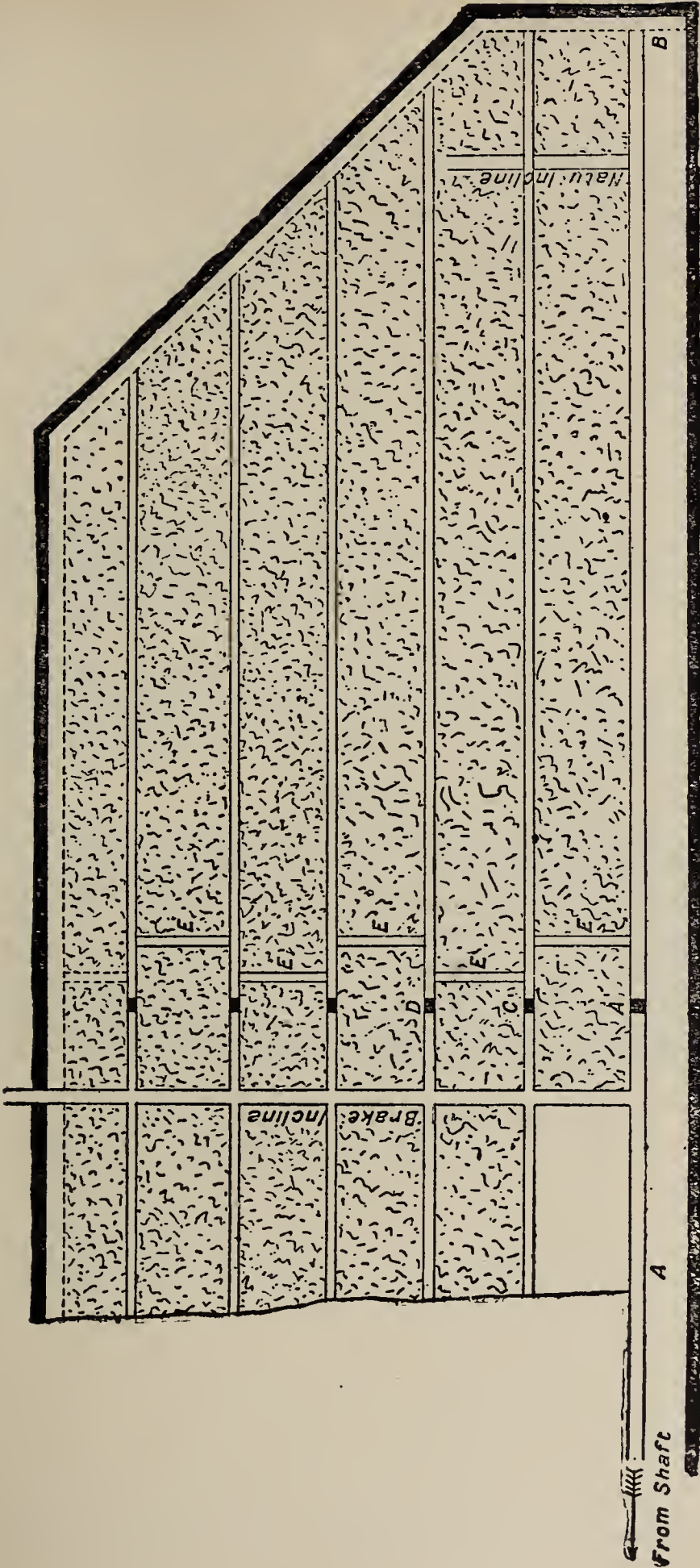


FIG. 200.—Plan of working at Niddrie when the seam is vertical.

the upper end of the crown being built into the timbering and the lower end into the stowage. The brake inclines are usually 200 fms. apart, and the coal is worked for a distance of 100 fms. on each side.

At Kilsyth, Stirlingshire, the seams sometimes lie at an angle varying from 30° to 45° , and here both stoop and room and longwall working are practised. A main incline is driven right from the surface to the dip of the seam, and the coal is worked in 'lifts' or 'benches' of about 100 fms. each. At each of these 'lifts' levels are set away on each side of the main incline (A B, fig. 201). These levels are filled on the dip side to make the rails lie as horizontally as possible. From these levels, which are usually 12 to 15 ft. wide,

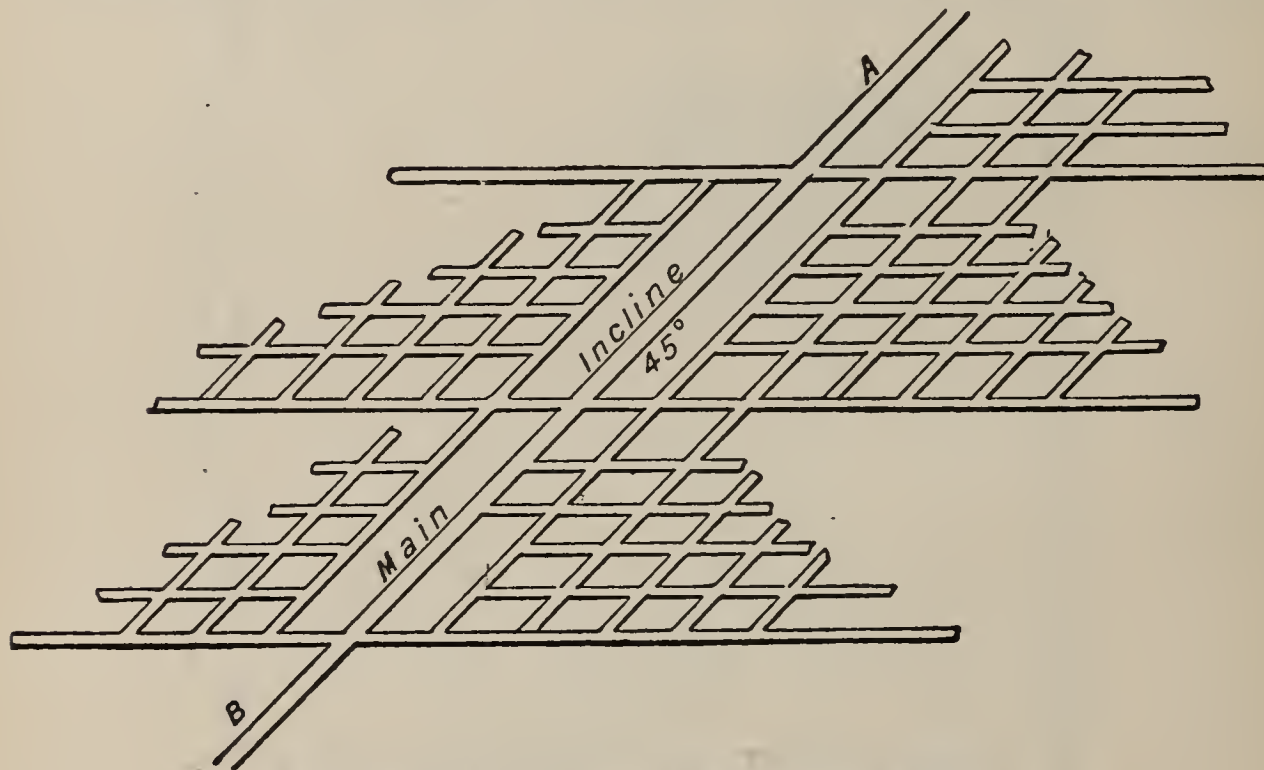


FIG. 201.—Highly inclined working at Kilsyth.

roads are set to the rise, and pillars formed about 22 yds. square, the openings being the same as in the levels. On every third or fourth roadway to the rise, the coal is brought down to the main level by 'cuddie braes,'* which work well enough up to inclines of about 1 in $1\frac{1}{2}$ or 1 in 2; when they exceed this, the coal slides down the openings, and is filled at the levels into the tubs, two planks being put across the mouth of the opening to prevent the coal from sliding out into the main levels.

As the coal is chiefly used for coking, the percentage of dross is of no consequence, pulverisation being, in any case, necessary. After a lift of 100 fms. has been turned into pillars, these are themselves extracted, by taking as many 'slices' as possible across level course, and a few lifts to the full rise. The tubs are drawn up the main incline in sets of six or seven, each tub holding about 8 cwts.

* Term used in Scotland for balance incline or jig brow.

of coal, and being filled only level with the sides. A comparatively wide gauge roadway is used, the width between the rails being 3 ft. 2 in. This is found necessary to prevent the tubs from overbalancing in the levels.

Main inclines from the surface work very well when the distance is not very great (300 fms. or so), and when the inclination is steep enough to admit of a cage being used, but where the inclination is between 30° and 45° vertical shafts are to be preferred, as offering better facilities for winding and for dealing with water.

At the Shamrock Colliery, Westphalia, where the seams dip at an

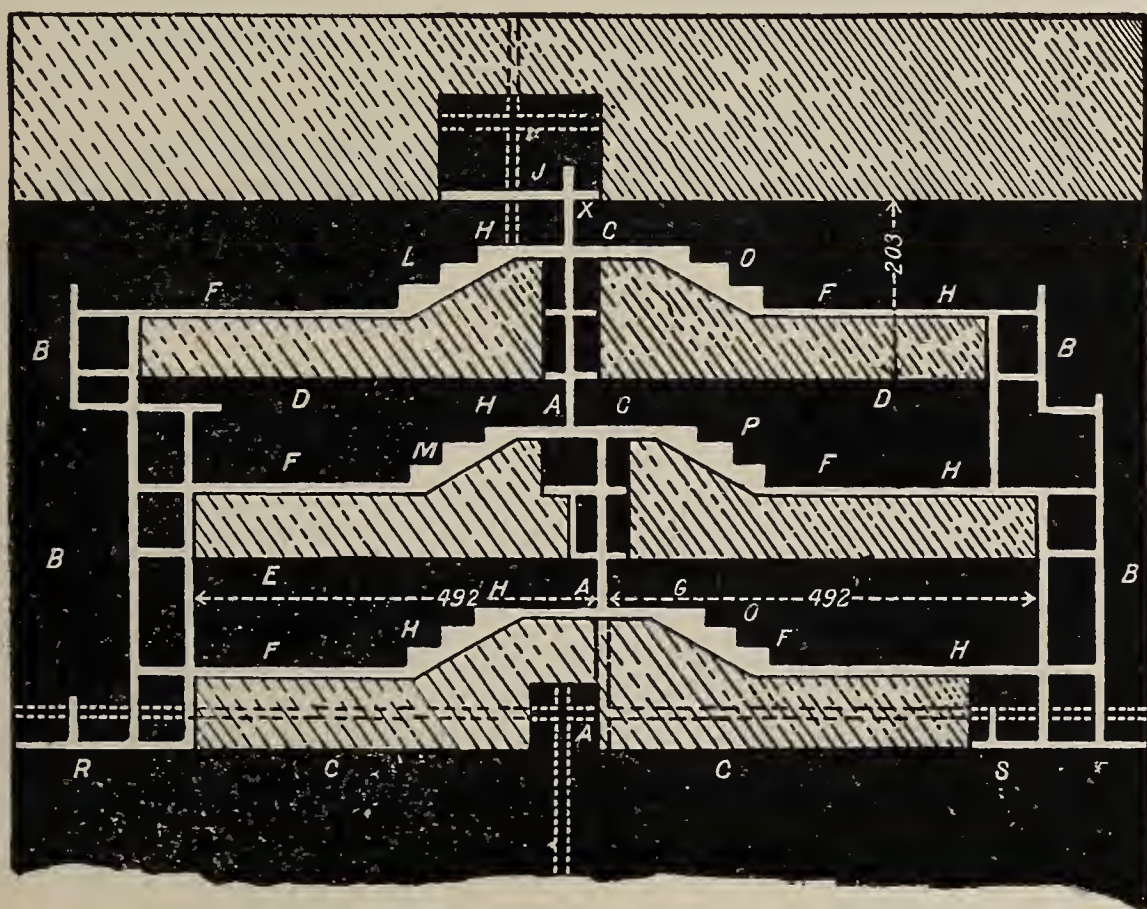


FIG. 202.—Method of working at Shamrock Colliery.

angle of 45° , a rather peculiar and complicated method of working is carried out.* The seam worked is from 7 to 8 ft. thick, and consists of a single bed of coal without any bands of dirt, the holing being done next the floor. The method of working adopted is illustrated in fig. 202. It is carried out in stages or panels of 200 yds. from top to bottom, each panel being served by a main or bottom level. The ground between two main levels is divided into great blocks or pillars 330 yds. wide, by headings B B driven to the rise, which serve as self-acting inclined planes for letting the coal down to the bottom level. These large blocks are subdivided horizontally into three parts by two intermediate levels, and into two parts on the

* *Lectures on Mining*, subj. 5, pp. 32–35, by Prof. Wm. Galloway.

line of dip by the central heading A, which is used for sending rubbish down into the workings.

At the point where the central heading A intersects the higher level, a small shaft X, 20 ft. deep, is sunk to serve as storage room for this rubbish, which is used for filling up the workings. The bottom of this shaft is connected to the central heading A by a short, level, cross-measure drift. All the rubbish which results from the driving of cross-measure drifts, and from the enlargement of roadways, etc., as well as that brought from the surface, is tipped into this shaft, from which the men who attend to the stowing of the workings derive their supplies, which are subsequently sent down the central incline A into the workings of the three sub-stages. The three sub-stages provide six level-course working places, three at the points L, M, N, and three at the corresponding points O, P, Q, on the opposite side of the heading A. While the three working places on one side of the central incline are producing coal, the three on the other side are being stowed, and *vice versâ*. When the work of stowing the three places on one side has been completed, the stowers and miners exchange places. Protection pillars having been left on the sides of the central incline, these are now taken out and replaced by stowing after the sub-stages have been exhausted. The lower end of the central heading is always stowed up carefully at the commencement, in order that the air which enters by the lower cross-measure drift may be compelled to pass to the right and left of the sub-stage workings. It then finds its way up the coal inclines, along the haulage roads to the working places, along the faces to the central incline, and through the latter into the next higher level.

In the accompanying illustration, A A are central inclined planes, in which the rubbish is let down from the higher level; B B are inclined planes for letting down coal into the lower level; C C, bottom level of the stage which is being worked, stowed with rubbish; D D and E E, levels of the sub-stages, also stowed up; and F F, roads along which the coal is hauled from the working places to the inclines.

G G are roads along which the stowing is hauled from the central incline to the working places. After these have been driven out to the limit of the sub-stage, these roads become hauling roads for coal from the working places next above. J J are protection pillars, and H H doors provided with regulators for distributing the ventilating current to each set of working places. L, M, and N are working places which, after producing coal for a certain time, are eventually stowed up, and O, P, Q are places which are in process of being stowed with rubbish, and are afterwards occupied as working places for coal.

R S are short cross-measure drifts to a lower seam, 16 in. thick, in which a level is driven under the workings of the lowest sub-stage to form a communication with the main cross-measure drift for haulage and ventilation. The cost of working by this method, with complete stowing, is returned at 2/0·7d. per ton got, including the

cost of landing the coal at the bottom level. It was found from experience that the additional proportion of large coal obtained by this system of working compensated to a great extent for the additional cost of complete stowing.

At the Tesla Mines, Alameda County, California, seams of coal are worked which have an inclination of 60° .* In one of these seams they have adopted a system called the Angle Method of Working. This system is illustrated in fig. 203.

In this system the gangway chutes are driven at right angles to the strike of the seam 40 ft. up the pitch; a cross-cut 6 ft. by 5 ft. is then driven parallel with the gangway. From this cross-cut, chutes are driven at an angle of 35° , at the same distance apart, as

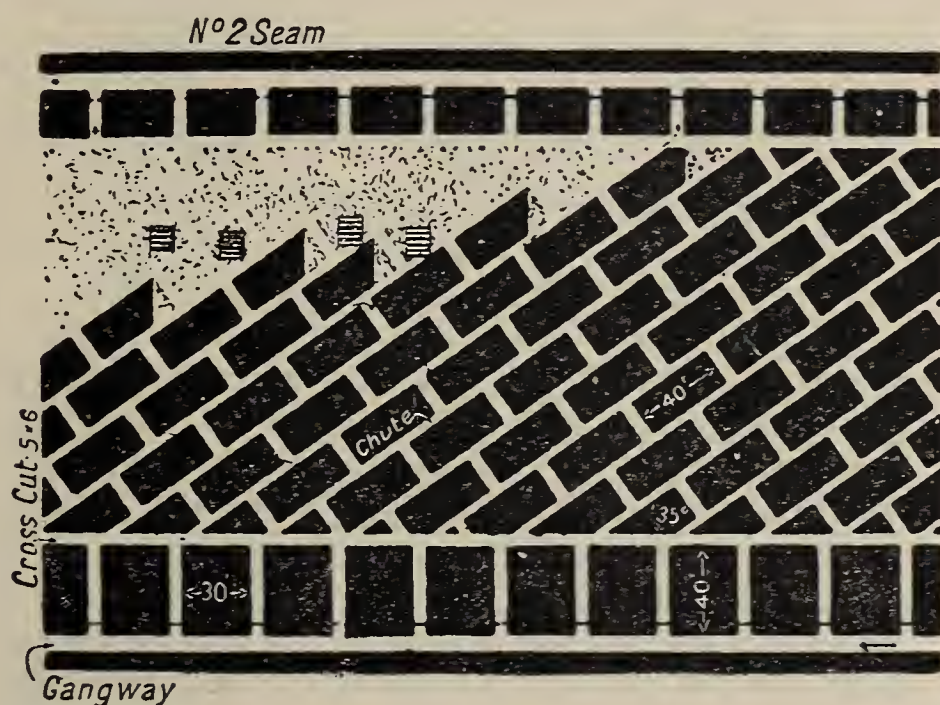


FIG. 203.—Angle system of working at Tesla Mines, California.

the gangway chutes (30 ft.); cross-cuts for ventilation being driven every 40 ft. between the chutes. After a panel of five or more chutes has been driven for the required distance, work is commenced on the upper outside pillar, the pillars on that line are drawn and the next line of pillars attacked, and this is continued until the panel or block is worked down to the cross-cut over the gangway. About every 80 ft. in this level it is found advantageous to pack a row of cogs, parallel with the strike of the seam, as the pillars are taken out. This serves to prevent the crushing of the pillars, and prevents any accidents from falls of rock.

In the regular working places little timber is required, as the roof is very good. It is claimed for this system of working that the coal undergoes less breakage than in the method where vertical chutes are used, while in the matter of timber a considerable saving can be effected.

* *Mines and Minerals*, Oct. 1898.

In the No. 7 or Summit seam a rather different method has been adopted. This seam averages 7 ft. of clear coal without foreign matter of any kind. The roof is good, being composed of a shelly slate, while the floor is a hard, compact sandstone, and it is in this that the chutes, airways, inclines, and other openings necessitated by this method of working and the characteristics peculiar to the seam are driven. The method of working this seam is shown in fig. 204. Two double chutes are driven up the pitch, or to the full rise, at a distance of 36 ft. apart, connected every 40 ft. by cross-cuts, one side of each chute being used for coal and the other side as a manway and air course.

When the seams are thin and highly inclined, a method of working termed '*Gradins Renversés*' or 'inverted steps' is often adopted in

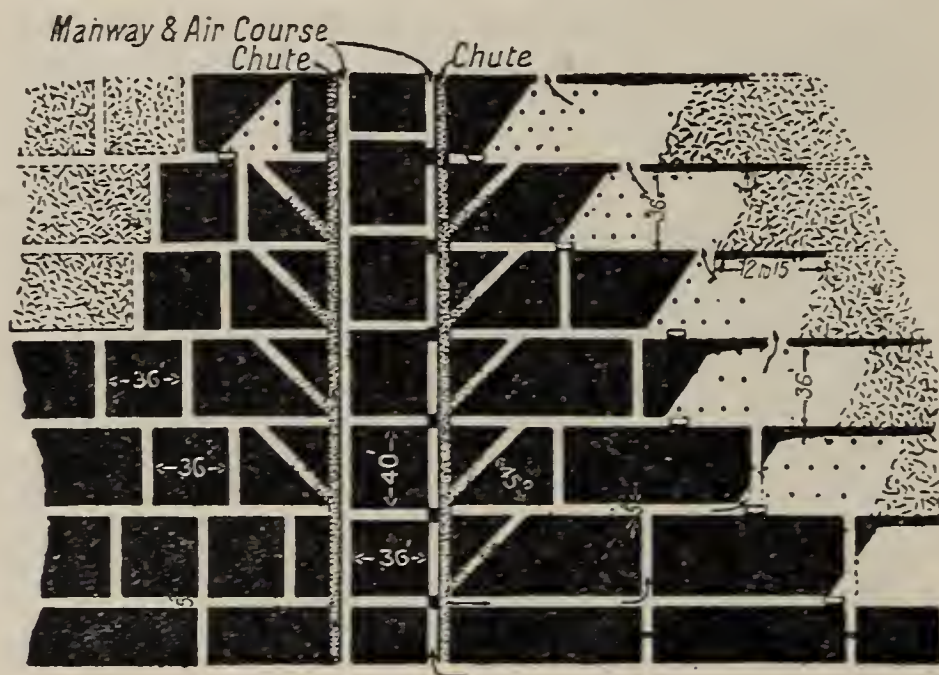


FIG. 204.—Method of working the Summit seam at Tesla Mines, California.

Belgium and other European countries. The seam is worked in the longwall method, with the face advancing in the line of strike. The system is much like that adopted at the Niddrie Collieries, which has already been fully described. The working face advances in the direction indicated by the arrow (fig. 205). The miner, standing on a planking beneath him or perched on the props stanchioned between the roof and floor, operates on the vertical face of the coal in his stall, while solid coal overhangs him. D is the drawing road leading to the main haulage road, and receives the coal from the stalls through the chimney C C, left in the gob or waste. Each chimney is provided at the bottom with a hopper door, which is opened when a tub is brought under it to be filled. As far as possible it is sought to keep the chimneys full of coal, but, even with careful supervision, this is not always attained. An obvious inconvenience attaching to these chimneys is the liability of the coal to get jammed, while they always present a source of danger to the passage of men

over them. The height of a step or stall, which varies in different localities from 2 to 4 yds., is determined by local circumstances, chief among these being the nature of roof and pavement, the amount of gas constantly given off from the coal, and the liability to sudden

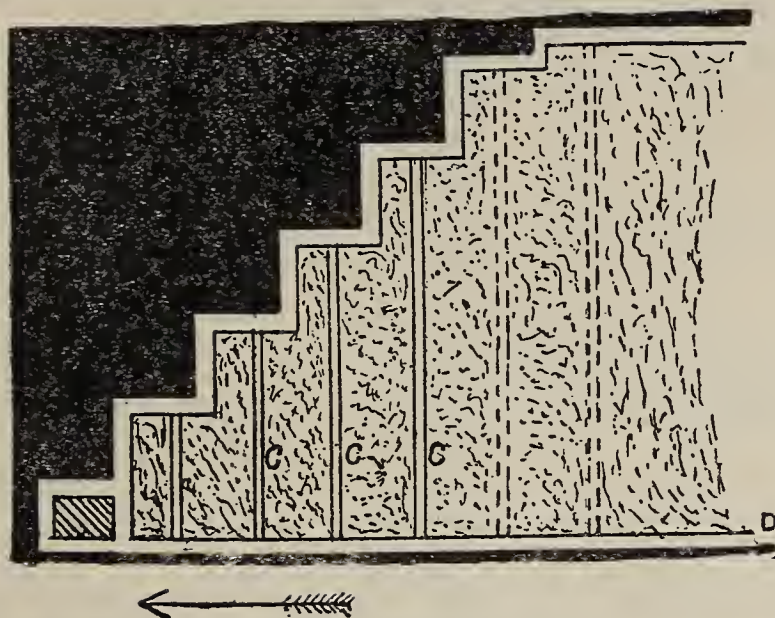


FIG. 205.—*Gradins Renversés*.

outbursts of gas. Other things equal, the height of each step regulates the rate of daily advance.

When the height given to a stage is considerable, one or more intermediate or false levels (A A, fig. 206), communicating with the

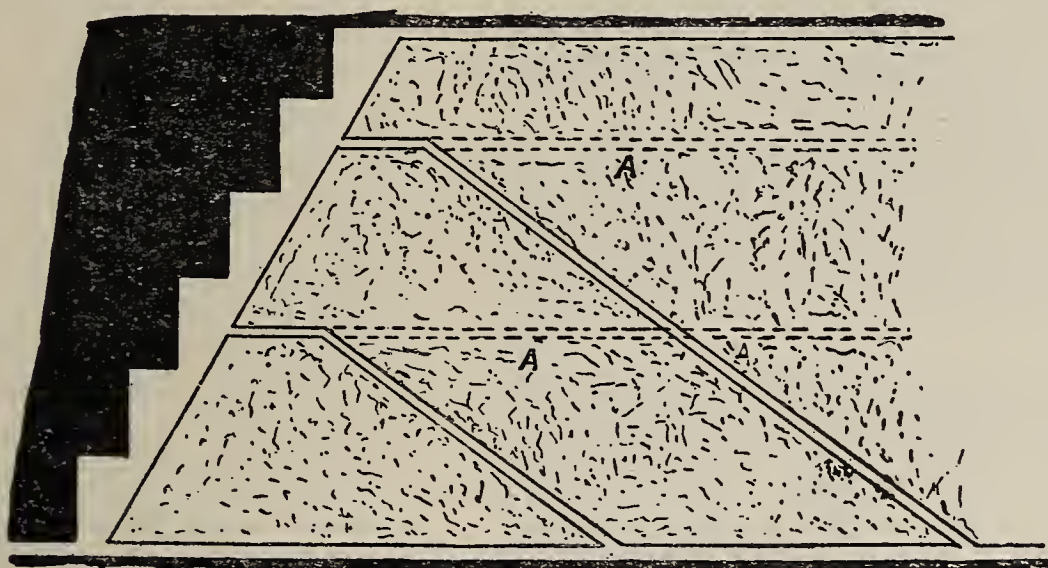


FIG. 206.—Working by 'inverted steps.'

lower level by means of inclined planes through the gob, serves to divide the distance and to limit the length of the chimneys.

Double and Single Stall Methods.—In South Wales a method of working is adopted which is peculiar to that district, although, in some of the other coal-fields of Britain, modifications of this system of working are sometimes met with.

This is the single stall system, in which, instead of a pair of narrow levels being driven as in ordinary pillar and stall working, a stall about 8 yds. wide is carried forward.

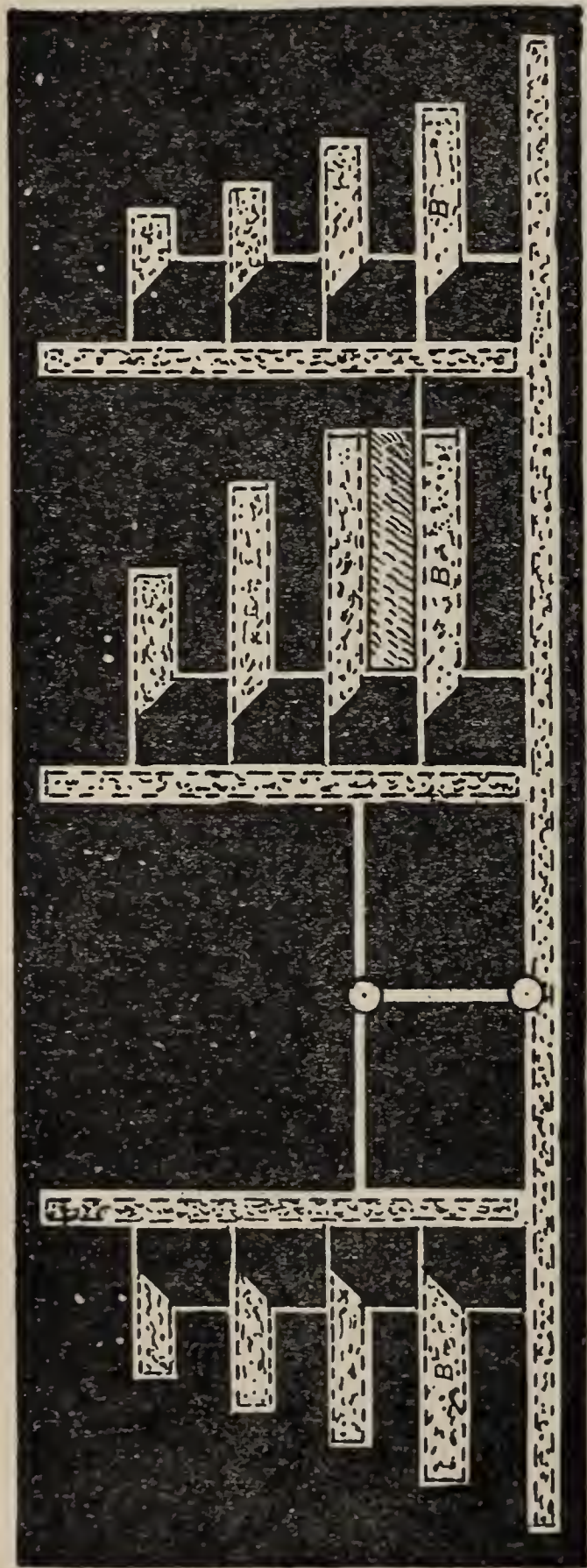


FIG. 207.—Single stall system.

Any rubbish that is found in the seam is tightly stowed in the place as it advances, a drawing road being formed on the down-side next the coal, while on the up-side an air-way is formed. When the main levels have been driven out a considerable distance to form a shaft pillar, headings of the same width as the levels (8 or 9 yds.) are set away to the full rise, with the drawing road and air-way formed in the same way. From these headings, stalls, parallel to the main levels or level-course, are branched off every 20 to 24 yds. between centres. These stalls are driven 6 or 7 ft. wide for a few yards, when they are opened out to 10 or 12 yds., according to the amount of rubbish available and the depth from the surface. The stalls are worked in exactly the same manner as the levels and headings, the roadway being formed on the down-side of the road and the air-way on the rise, the space between firmly packed.

Headings are driven off the levels at distances of 100 yds., forming a sort of panel, and a narrow drift is formed in each of these panels between the main level and the lower stall (B, fig. 207) to admit of air passing from the main level into the stall workings.

The stalls of the heading are driven out for a distance of 75 yds., and then they are driven narrow for a distance of 15 yds. or so, until the next heading is cut into, thus forming a connection between the different headings for ventilation. When the second heading has been put up and stalls formed off, parallel, and opposite to those on the first heading, the stalls first driven are then worked (to the first heading), leaving a pillar 15 yds. thick to protect the headings. The pillar between the stalls is thus brought back to the required distance. The same process is carried on throughout, headings being turned off the main levels at regular intervals, and a block of coal is left alongside the main road until the latter has reached its destination, when it may be worked back towards the shaft.

The double stall system is similar to the single stall system, with the exception of the stalls being driven very much wider, and two roadways being formed instead of one roadway and an air-way. Main levels are driven out from each side of the shaft as before (fig. 208),

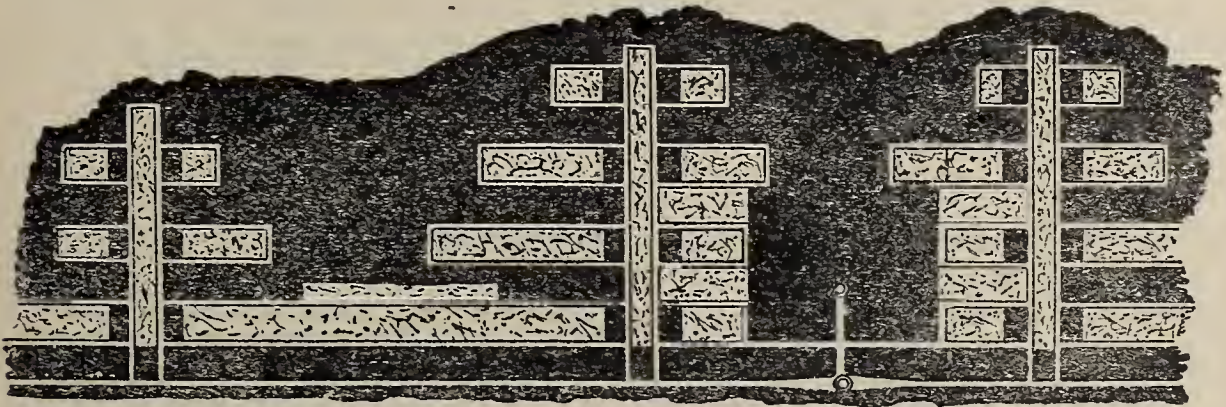


FIG. 208.—Double stall system.

but in double-stall working these levels are often driven narrow, 9 ft. or so, and headings are put away to the full rise, 8 yds. wide, a block or panel of coal of 90 yds. being left between the headings. From these headings roads are turned, parallel to the main level, but instead of these being single roads, as in the single stall system, two roads are set away with 18 yds. of coal between them, and after being driven a short distance they are opened out and connected, thus forming a place 22 to 24 yds. wide. The roads are formed one on each side next the solid coal, and the space packed between them, and in this way they are carried forward until within 15 yds. of the next heading, when one of the roads is continued through to the heading and forms an air-way. The pillar left between two stalls is then worked back, each road taking a slice equal to half a pillar or about 6 yds. This back-lift is brought to within 30 ft. or so of the heading, being left for protection until the latter is finished, when it may be extracted.

These systems of working are best suited for seams having a large amount of rubbish with which to stow the space between the roads properly, because if the separating pack is not well built the ventila-

tion will suffer. A large percentage of the coal in both systems is either lost or rendered useless through the small pillars or stumps left, for protecting the headings and levels, becoming crushed to dust, and it is a question if longwall would not be in any case better suited in districts where single or double stall is now worked.

Working Thick Seams.—In Britain, and also in other European countries, seams of great thickness are found, which are often difficult to work.

At Quarrelton, near Johnstone, Scotland, a very thick seam of coal is found, 60 to 90 ft. thick, in a trough or basin; but here the seam, being found near the surface, was worked chiefly on the open-cast principle. Shafts were also sunk to win some of the deeper and more thickly covered parts, but the coal being of poor quality and ready to take fire, little has been worked.

In South Staffordshire the valuable 'Dudley Thick' or 'Ten Yard Seam' is found, of a thickness varying from 25 to 37 ft. (fig. 212).

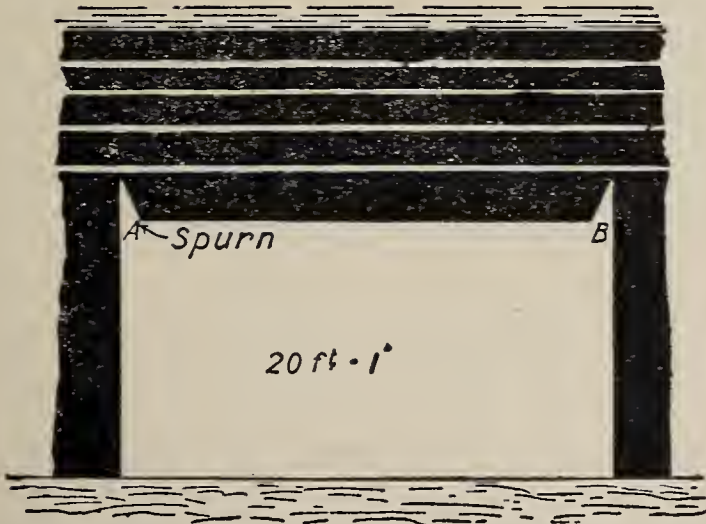
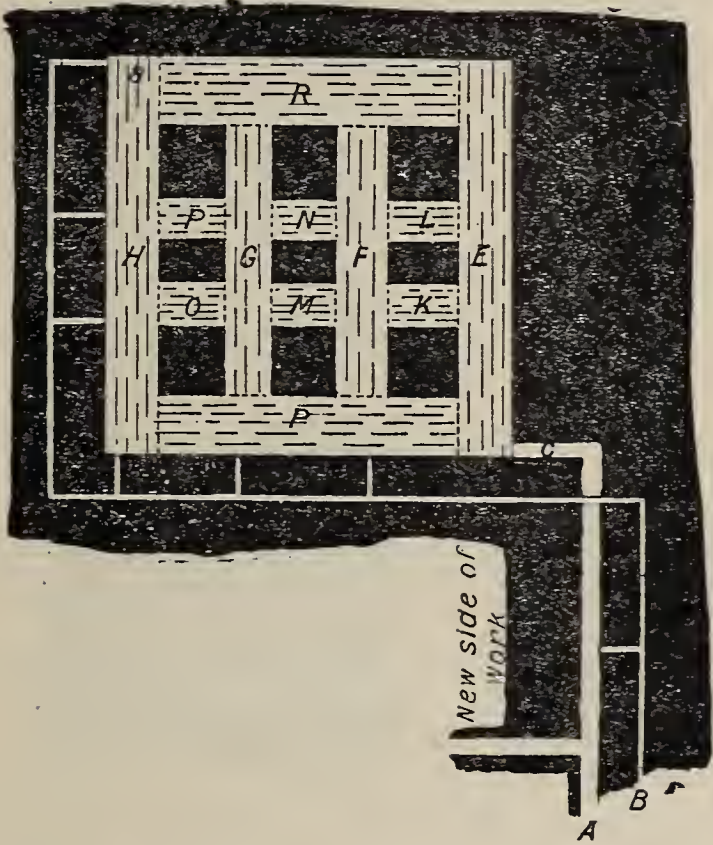
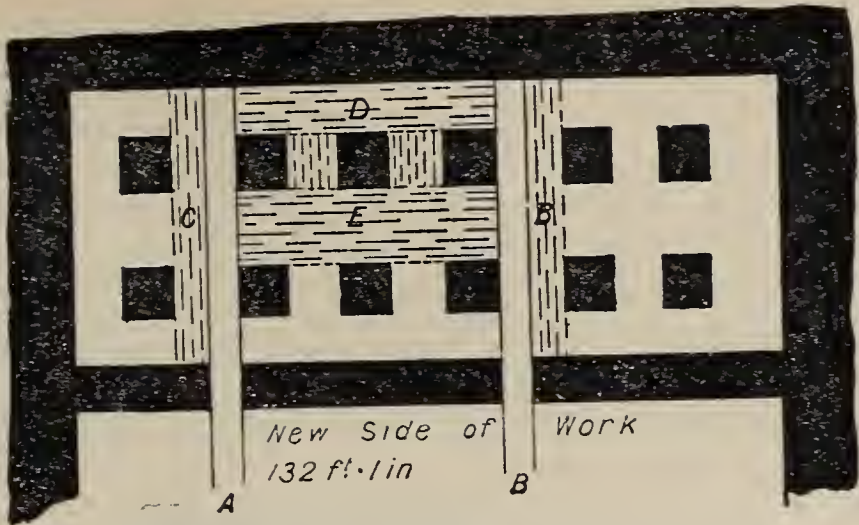
This seam has been worked at various times in three different ways, viz.:—(1) Square work; (2) longwall, the face being extracted in two portions; (3) longwall, the whole of the face being removed at one operation.

The first method is the one that is most largely and successfully followed, the other two methods being almost, if not quite, abandoned.

In working this thick seam, no matter what system is adopted, it is an invariable rule to first drive out the roads to the boundary of the royalty and then work back towards the shaft.

Square Work.—In this method of working after the gate-road and air-head (A and B, fig. 209) have reached the boundary of the royalty, the first operation is to drive a narrow head called a bolt-hole (C, fig. 210). When this has reached a distance of between 10 and 15 yds., its width is increased to 6 or even 9 yds., when it practically becomes a stall (D, fig. 209). In driving these stalls, the next step taken is to 'hole' or 'kirve' the benches for a distance of about 9 yds., the roof being supported by small cogs of slack and timber or by short sprags. The 'slipper' and 'sawyer' parts of the seam will then be vertically cut on both sides of the stall, 'spurns,' or intervals of solid coal, being left every 6 ft., as in the case of gate-roads. When these 'spurns' are removed, it frequently happens that the 'patchells' and 'stone' coal fall with the 'slipper' and 'sawyer'; if this occurs, the sides are dressed straight and timbering put in to support the roof. Further lengths of 9 yds. are similarly attacked in the stall (C, fig. 209), until the back rib is reached. While this has been going on, other stalls (E, F, G, fig. 210) have been driven in the directions shown by the dotted lines, and finally the pillars will be 'thirled' by means of the short lengths shown (K, L, M, N, O, fig. 210). After the driving of the stall R, the side of work is fully opened, and cutting in of the top coal begins.

This proceeds in the same manner, each layer or bed of coal being



FIGS. 209, 210, and 211.—Working thick seam in South Staffordshire.

removed in ascending order. If the roof is very good a long range can be taken, but it is far better to carry out this part of the work in short ones; distances of 20 yds. being taken and a timber cog built up to the roof. A vertical groove will then be cut on each side of the stall and on the face of the cog, treating the latter as if it were solid coal, spurns being left as before; their appearance when the 'brazils' and 'heath' coal are being cut is shown in fig. 211. When these 'spurns' are taken out (as shown at B, fig. 211) and the timber removed, the whole mass within the portion cut falls down and is ready for loading up. Similarly a layer of top coal is removed

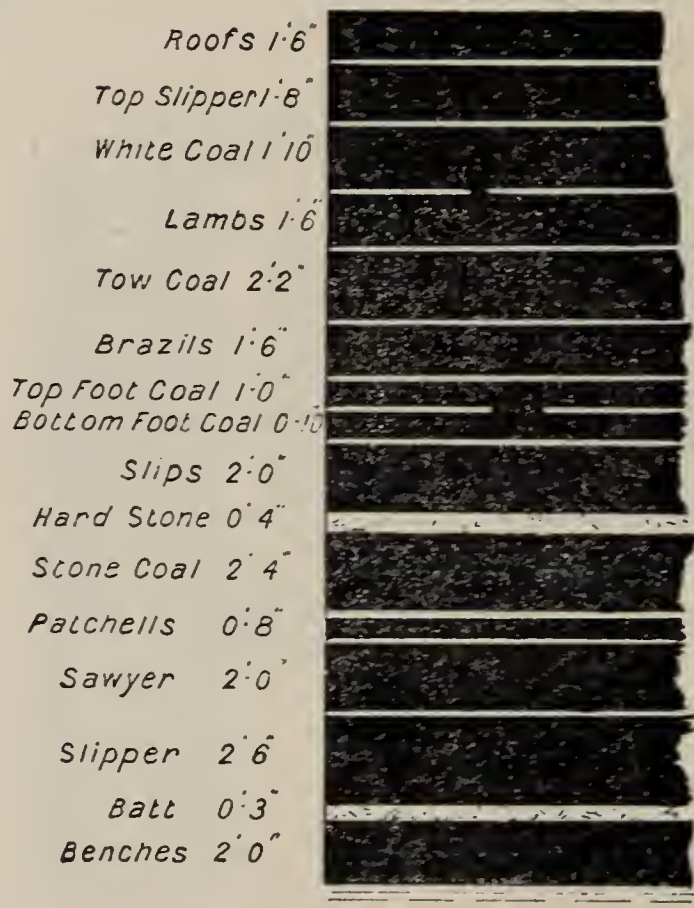


FIG. 212.—Section of 'ten yard seam,' Dudley.

all over the side of the work, and when this is done the 'white' coal and then the top 'slipper' coal will be attacked, and the same cycle of operations gone through, until these seams have been cut in. No attempt is made to get the 'roofs'; sometimes, however, they break down without the superincumbent strata following, in which case they are removed as speedily as possible. In cutting the top coal, the workmen stand on a platform, constructed by cutting two holes in the solid coal and inserting short timbers, on which a plank is placed. A frequent occurrence is to build timber cogs up to the roof when the bottom coals have been removed; these cogs being placed in the openings between the solid pillars of coal, and when this is done the latter are reduced in a marked degree.

The other two methods of working have not given such good

results as 'square' work, owing to the large amount of dross produced, while a smaller yield per acre was likewise obtained by both the alternative longwall systems.

Order of Working Seams.—Where a number of seams occur in a coal-field, often with little intervening strata between them, it is a difficult problem as to which should be worked first so as to reap the greatest advantage. There is a considerable difference of opinion amongst mining engineers as to the order or rotation in which seams should be worked, some being strongly of the opinion that the descending order is the best, *i.e.* working out the upper seam first, and taking out the others in their descending order. Others, again, maintain just as strongly that the ascending order is the proper one, *i.e.* sinking to the lower seam first, and afterwards working the others above in rotation, until the one nearest the surface is reached. We do not believe that any hard and fast rule can be laid down as to the order of working seams which will suit the conditions of all coal-fields or districts alike, as so much depends on the circumstances in each district, or even in each royalty, and therefore each will have to be carefully considered on its merits and the conditions under which the seams are found. The general rule in all coal-fields is to work the best seams first, irrespective of their order, and irrespective of the effect that the working of one seam may have upon the others either above or below it. The aim of a colliery owner is to get a return as quickly as possible on the capital expended in sinking and fitting shafts, and also to get a reputation in the market for his best class of coal. Looked at from a national standpoint, without considering the cost of working or the price per ton to the consumer, there would be great advantage in working seams in a specified order; but we fear that is an ideal that will not be attained.

Where seams are dislocated or disturbed by faults it is sometimes difficult to work them in any definite order. As a general rule, when the distance separating seams is above 30 or 40 yds., and the intervening strata strong, the working of one seam will have little effect on the other, above or below, and it is immaterial which is worked first. This is especially the case if longwall is the system employed, and the waste well packed. Where there is less than 30 yds. between the seams, the working of the one will affect the other. For instance, in the Lanarkshire coal-field the Ell, Main, and Splint seams are each separated by 20 to 25 yds. of strata, and the working of any of these seams always prejudicially affects the others, and generally makes them more difficult to work.

If the Ell seam is worked first, then the Main seam below becomes 'woody' or 'corked,' as the miners term it, by the weight being taken off, with the result that it is more difficult to get, and requires an extra cost in the hewing rate of 10 to 15 per cent.

On the other hand, if the Main seam is worked first, then the

roadways are much more difficult to keep and expensive to secure in the Ell seam.

Mr John Gemmell,* a well-known mining engineer, states that in Lanarkshire he had known the Main coal, a soft seam, to be hardened by the prior working of the Pyotshaw seam, 8 yds. above, and also by the simultaneous working of the Splint seam, 24 yds. below. In the latter case the cost of hewing the Main coal was increased by 7d. per ton, but this was counterbalanced by a diminution in the quantity of small coal from 50 per cent. of the seam to 25 per cent., which gave an increase in value of $7\frac{1}{2}$ d. per ton.

Sometimes the working of a lower seam is a decided advantage to the working of an upper seam, but, as stated, it depends on the distance separating the seams.

Loss or Waste in Working.—In working a mineral bed or vein there is invariably a certain percentage of loss or waste. In the working of coal seams this loss or waste is due to a variety of causes, such as: (1) pillars to be left for supporting shafts; (2) pillars requiring to be left for support of surface buildings, beds of rivers, canals, railways, roads; (3) areas left as barriers between properties on different leaseholds or royalties; (4) pillars left for protection of main underground roadways, air-ways, water-courses, etc.; (5) losses due to faults and disturbed ground; (6) losses due to inferior coal, burnt coal, bands of dirt in seams; (7) waste due to small coal left in the workings; (8) ordinary losses in the regular extraction of the seam.

From these various causes the loss varies very much for different seams, different systems of working, and in different districts of the country.

The total loss or waste from all causes varies from 10 to 40 per cent., and in exceptional cases as much as 50 per cent. For the losses under each of the causes, the following rough approximation may be taken: (1) minimum 1 per cent., maximum 2 per cent.; (2) minimum 1 per cent., maximum 4 or 5 per cent., but where royalties are much intersected by rivers, canals, railways, etc., the loss may be as much as 10 to 12 per cent.; (3) minimum $1\frac{1}{2}$ or 2 per cent., maximum 5 or 6 per cent.; (4) 1 to 2 per cent.; (5) 5 to 10 per cent.; (6) 5 to 10 per cent., but very variable, sometimes reaching 20 to 30 per cent.; (7) very variable in different districts, 5 to 20 per cent. according to method of working and method of filling the coal; (8) depends greatly on nature of seam and system of working, and whether coal is undercut by machines or hand labour; may vary from 2 to 3 per cent. for longwall, and 4 to 6 per cent. for pillar and stall.

* Royal Commission on Coal Supplies, Digest of Evidence, p. 183.

CHAPTER IX.

TIMBERING ROADWAYS, Etc.

Necessity for Timbering.—When roadways are opened out underground, they must be efficiently supported in order to keep them secure and safe for traffic. Upon the artificial supports which are generally used depend the lives of those employed, and the regularity of the coal output. The proper timbering of underground workings is therefore of the utmost importance. When it is remembered that about 50 per cent. of the total number of accidents, fatal and non-fatal, which occur underground are due to falls of roof and sides in the roadways and at the coal-face, the necessity of paying considerable attention to proper spragging and propping becomes even more manifest.

A bad roof is often defined as one which requires the application of systematic support, while a good one is supposed to require no such methodical treatment. The principle which should, however, be observed is “that timbering is not for a bad roof only: it is intended to prevent a so-called good roof from becoming bad.”

A roof may be bad in itself, owing to the slight cohesion of its particles, as in a fireclay roof. It is then called ‘tender.’ What may be a good, strong roof in itself may, on the other hand, be so full of ‘lipes’ and ‘faults’ as to form a very treacherous one. As already shown (Chap. VIII., p. 211), the difficulty or otherwise of maintaining a roof will depend a great deal on the direction in which the roadways are driven, *i.e.* at right angles to the cleavage in the roof or parallel with it.

Mr A. R. Sawyer* says:—“It is fallacious to suppose that what are called bad roofs need be productive of more accidents than good roofs. Roofs containing numerous dislocations, lying much in the same direction, necessitate a specially systematic kind of timbering, which will ensure comparative safety; while a good roof, in which dislocations occur rarely, but unexpectedly, may be fraught with danger from the want of a rigid method of timbering, irrespective of whether, in individual places, posts are absolutely required or not.”

The timbering of the working is done sometimes by the miners

* *Accidents in Mines*, 1886, by Mr A. R. Sawyer, p. 28.

themselves, and sometimes by special workmen. In Northumberland and Durham the whole of the timbering at the faces or walls is done by 'deputies,' these men devoting their whole time to the work, while the colliers at the face have nothing to do with the propping. In other districts, as in Lancashire, Wales, and Scotland, the whole of the timbering at the working faces, and often for some distance back in the roadways, is performed by the colliers themselves.

Much difference of opinion exists as to which is the better and safer method. The writer is of opinion that the system of making the miner responsible for the timbering is the one most likely to secure attention to the safety of the roof and sides. The miner, being continually employed in his own stall, is more likely to detect any change in the condition of the roof than a deputy who only visits the stalls at intervals. The miner, too, is always at hand to set a prop in cases of emergency, and will take an interest in keeping his place secure for his own sake. It has been urged that no man is more careless of his own safety than the miner, and perhaps there is some truth in this—even good, reliable men often falling victims to their own carelessness. While the miner ought to have the proper timbering of his working place in his own hands, more power ought to be invested in those supervising his work, such as firemen, to compel the adoption of a systematic method of propping and to put up props at stated distances apart, whether the roof appears good or not.

The Royal Commission, in their Report on Accidents in Mines, recommended attention to the following points:—

The maintenance of ample supplies of timber in localities convenient to the workmen.

The proper training of each miner to the best method of timbering and otherwise protecting his working place.

The exercise of increased care on the part of the workmen in watching the roof, sides, and faces, and protecting themselves in time.

The introduction, as far as possible, of arrangements with the workmen, which will make it their interest not to avoid the labour of putting up the necessary timber for their protection.

The employment of special timber-men or deputies for the timbering of the main-ways, and also for the repair as well as the drawing of timber.

Preventing timber being left in the goaf of longwall workings which would have the effect of breaking the roof.

Driving the working places as rapidly as possible, by shifts of an ample number of workmen at each face, and so reducing the risk of falls and exposing the least number of men to danger at any one time.

These instructions, if they could be rigidly carried out, would tend in a large measure to reduce the number of accidents to a minimum. It is, however, impossible to lay down any general rule for the timbering of mines which would be satisfactory under all circumstances, and applicable to all mining districts.

Timbering or supporting the roof may be divided into—

- (a) Timbering the working faces.
- (b) Timbering the ordinary drawing roads.
- (c) Timbering or supporting the main drawing roads.
- (d) Timbering and supporting the haulage roads.
- (e) Supporting the roadways at the pit-bottom.

The timbering or supporting of the roof may be done in a variety of ways, according to its nature and that of the floor and sides, and may be carried out as follows:—

By packing the waste entirely, where sufficient rubbish can be got, and timbering at the face and roads, as in longwall.

By partial packing of the waste or goaf, by buildings or stone pillars with intervening spaces, and timbering at the face and roads.

By timbering at the face and in the roads, and leaving wood or stone pillars in the waste as supports. This is the usual practice where the seam yields no packing material.

By timbering alone, without any packing whatever, as in bord and pillar, or in longwall on the retreating system.

Supporting the main roads with brick arching, steel or iron, or with combinations of masonry and iron, or steel work.

Timber.—For underground work the different varieties of timber used in Great Britain are larch, Scotch fir, Norwegian pine, and sometimes oak and beech.

For ordinary timbering Scotch fir is largely used, particularly where the timber does not require to be heavy, and where the pressure is not great; but in haulage and main roads that require to be kept open for some time, larch has been found to give the best results, both as regards durability and economy.

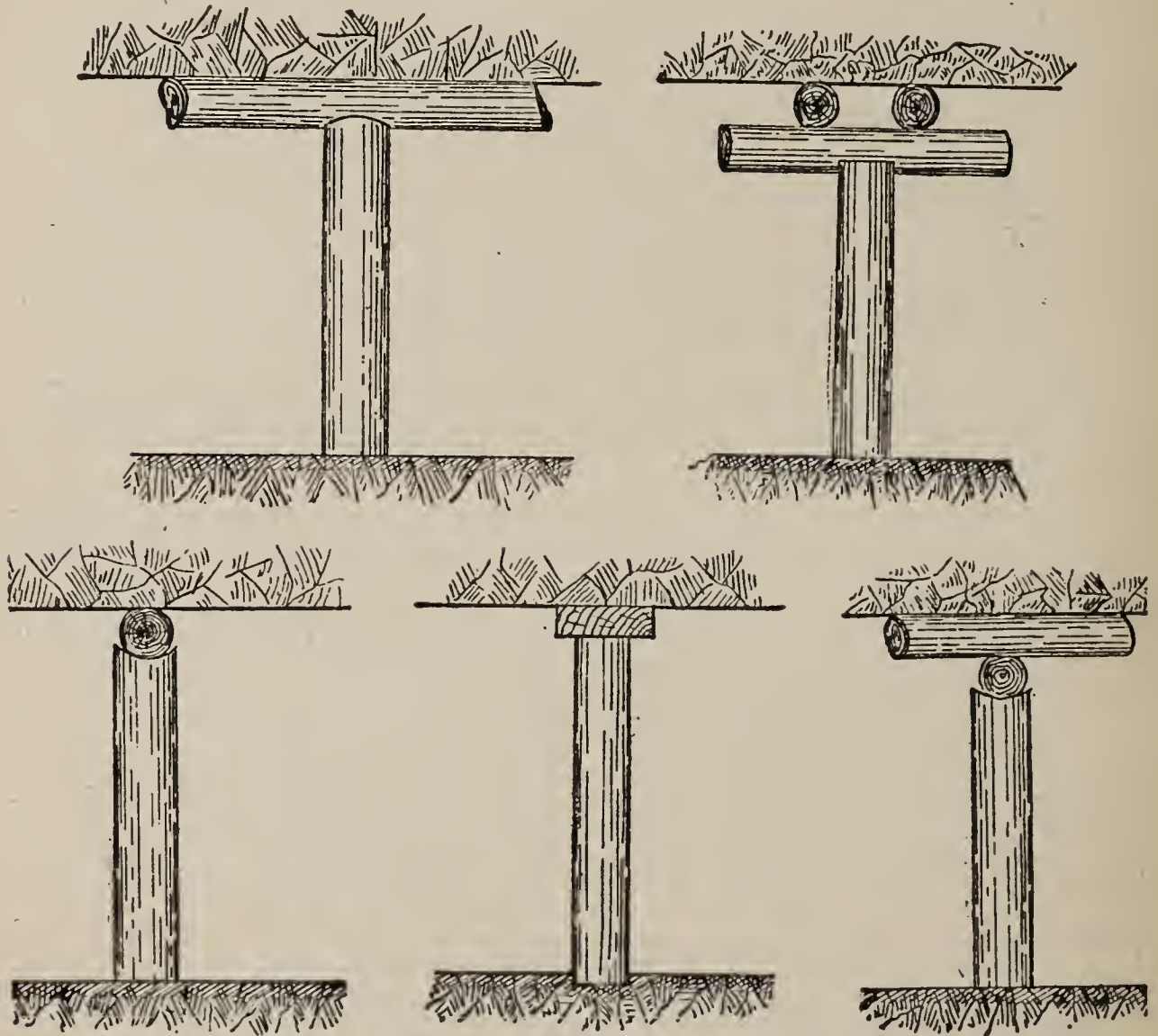
Oak is an excellent wood where great pressure is likely to be met with, but the cost of such timber is high, and it is difficult to get straight pieces of any length, while it is also a difficult wood to dress.

Beech props are short-grained, more or less brittle, and break off short under crushing stresses. Their great weight, too, as compared with other wood, and the consequent difficulty in handling them, militates against their use underground.

Any wood that is long-grained and elastic, and will yield to pressure or thrust, is suitable for mine timber, because props of such wood often serve the purpose for which they are used, even when partially fractured.

Supporting the Working Faces.—In longwall working, where material is got from the ripping or brushing of the roof, packwalls or buildings are put in to form the roads, and also at right angles to and along the line of face. The proper building in of these packwalls is of great importance in the securing of the roof in the roadways and faces. The hardest and largest pieces of the débris should be selected for forming the packs along the roadside, and also the portion next the face, filling in the interior with the loose, small stuff. 2 ft. in thickness at least should be built of large, solid pieces of rock (where obtainable), and care should also be taken that

the packwall is built up perpendicular between the floor and the roof. If this is not done, or is carelessly done, the packwalls will soon be crushed out into the roadways, causing much inconvenience and expense to keep them in proper repair. If the ripping does not yield sufficient large pieces for the roadside buildings, disused broken timber may be added to strengthen them. In timbering at the face, single props are usually set with a 'lid' or 'bonnet' on the top. The 'lid' may be either a square, half-round, or round



FIGS. 213-217.—Methods of supporting roof.

piece of timber. Very often fractured props are sawn in pieces for this purpose. The lid prevents the weight from coming on the prop suddenly and fracturing it; it also gives additional resistance to the prop, and spreads the pressure over a larger area. In mines where the pavement (floor) is hard and slippery, the props are often pointed, so that they will crush up when the pressure comes on, and grip the floor, and thus be prevented from sliding away at the foot. Figs. 213 to 217 show some of the methods of setting single props adopted at the working face. Where the floor is of a soft nature it is frequently the practice to set the prop on a cap, to act as a sole-

piece to prevent the prop from sinking when the weight comes on. If this is not done, the prop may cut the floor, and fall out or give little support to the roof.

Tapered Props.—We have already mentioned that when the floor of the seam is hard and slippery the props are sometimes tapered at the foot, so as to allow them to crush up when the weight comes on. This system has been used to a limited extent for a long number of years in collieries in Scotland, but during the past ten or twelve years the use of tapered props has been greatly increased.

In the Scotch mines the practice was for the underground workmen, usually oncostmen, to taper the props by means of a hatchet, and of course such a method could only give a small supply.

In 1899 Mr W. H. Hepplewhite, one of H.M. inspectors of mines, took out a patent for tapered props, and at the same time invented a machine to do the tapering on a definite principle. From that time the advantages of tapered props were brought more prominently before mining officials, although there was really nothing new in the idea, as the writer had mentioned the fact of tapered props being used in the first edition of this work, the paragraph mentioning the fact being written before Mr Hepplewhite had taken out his patent. We think it expedient to state this, as an idea has got abroad that the use of tapered props only date from their introduction by Mr Hepplewhite in 1899. Of the benefits of using such props there can be no doubt, under certain conditions.

It is well known that in longwall working especially there is a creeping down of the roof going on continuously, with the result that when the floor of the seam is hard and strong, large numbers of props very soon get fractured, generally by buckling near to the centre, or sometimes they split longitudinally. Face props at the best can only be looked upon as temporary supports; that is, they are set for the purpose of supporting the roof temporarily until the line of face has been advanced and a new area of roof is exposed. Mr Hepplewhite states that the length of the taper required on the prop depends upon its length and diameter, and also on the rate of depression or sinking of the roof behind the line of working face. Thus he estimates that with a prop 5 ft. long and 6 in. diameter, with a roof depression of 6 in. 6 ft. back from the line of face, the length of tapering required to give the best results should be about 11 in., leaving the thin end 3 in. diameter. When the weight comes on the thin tapered end crushes or 'burrs' up, and prevents the props from being fractured, and hence there is a saving in the number of props required. The props can also be withdrawn more readily, and after the crushed part is cut off, can be used over again in a thinner seam, or in the same seam with a thick sole-piece placed underneath them. Some of the advantages claimed for the use of tapered props are: greater economy, efficiency, and safety; distributes the roof weight more regularly, which assists

in breaking down the coal ; and also that a better roof is kept. In one colliery it was stated that there was a saving of 25 per cent. in the number of props used after tapering was introduced.

Wooden chocks or pillars are also largely used for supporting the roof in main roads, particularly in longwall workings where there is insufficient rubbish to completely pack the entire goaf.

These wood chocks are composed of either round or square pieces of timber, from 3 to 6 ft. long, and sometimes larger (fig. 218).

The pieces of timber are laid lengthwise horizontally, and another layer is laid on at right angles, this alternation being continued up to the roof, the topmost layer being driven tight with wedges.

The open spaces between the layers of timber are filled in with rubbish when it is available. If the chocks have to be withdrawn, as they often are in longwall faces, they are built on a layer of loose

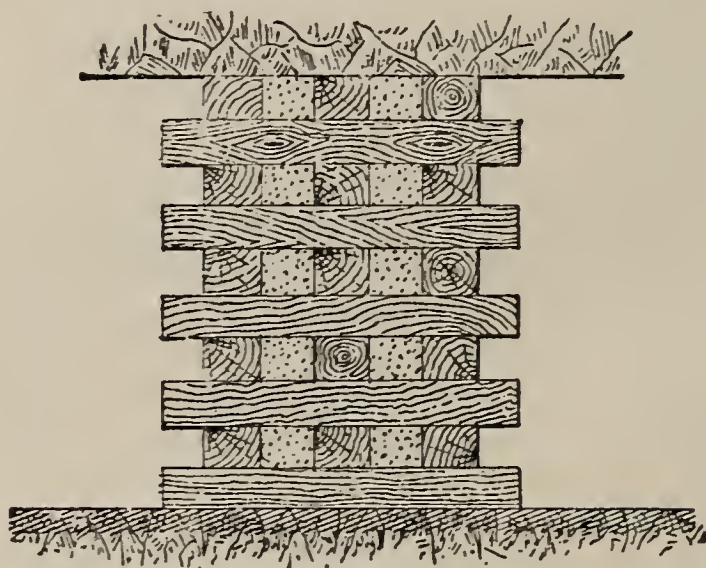


FIG. 218.—Wood chock.

material, 3 or 4 in. thick, which facilitates their removal when required (fig. 218).

In setting props at the face it is unusual to set them at right angles to the inclination except in flat seams ; the greater the inclination the more the props are set off the perpendicular, or 'underset.' The props when set should lie towards the rise of the seam ; this is found to give the best results, for when the weight comes on it tightens up the prop and tends to bring it nearer the perpendicular. On the other hand, they should not be too much underset, as in such circumstances they offer less resistance to the roof, while, as they also require to be longer, they are consequently weaker.

The amount of 'underset' or inclination given to the props ought to vary with the degree of dip, for props set with the same inclination in a steep seam as in a comparatively flat one, would be apt to fall out before the pressure of the roof had tightened them.

Mr A. R. Sawyer gives the following table, which shows the

maximum and minimum angles at which props should be set in varying inclinations.

Degree of inclination of seam.	Angle or 'underset' of props.			
		Minimum.		Maximum.
6°	...	0°	...	1°
12°	...	0°	...	2°
18°	...	1°	...	3°
24°	...	1°	...	4°
30°	...	2°	...	5°
36°	...	2°	...	6°
42°	...	2°	...	7°
48°	...	3°	...	8°
54°	...	3°	...	9°

Systematic Timbering.—The systematic timbering of underground workings—especially those parts of the workings where the miners are immediately employed, *i.e.* the working face—is a subject which has attracted a great deal of attention within the last few years. When it is remembered that of the total number of underground



FIG. 219.—Systematic timbering.

accidents more than half of them are due to falls of roof and sides, it cannot be wondered that more attention should be paid to the proper and systematic timbering of the mine workings. Nearly every mining district has now adopted special rules for the guidance of the workmen in the propping and securing of their working places, but it cannot be said that the effect of these rules up to the present time has made any appreciable reduction in the percentage of accidents resulting from falls of roof and sides. From the writer's own experience the failure to reduce the number of such accidents is not so much due to the unsuitability of the rules, but from the neglect of the workmen to carry them out in their everyday work, and in not a few cases from the laxness with which they are enforced by the colliery officials. Unless such special rules are enforced and carried out in their entirety, we need look for little diminution in the large number of accidents which take place every year from falls of roof and sides.

Systematic timbering of the working face usually takes the form

of setting the props at a specified distance apart and in a regular manner, so as to ensure the greatest safety to the workmen. Fig. 219 shows a longwall face secured by systematic timbering, where the props are set as nearly as possible in a straight line and a specified distance apart. Chocks *aa* (fig. 219) are put in opposite the open spaces in the goaf or waste, and in front of these are kept one or two rows of props, *bb* and *cc*, set at regular intervals. As the face moves forward the chocks are withdrawn, the packwalls extended, and the chocks placed in a new position. The back line of props *bb* is then withdrawn, and set up again in front of the row *cc*. It should, however, be borne in mind that no special system of timbering will be suitable for all seams; each seam and each colliery will require to be timbered and supported as demanded by the special circumstances prevailing, so that no rigid code of rules can be laid down to be applied universally to all collieries.

Courrieres' System of Timbering.—This is a system of timbering which is practised at the Courrieres Collieries, near the town of Lens, in the Department of the Pas-de-Calais, France, and is said to have greatly diminished the number of accidents from falls of roof in the working faces of these pits. The system consists in regular timbering, and chiefly in the use of a series of short iron bars to form a sort of temporary shield next the coal face in advance of the last row of props. Figs. 220 and 221 illustrate the system. The iron bars *aa*, $1\frac{3}{8}$ in. square and 4 ft. 3 in. long, are driven forward next the roof and supported by cross crowns *cc*, the latter in turn being supported by vertical props *ee*. The iron bars are placed 15 to 20 in. apart, and securely fixed at the ends by wooden wedges. As the face advances and another row of props is put in, the bars are then withdrawn, and driven on in advance beyond the new set of supports. Each workman is supplied with three of the iron bars, and it is said that the men have become so practised in their use that it takes them a very few minutes to knock out the wedges, drive the bars forward, and wedge them up again. If the bars get bent they are easily straightened by the smith at the colliery, and the loss of bars is trifling. About six thousand of these iron bars are in daily use at the Courrieres Collieries. It will be seen that this method is very little different from the system known as 'spiling,' which is described earlier in this chapter, and which was known and used for a long number of years in this country, where the roof was of a soft and friable nature.

The peculiar circumstances under which the seams are worked at the Courrieres Collieries enable this system to be practised with good results, but it is very questionable if it is superior to the method of timbering practised in British mines. As already remarked, the conditions prevailing in all mines are not alike, and a system suitable to one may not be suitable to another mine.

A somewhat similar system is employed at Lens Colliery, in the

same district, but instead of using flat iron bars, light **I**-shaped sections are employed. These bars are 4 ft. long and weigh about 11 lbs. One of the ends, for a distance of 8 in., is tapered out to a point by heating and flattening out under the hammer until the two flanges of the **I**-iron are welded together. Each workman is supplied with three of these bars, and he keeps these supports to the roof within 8 in. of the coal face, advancing them in succession 1 ft. 8 in. apart. It is stated that, as the result of several years'

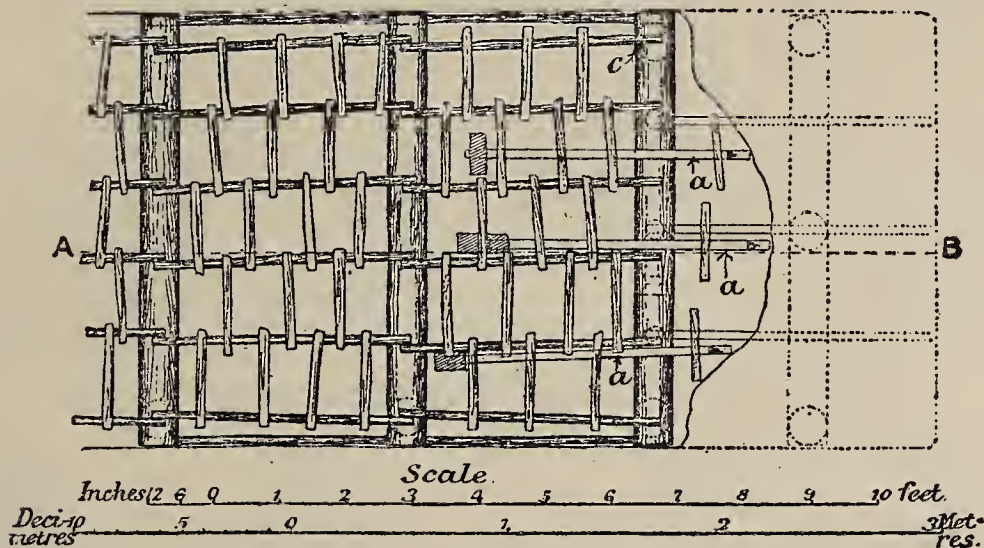


FIG. 220.—Courrieres' system of timbering. Plan of roadway.

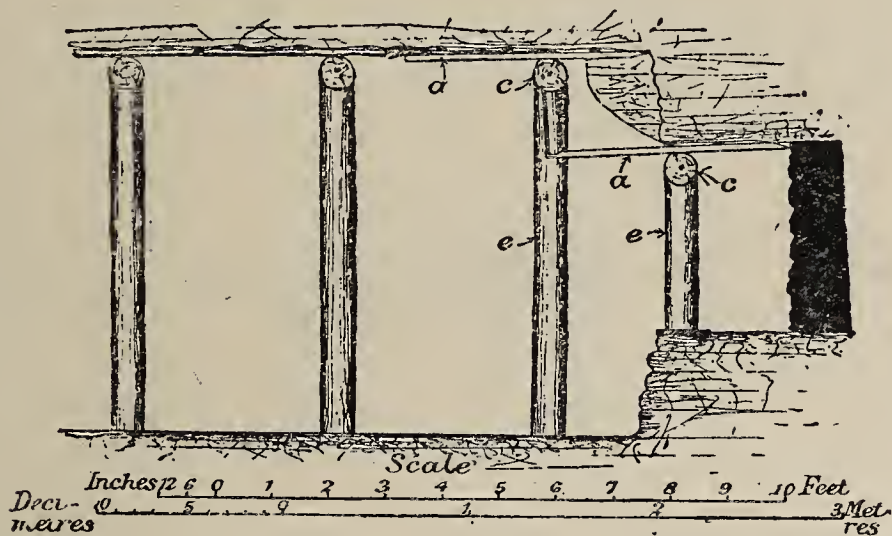


FIG. 221.—Courrieres' system. Section of roadway.

experience of this system, accidents from falls of roof have been reduced 50 per cent.

Withdrawing Timber.—The withdrawing of timber is often required in longwall workings, and in the extraction of pillars in pillar and stall working. Such work is nearly always more or less dangerous, and should only be done by careful and experienced men. The method of withdrawing the timber may either be:—

- (a) Striking the props out with a heavy hammer.
- (b) Drawing them by ringer and chain.
- (c) Drawing them by patent prop drawer.

The first method is largely used in some districts. A flat-headed hammer, 12 to 16 lbs. weight, with a shaft $3\frac{1}{2}$ to 4 ft. long, is generally used, the miner or other workman knocking out the props



FIG. 222 — Ringer and chain.

by repeated blows. Two men are usually engaged, one knocking out the props and the other throwing them back out of the way. The ringer and chain consists of an iron lever (*a*, fig. 222), 5 or 6 ft. long, with a curved point at one end. At a distance of 8 or 9 in. from this curved end a hook is attached to fix the chain on to. Suppose the prop *c* has to be withdrawn; a firm prop *b* is selected as a fulcrum for the lever, and the chain is given a turn round the

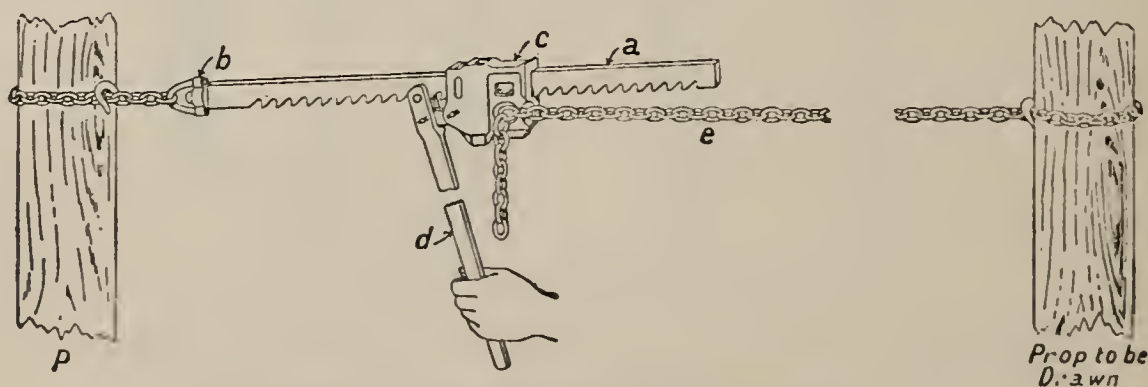


FIG. 223.—Sylvester prop-drawer.

prop *c*. By repeated pulls of the lever the prop is withdrawn. As the leverage is so small this method is not very efficient, unless the props are easily withdrawn. To give increased leverage and efficiency the Sylvester patent prop-drawer or 'pull-jack' was invented. This apparatus consists of a notched bar or ratchet (*a*, fig. 223) 3 ft. long, the teeth being cut along the upper edge 1 in. apart and $\frac{7}{16}$ in. deep. At one end of the bar is a swivel hook *b*, to which is attached a short length of chain for fixing to a firm prop *P*; while at the other end is

a sliding block *c*, connected to a lever *d*, 3 or $3\frac{1}{2}$ ft. long. A chain *e* is attached to the sliding block *c*, this chain being fastened to the prop to be withdrawn. By repeated movements of the lever *d*, which travels on a bolt over the teeth or notches in the bar *a*, a pull is exerted on the prop to be drawn sufficient to displace it. The block *c* may have to travel more than once along the notched bar before the prop is drawn. The leverage got by this apparatus is 30 to 1, three times at least greater than the leverage of the ordinary ringer and chain, and therefore it is much more effective and efficient. The Coal Mines Act, 1911 (52 (25)) makes it compulsory, under certain conditions, to use a safety contrivance for the withdrawal of props.

Timbering the Ordinary Drawing Roads.—The drawing roads next the working faces are usually only of a temporary character, with the exception of the main levels, headings, or slopes, as they are meant only to be kept open for a short period, especially in longwall working. These drawing or gate-roads are in ordinary

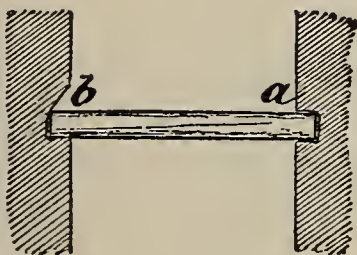


FIG. 224.—Cross-bar.

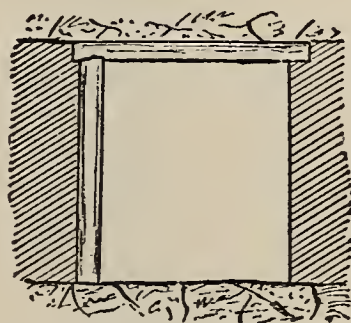


FIG. 225.—Cross-bar and prop.

circumstances driven for 50 to 100 yds., when they are cut off by a level or heading, and new gate-roads set away to take the place of those cut off. In such circumstances the supporting of these gate-roads or side-roads is not carried out in such a permanent manner as in the main drawing roads and main haulage roads. All that is required is to keep them open and secure during the time that they are being driven 50 or 100 yds., after which they are abandoned and become part of the goaf. If the roof and sides are both good, frequently no timber at all is required. Roofs composed of tough shales or fakes will often bend down to a considerable extent without fracturing, and such a roof may require very little timber support. Where a part of the seam itself is left on for a roof, especially coal of a hard coherent nature, no timber is usually required, unless where there is a heavy crush on the workings. Timbering the roadways is practised in a variety of ways, according to the nature of the roof, floor, and sides. When the roof alone requires support and the sides are hard and firm, a cross-bar will be sufficient without it being supported by props. The cross-bar is fixed by 'needling' it into a hole *a* on one side and a tapered rest *b* on the other side for it

to rest on, as in fig. 224; sometimes the bar rests on a prop at one end, and is 'needled' in at the other end, as in fig. 225.

When the sides alone cannot be depended on to support the cross-bar, a common arrangement is to put up a cross-bar or crown, either round or half-round, and to support it at each end by a prop set at a slight angle towards the centre of the road, so as to decrease the span and strengthen the 'crown' (fig. 228). In this method nothing requires to be done to the timber beyond cutting it to the required

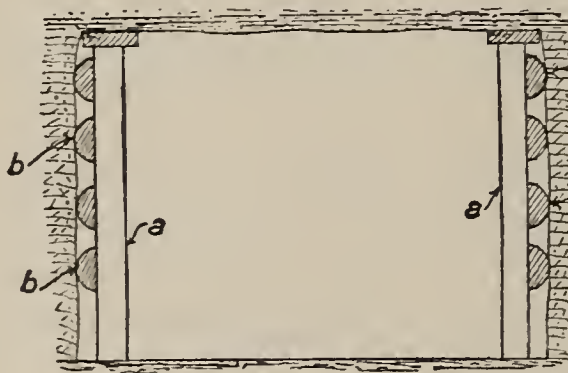


FIG. 226.

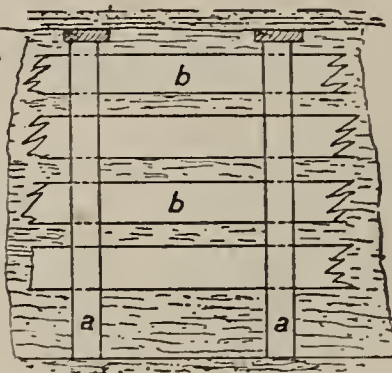


FIG. 227.

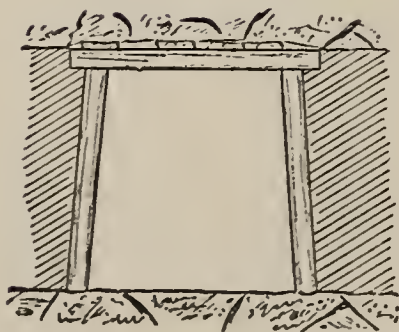


FIG. 228.—Cross-bar and prop.

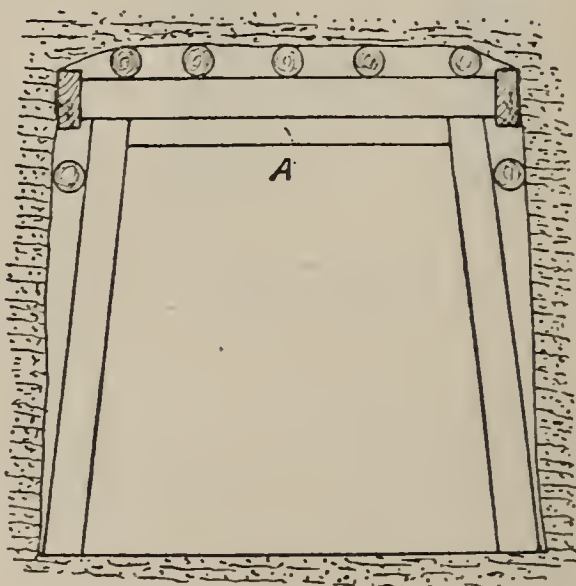


FIG. 229.

length. Or sometimes, with a hard roof and soft sides, single props, *a a*, figs. 226, 227, with a cap are set up at each side, and lagging pieces *b b* put in behind, stretching between the props and parallel with the roadway. If the roof and sides are both weak more care will require to be taken in the selection and erection of the timber sets. The ordinary set shown in fig. 228, if carefully erected and lofting timbers put on the top and lagging pieces at the sides, will resist a great deal of top and side pressure. Sometimes a stretcher piece (*A*, fig. 229) is fixed immediately under the crown and between the props. This has the effect of strengthening the crown and preventing the props from being displaced. Where heavy side

pressure along with roof pressure has to be dealt with, various methods are used in which the timber is cut or notched in a special manner. That known as the Welsh system of timbering is much employed under such circumstances, and is found to give good results when the timber is properly put up. The top of the props are cut to the shape of an inverted **V** (fig. 230), and notches are cut out of the crown corresponding with these. The props are set up at an angle off the perpendicular, so as to decrease the space, and thus increase the strength of the crown. The sides are supported with 'lagging' of either square or round pieces of timber placed longitudinally behind the props. The 'sets' are often prepared at the surface and sent down ready for use. The advantages claimed for the Welsh

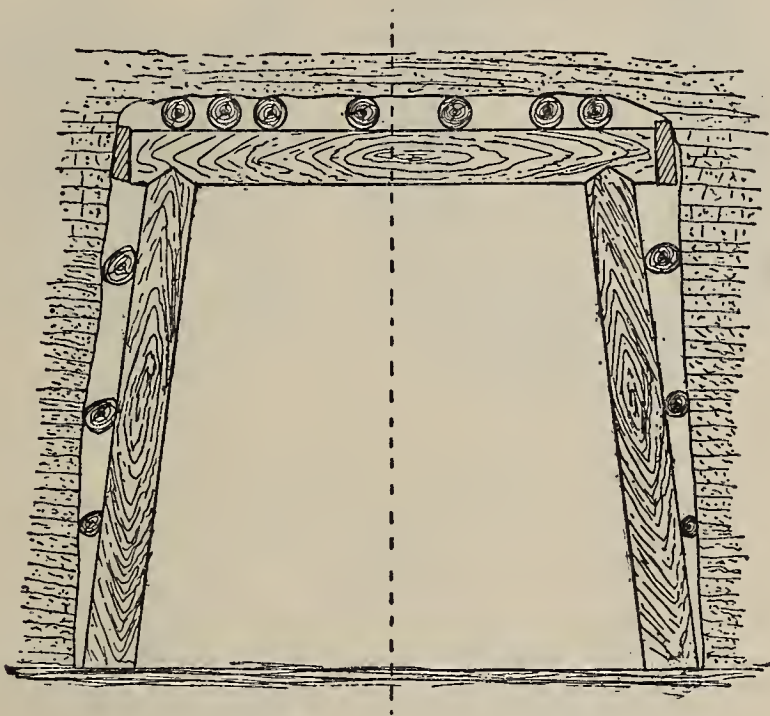


FIG. 230.—Welsh timbering.

system are: smaller cost than in using ordinary timber, as shorter crowns are employed, and greater resistance to side pressure. On the other hand, the increased cost of preparing the wood and setting it must be taken into account.

When the roof and floor are both bad, it is very difficult to keep the roads in good condition by ordinary methods of timbering, and to prevent the floor from creeping, 'sill' pieces are often used, and the props notched into them as well as into the crown-pieces. These 'sill' pieces, however, have their disadvantages where the floor is given to creeping, for when the road requires adjustment, as it frequently does under such circumstances, the 'sill' pieces give trouble and often occasion the danger they were meant to avert.

Main Drawing Roads.—By these are meant levels, headings, and slope roads, which require to be kept open a much longer time than the gate or side branch roads, and on which the haulage may be either manual labour, horse power, or auxiliary mechanical power.

If the roof and sides are both good, the sets already described in figs. 228, 229 may be all that is necessary. Where there is side and top pressure the sets shown in figs. 230, 231 should also be quite suitable. Where the roof is bad and the floor fairly good, 'herring-bone' or double timbering is sometimes practised. A crown-piece is set up (*a*, fig. 232), notched into one side and supported at the other by a prop *b*. At the centre of the crown a longitudinal timber *c*, parallel to the roadway, is fixed and held in position by the two sloping props *d d*, set at an angle of about 45° . On the top of the

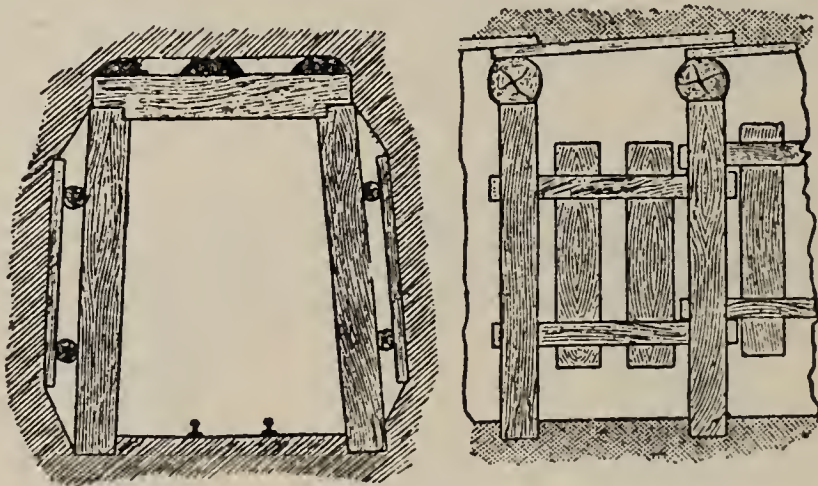


FIG. 231.

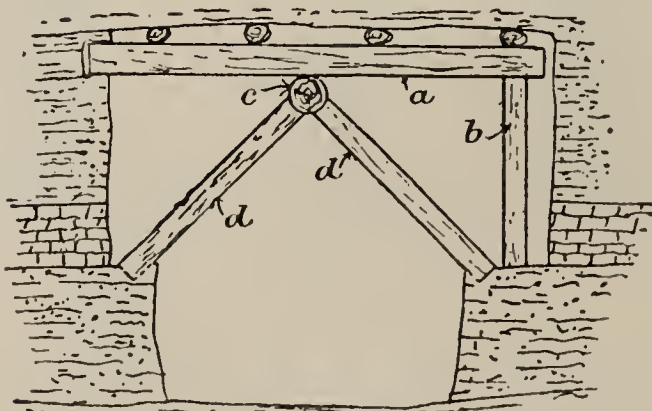


FIG. 232.

crown, lagging or lofting pieces are placed parallel to the roadway and stretching between the sets, the latter being erected 2 to 4 ft. apart, according to the condition of the roof. The ripping, as shown in the figure, has been taken out of the floor instead of the roof.

When the roof and floor are both soft, the system of timbering shown in fig. 233 is sometimes adopted. A cross-crown as before is set, and two short vertical props *a a* are placed from the floor. On the top of these are placed two longitudinal timbers *b b*, and a longitudinal timber *c* is also placed underneath and at the centre of the crown. The two angled pieces *d d* are fixed between the timbers *b b* and *c*. Both of these systems require careful setting, and the main idea in each is to give additional support to the crown at the centre, where

it is most liable to fracture when the top weight comes on. Where the side and top pressure is great, heavy timbers notched and set in the method illustrated in fig. 231 may be used, or the Welsh system as already described may be adopted. If the roadway is driven wide, and the whole width is not required for the passage of the tubs, timber pillars are set up at each side, and then ordinary or special sets of timbers placed between them, as shown in fig. 234.

When the roof is heavy, yet has no bed of hard rock in it, and the roads have to be driven comparatively wide, such as in pony roads and at branchings, it is often very difficult to properly support it. In such circumstances, only the best heavy larch should be used, both for crown-pieces and supports. Figs. 235, 236 show a method of supporting such roads. Crown-trees are set, and temporary props put up to the centre. Other crown-trees are placed at right angles to these, or parallel with the roads along the ends of the cross-bars,

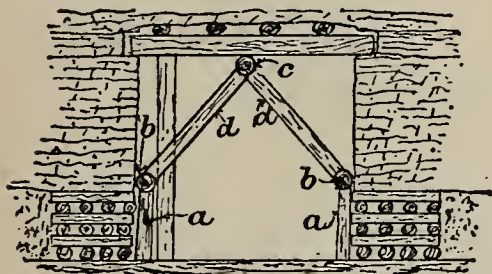


FIG. 233.

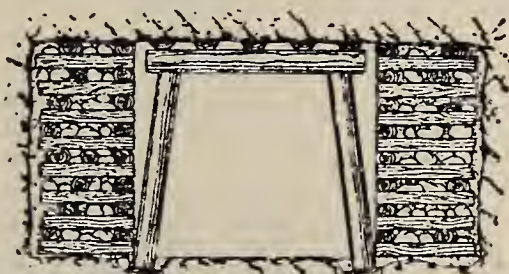


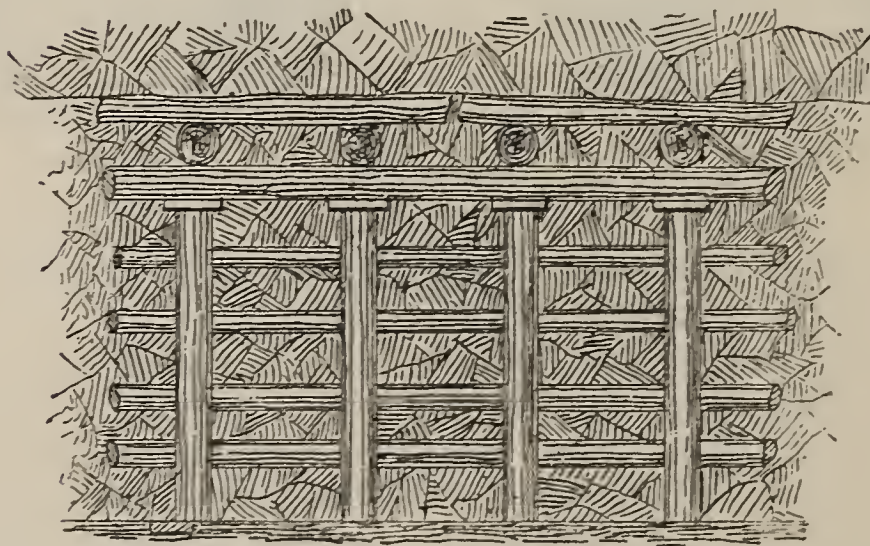
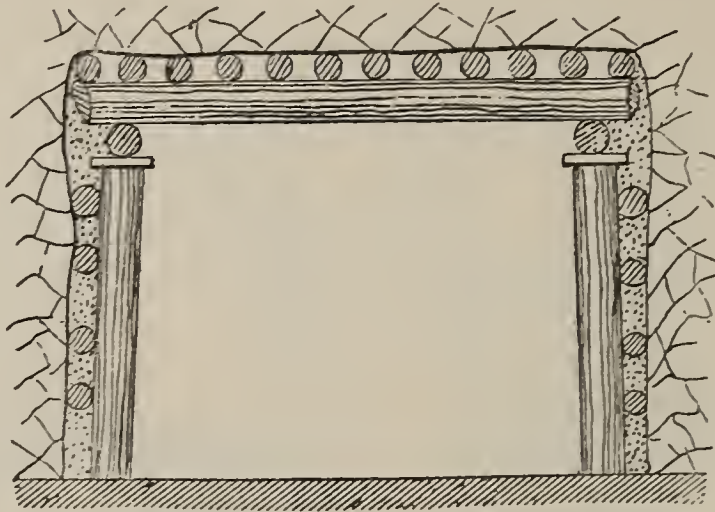
FIG. 234.

and to those crowns parallel with the road the props are set, the whole being firmly fixed with lofting and wedges.

On the Continent, where great attention is paid to timbering, some methods analogous to the 'herring-bone' system are adopted for the support of roads. Figs. 237, 238 show two of these methods. In the former an ordinary set of timber is first placed in position, short props *dd* are then placed close against the main props, and on the top of these other pieces *cc* laid longitudinally and parallel with the roadway. From these longitudinal pieces *cc*, other posts *bb* are set at an angle so as to meet in the centre of the cross-bar, where another bar *a* is fixed in line with the road. On wide roads this system has the disadvantage of reducing the space considerably. To overcome this difficulty the method shown in fig. 238 is adopted.

Here an ordinary set is fixed as before, and the short pieces and the longitudinal pieces *cc* placed as already described. In the centre, and immediately against the crown-tree, is placed a shorter crown *b*, and against the ends of this short piece are two pieces *aa* placed at right angles or parallel with the roadway. From the longitudinal pieces *cc* on the top of the short props, posts are set at an angle to meet the pieces *aa*. By this method increased space is gained in the roadway.

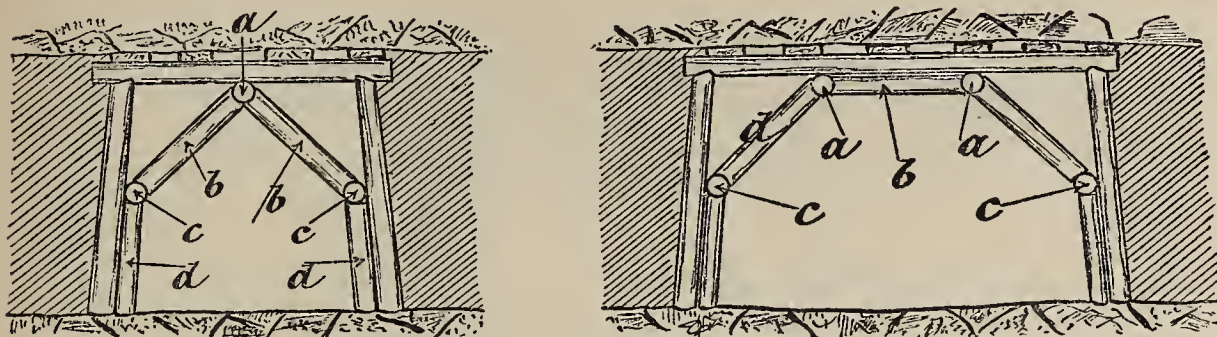
In fixing the timber by either of these methods, the ordinary sets are first placed in position, and then all the auxiliary pieces are fastened to the former by thin wire until the set is completed, and when the pressure comes on the wire is of no more use. These two systems very much resemble the methods described on p. 134, and which are used in some of the Scotch mines where there is heavy top pressure.



FIGS. 235 and 236.—Supporting heavy roofs.

Supporting Wide Spaces.—At the junctions of two or more roadways there are usually large areas exposed which may require support of some kind. If the roof is strong, nothing more than a few props may be necessary, but in softer roofs the support may require to be more systematic. Where one road meets another at right angles the usual custom is to set up a carrying crown or bearer, heavier and stronger than what is necessary in the regular roadway. This carrying crown (*a*, figs. 239, 240) is supported at each end by props, and then the other crowns *bb* are rested on it at one end, and supported by props *dd* at the other side of the roadway. If two roadways meet a third one at right angles, then a carrying crown will require to be set across the mouth of each of the

branch roads, and the crowns supporting the main road rested on these. Instead of using bearer crowns, sometimes a crown-piece (*a*, fig. 241) is set diagonally across the main roadway opposite the two branch roads, but unless for comparatively short spans this system is not so effective as the methods described above.



FIGS. 237 and 238.—Continental methods of timbering.

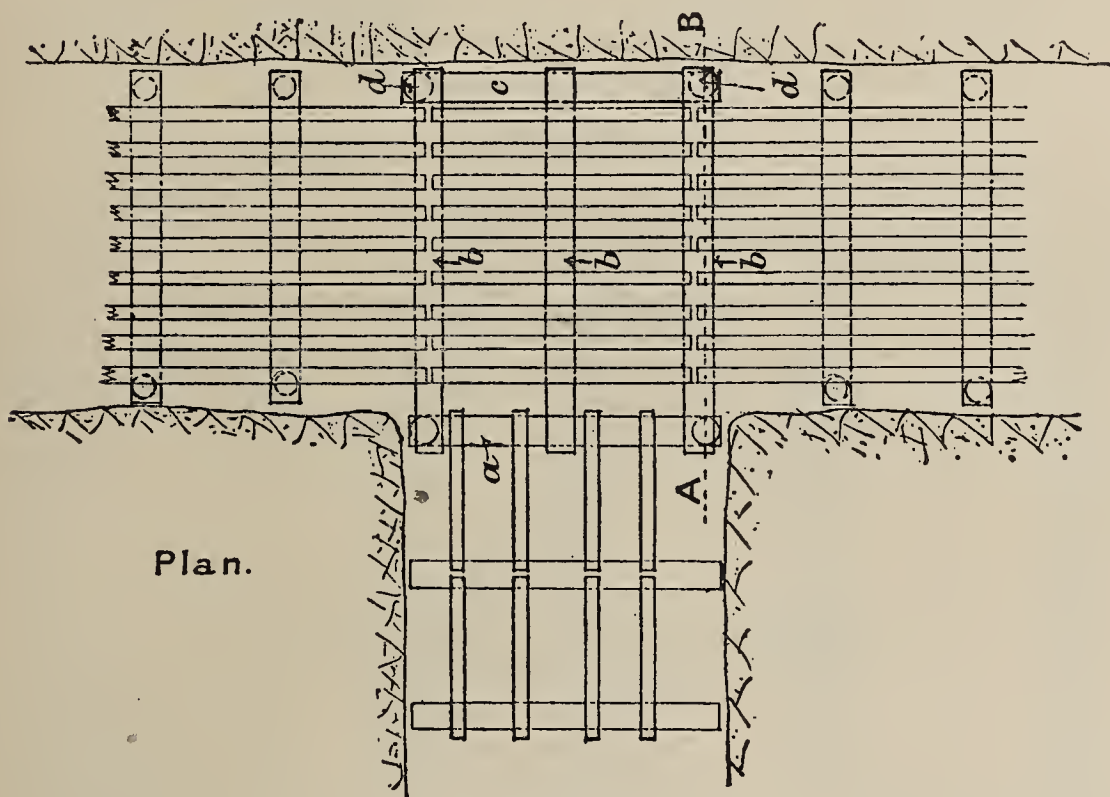


FIG. 239.

Driving through Loose Ground.—When the roof or sides are very loose and have a tendency to ‘spill’ or run, a special method of timbering must be adopted.

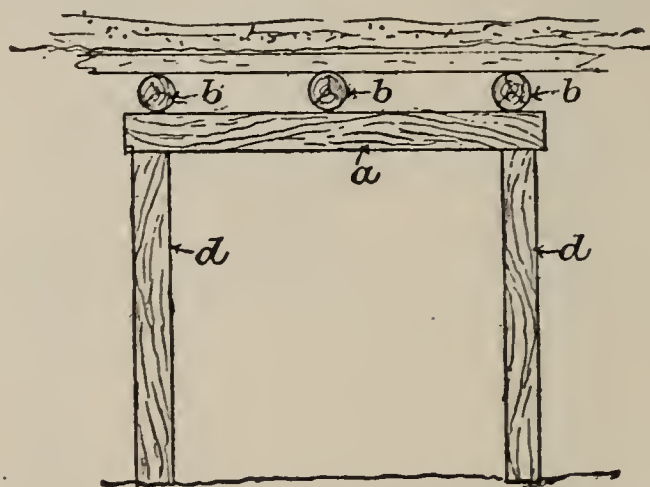
The methods of securing such loose ground are shown in figs. 242 to 244.* An ordinary set is first placed in position with posts *a* and crowns *c*, and if the floor is very soft a sill-piece *d* is added.

Pieces of lagging are then inserted above the crown-trees, and driven in with the ends pointed upwards a few degrees off the horizontal. These continue to be driven forward as the work proceeds.

* “Methods of Timbering,” *Cal. State Mining Bureau Bulletin*, No. 2.

Other lagging pieces are driven in behind the posts, and also inclined *outwards* at the end 10° to 15° , the side pressure gradually bringing them to bear closely against the props.

The two systems shown in figs. 242 to 244 are practically the same in principle, but differ materially in detail. In fig. 242 the lagging is inserted between the two crown-trees, which are separated



Section on A.B.

FIG. 240.

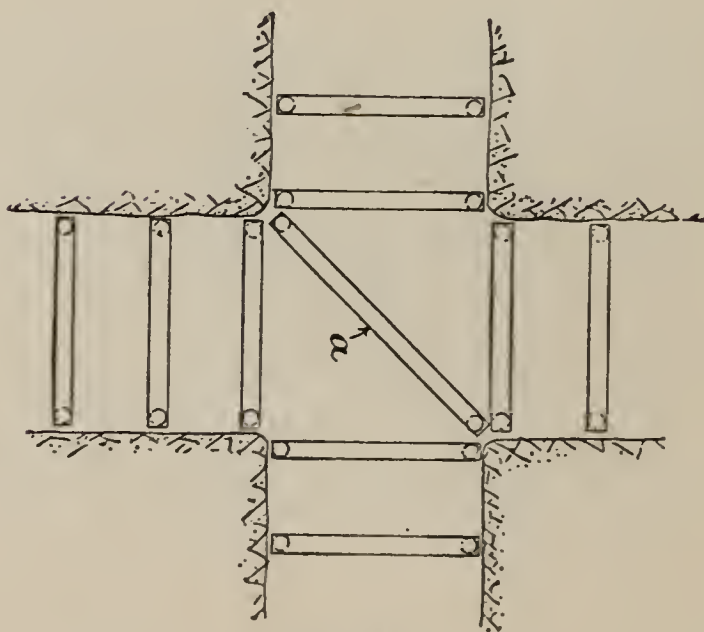


FIG. 241.

by wedge-shaped blocks, one of which is placed at the centre and one at either end. The lagging is then driven forward, as already described. If the ground is very heavy, a 'false set' is erected and the ends of the lagging rest upon it. As the excavation progresses the lagging is driven forward until the further ends find a secure resting-place on the regular sets. The false set is then knocked out, and the same operation repeated with the next set.

The only difference between these two methods is that in the one (fig. 242) there are two cross-bars—one light and one heavy; while

in the other (fig. 244) the lagging is inserted beneath the advancing ends of the set next behind. In both methods the lagging is kept pointed slightly upwards by the insertion of a block of wood *b*, which is placed between the portions already fixed and those being driven. When the lagging is driven forward a certain distance, this block is allowed to drop out.

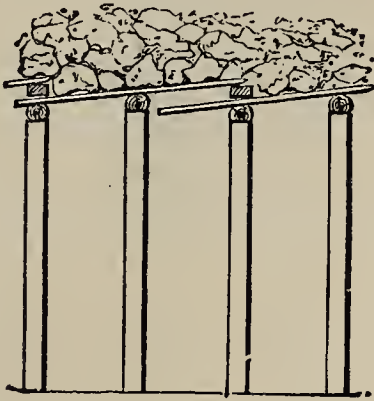


FIG. 242.

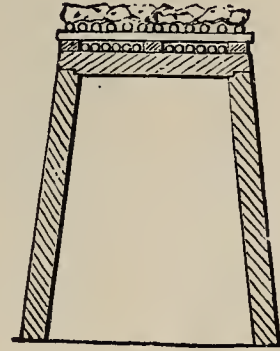


FIG. 243.

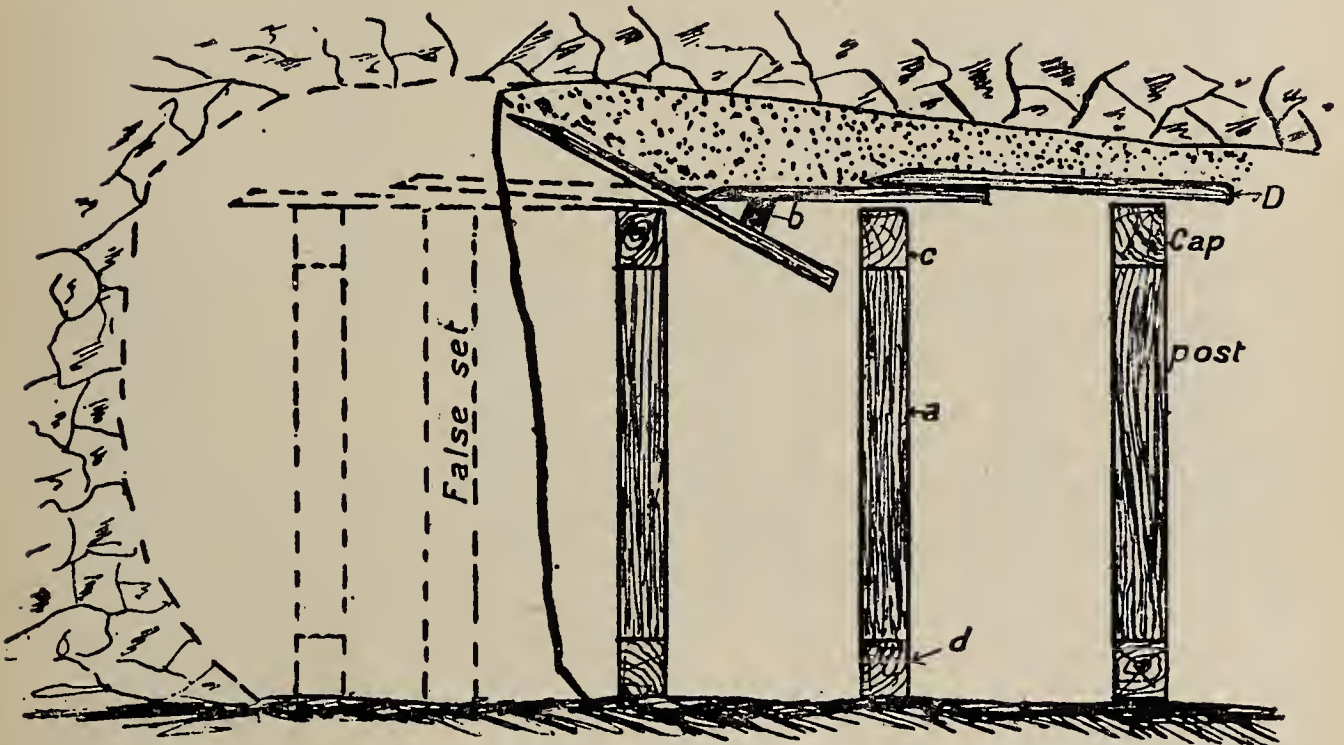


FIG. 244.—Timbering in loose ground.

To facilitate driving, the lagging is sharpened to a point at the one end. With these methods, even with the greatest care a quantity of the loose ground will run through and leave cavities behind.

Haulage Roads.—The supporting of the main haulage roads requires careful consideration, as they are usually required to last a long time, often during the entire lifetime of the colliery itself; and in addition they are the main arteries of the underground working for the passage of workmen and minerals. As to the method of supporting such roadways, there will naturally be a great diversity,

according to the conditions of the roof and floor prevailing. Where the roof strata are composed of hard sandstone, strong fakey sandstone, or tough shales, frequently no timber supports are necessary. There is some difference of opinion as to whether the roof in all roadways should be supported, whether the strata is strong or not. The writer is of the opinion that this is quite unnecessary where the roof is strong and not liable to fall. Timbering under such conditions is not only a waste of money, but often gives a sense of false security. No system of timbering can ensure immunity from falls of roof, as the writer has seen extensive falls take place and accidents happen where the roof has been closely timbered. If close timbering is adopted, irrespective of the conditions, it is difficult to make an efficient examination of the roof, with the result that it may develop into a dangerous condition and escape detection. The late Mr Pickering, H.M. Inspector of Mines,

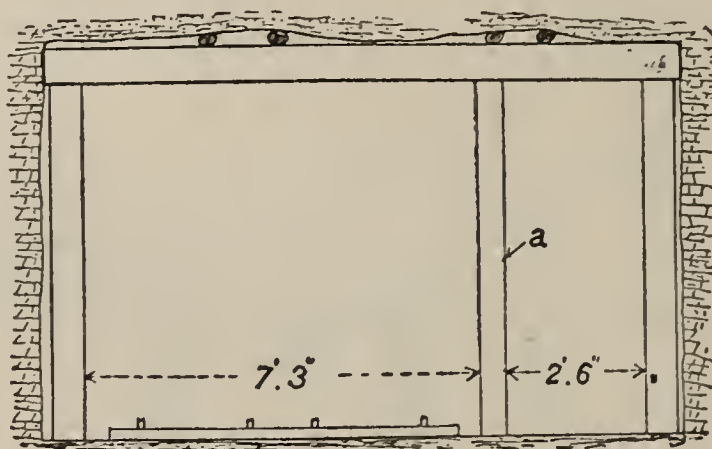


FIG. 245.

in one of his annual reports, says: "In my opinion it is neither necessary nor desirable to timber every part of every underground road as distinct from working places." In that opinion we agree, but at the same time careful discrimination will have to be used and each roof treated on its merits. A careful watch will also require to be kept on the roadway from day to day, in order to detect any crush which may come on.

When the roof and sides are fairly strong some of the ordinary sets of timber already described may be quite sufficient to give the necessary support. Of course, where the haulage roads are wide, 9 to 12 ft., to allow of double sets of rails, the timber used will require to be heavier than that used in the narrower drawing roads, as the weight of the timber ought to bear some relation to the span of roof which it has to support. Where two set of rails are required, as in endless rope haulage, the crown is sometimes strengthened by setting up an additional prop (*a*, fig. 245). Sometimes the sets are composed of square timbers, 9 to 12 in. square, where the roof pressure is great, and lagged or lofted on top between the cross-timbers. If there is

sufficient width, sometimes timber pillars are built up between roof and floor at short intervals, or closely together along the sides of the road, and the roof between these pillars supported by ordinary sets of timber, as shown in fig. 246. In some of the deep German mines,

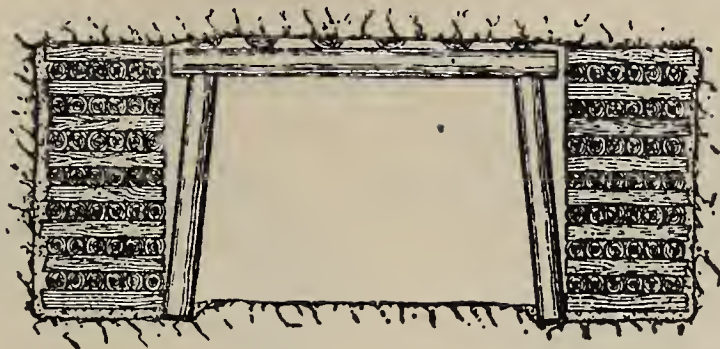


FIG. 246.

where the top and side pressure is very great, the systems shown in figs. 247, 248 have been applied with much success. Timber cogs are built in to a radius, and on top of these radial timbers are fixed; the space all round is rammed tight with concrete. In one case old rails, placed longitudinally, were used in the cogs (fig. 248), and this was found to greatly increase the strength of the lining.

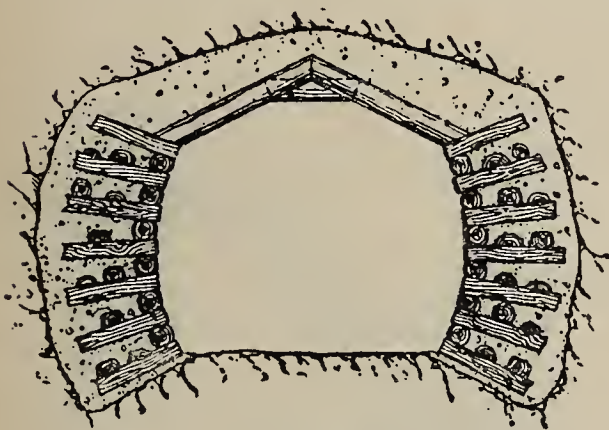


FIG. 247.

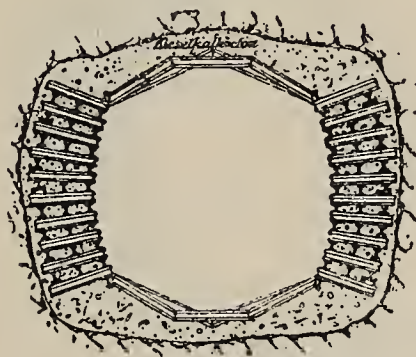


FIG. 248.



Supporting Roadways in Highly Inclined Seams.—Where the seams are highly inclined the method of supporting the roadways has to be carried out on a somewhat different system from that adopted in flat workings. Roadways in steep workings are always more difficult to keep, especially when the strata are soft and broken, as the pressure always tends to act downhill. In the Niddrie Collieries, where the inclination varies from 50° to nearly 90° from the horizontal, the roadways are very difficult to keep open. Roads supported with timber 8 and 9 in. diameter have been found to be sometimes entirely closed within twenty-four hours. Where the strata are hard and firm little support may be required, but frequently under such conditions a crown-piece (*a*, fig. 249) is set to the roof, and

held in position by a prop *b* at right angles notched into the floor. Where the roof is soft and more difficult to keep, the method shown in fig. 250 is adopted; and where both roof and floor are bad, fig. 251 illustrates a system which is frequently practised. In seams which are nearly vertical the systems of timbering shown in figs. 251, 252

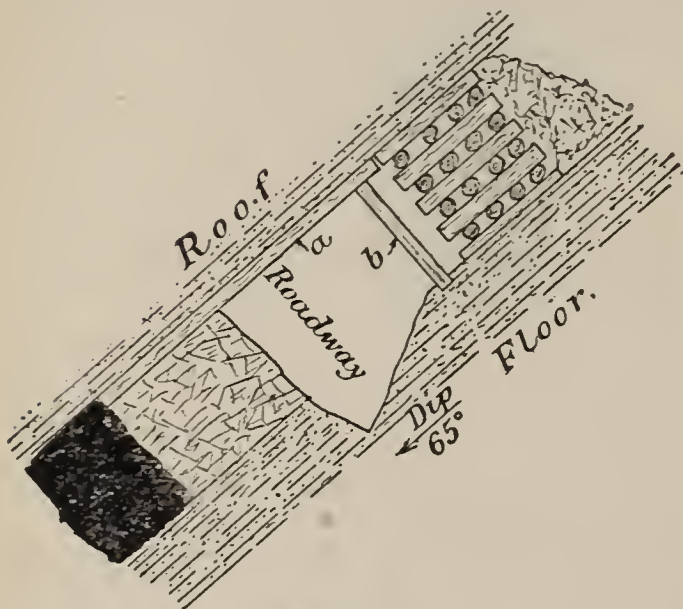


FIG. 249.

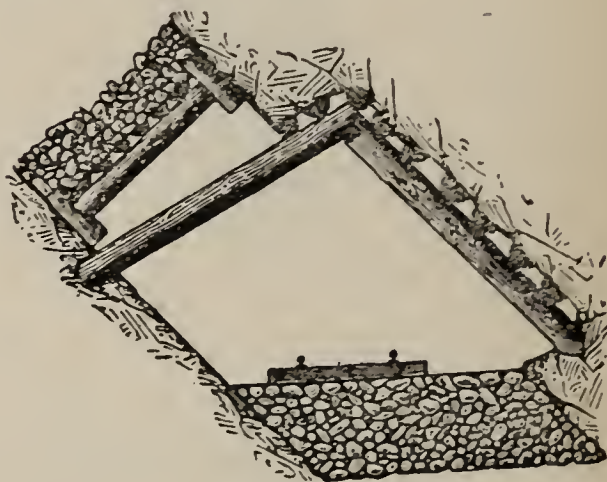


FIG. 250.

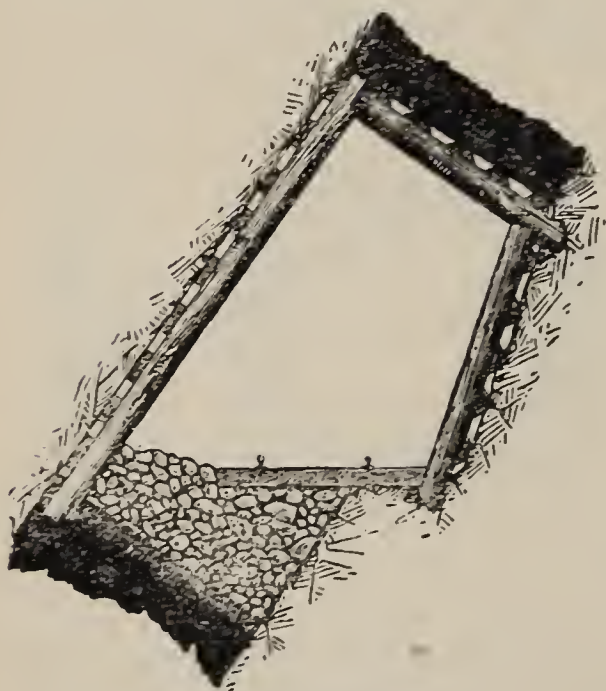


FIG. 251.

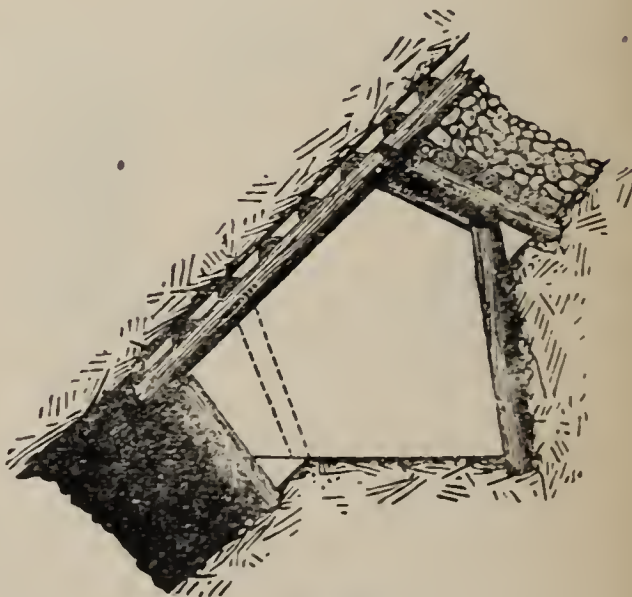


FIG. 252.

are often carried out. In longwall workings where the inclination is great the packwalls or stowage is difficult to keep in position, and would run out into the roadway if not held back in some way. This is usually done by setting props 3 or 4 ft. apart, and putting boards or props behind them (figs. 252, 253), or by building in timber chocks close together along the high side of the road, as shown in fig. 254. Great care must be always exercised in supporting such roadways in steep workings.

Cost of Timber.—The cost of timbering varies greatly in different districts and at different collieries, and even in different sections of the same colliery, and may be anything between $\frac{1}{2}$ d. and 9d. per ton of coal raised. It will vary with the nature of the roof and



FIG. 253.

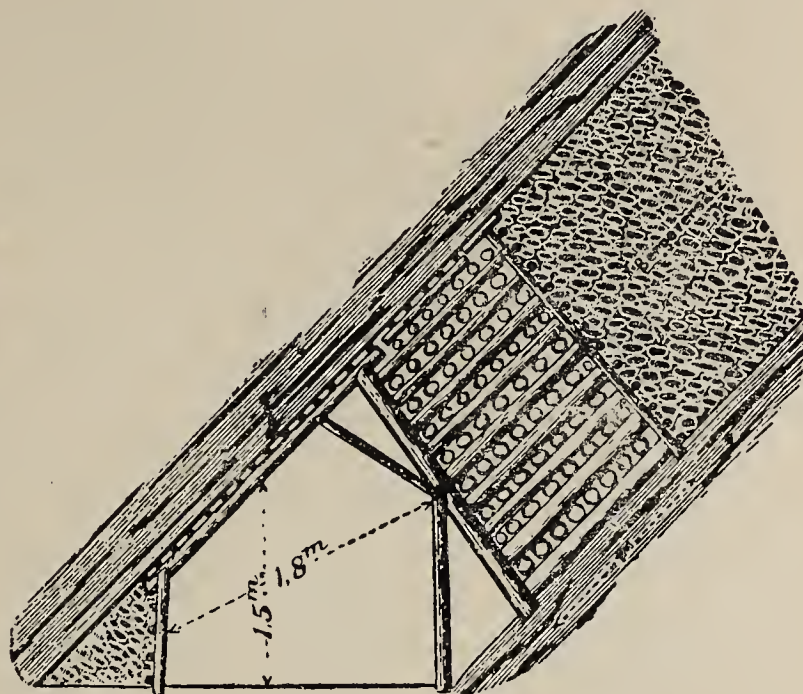


FIG. 254.

floor and of the coal, the inclination of the seam and its depth from the surface, and also according to the method of working employed. The following table shows approximately the cost of timbering in a number of collieries in Great Britain.

TABLE SHOWING COST OF TIMBER PER TON OF COAL RAISED.

No. of Colliery	Thickness of Seam.	Depth from Surface.	Nature of Roof.	Inclination of Seam.	Method of Working.	Cost per Ton.
	Feet.	Feet.				Pence.
1	$2\frac{1}{2}$ to $3\frac{1}{4}$	1000	Fairly good	Comparatively flat.	Longwall	3
2	2 to 4	1050	„	„	„	$2\frac{3}{4}$
3	2 to 5	1100	Soft roof	„	„	4
4	$1\frac{1}{2}$ to $1\frac{3}{4}$	1550	Bad roof	1 : 5, and upwards	„	$8\frac{1}{2}$
5	$2\frac{1}{2}$ to 4	100 to 250	Fairly good	Flat, to 1 : 15	Longwall and pillar and stall	$2\frac{3}{4}$
6	$5\frac{1}{2}$	1050	Soft shale	1 : 9	Double stall	8
7	$6\frac{1}{4}$	850	„	Flat, to 1 : 14	Longwall	5
8	$5\frac{1}{4}$	1200	Shale	1 : 20	„	$2\frac{1}{2}$
9	7	1700	Good roof	Flat	Pillar and stall	$1\frac{3}{4}$
10	$5\frac{3}{4}$	1600	„	1 : 16 to 1 : 18	„	$2\frac{1}{2}$
11	$6\frac{1}{2}$	600	Bad roof	1 : 20	Longwall	3
12	5	1800	Good roof	1 : 12	„	1
13	5	1200	Fairly good	„	„	2
14	4	1150	Good roof	1 : 6	„	$\frac{1}{2}$

In the Government collieries of the Saar district of Germany, the cost of timber is estimated to average 6d. per ton of coal raised.

Iron or Steel Supports.—Within recent years the use of iron and steel supports for underground workings has greatly extended, and, in certain circumstances, it is to be recommended. It must, however, be remembered that the conditions under which steel or iron girders can be used underground are altogether different from the conditions under which the same materials can be used at the surface. In the latter case, all the conditions to be satisfied can be accurately predetermined, the size and resistance of any supports required being ascertained by calculation.

Underground, these conditions can scarcely be ascertained at all, or, at least, only very partially; the top weight to be supported may be unknown, and further complications are introduced when heavy side pressure is encountered. Instead of the load being uniform, as on the surface, it is very varied, and the supports are subject to great and suddenly applied pressure. Steel girders, however, seldom break under sudden pressure or weight, but nearly always give indications of such pressure by deflecting in the centre. Girders have been known to show 5 to 7 in. of deflection under great top pressure before breaking. The writer has seen steel girders, with a span of 9 ft., supporting a haulage road, deflected at the centre to 10 in. without fracturing.

In a large number of mines there are main haulage roads and horse roads where the strata have settled, and where the pressures are fairly uniform. In such roads, steel or iron supports can be used with advantage. Again, in return air-ways where the air is hot and foul, and contains a good deal of moisture, wood very rapidly decays and requires frequent renewal, and, in such circumstances, steel or iron supports may beneficially replace it.

Iron or Steel Props.—While iron or steel is better suited for use as cross-bars or crowns, props made of these materials have also been used at the face of longwall workings. They are, of course, much more expensive than timber props, and it is therefore necessary that they should be used only where they can be withdrawn and re-set, otherwise the cost would be too great. Cast-iron props are not of much use, as they are heavy and easily broken.

Steel girders, of H-section, present a somewhat sharp and uneven surface to the roof or to the timber lids when used. To overcome this difficulty Firth's patent prop is used.

A piece is cut out of the web at each end, and the top and bottom flanges turned over, which enables the ends to present a flat surface to the roof and floor. A hole *aa* is punched in the web, about a foot from each end, for the insertion of a hook to assist in withdrawing the prop (see figs. 255, 256).

Steel or Iron Sets.—These are used in many collieries in Great

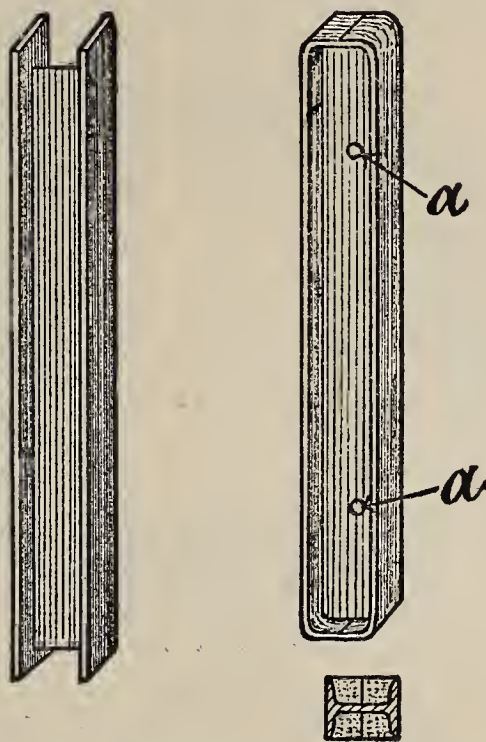
Britain for supporting main roadways, particularly for roads where the strata have settled.

Steel or iron supports may be used in three distinct ways: (a) combination of steel with timber; (b) sets composed entirely of steel or iron; (c) combination of steel or iron with brickwork.

The first method is largely employed both for supporting main drawing roads and main haulage roads. A cross-girder is placed next the roof and supported at the ends by heavy upright timber props, as shown in fig. 257. To prevent the props from being displaced a Z-piece (*a a*, fig. 258) is sometimes used. This Z-piece grips the top of the prop and the end of the girder. The girders themselves should be well braced by fixing horizontal stay-pieces between them to prevent canting and displacement.

At the Nunnery Colliery, Sheffield,* the main roads were supported by steel girders, which were themselves supported on props of larch wood (see fig. 259). The girders were I-section, 4 in. wide, 5 in. deep, with a web $\frac{3}{8}$ in. thick, and this was calculated to give the same support as a Norway larch beam 12 in. square.

The girders were supported on props of larch, 8 in. to 10 in. diameter, and the sets were put in about 3 ft. apart, with lagging above.



FIGS. 255, 256.—Steel pit props.

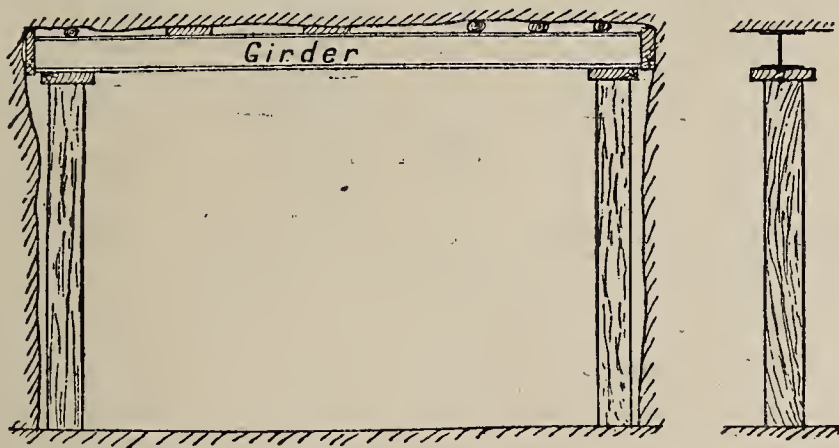


FIG. 257.

To prevent the girders from being pushed out at the top by side pressure, a lug or band of iron, $1\frac{3}{8} \times \frac{3}{4}$ in., was shrunk on at each end immediately in front of the prop.

* *Ore and Stone Mining*, Sir C. Le Neve Foster, Sixth Edition, p. 273.

Instead of using timber supports for the girders the sets may be wholly composed of steel or iron, although many prefer the timber, as it takes the weight better than steel.

The simplest steel set is to use a crown or cross-piece of H-section, and support it by legs or props of a similar section. Such sets have, however, to be very carefully erected and watched, as the uprights are apt to slip when the weight comes on. To obviate this

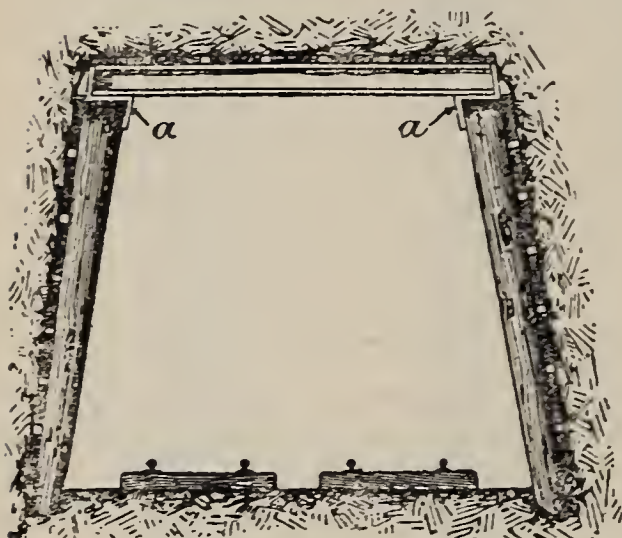


FIG. 258.

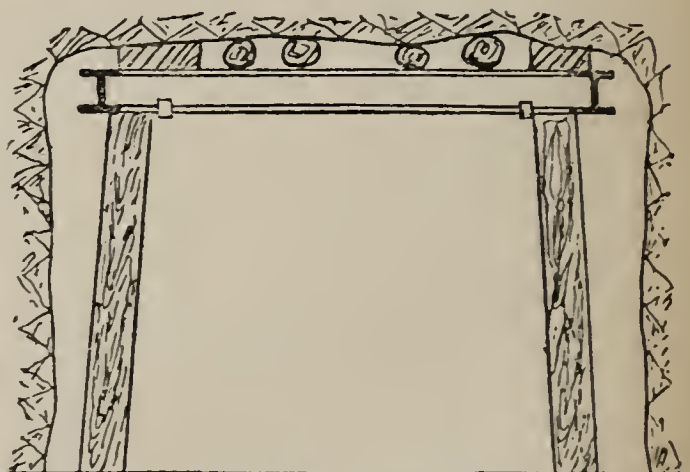


FIG. 259.

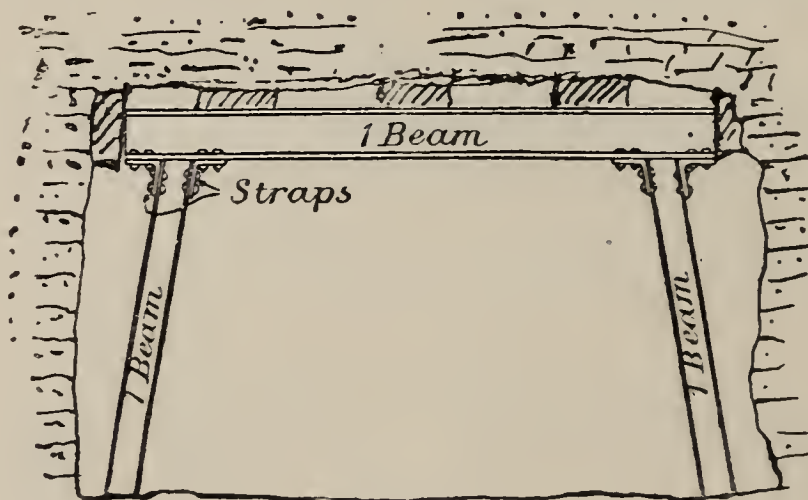
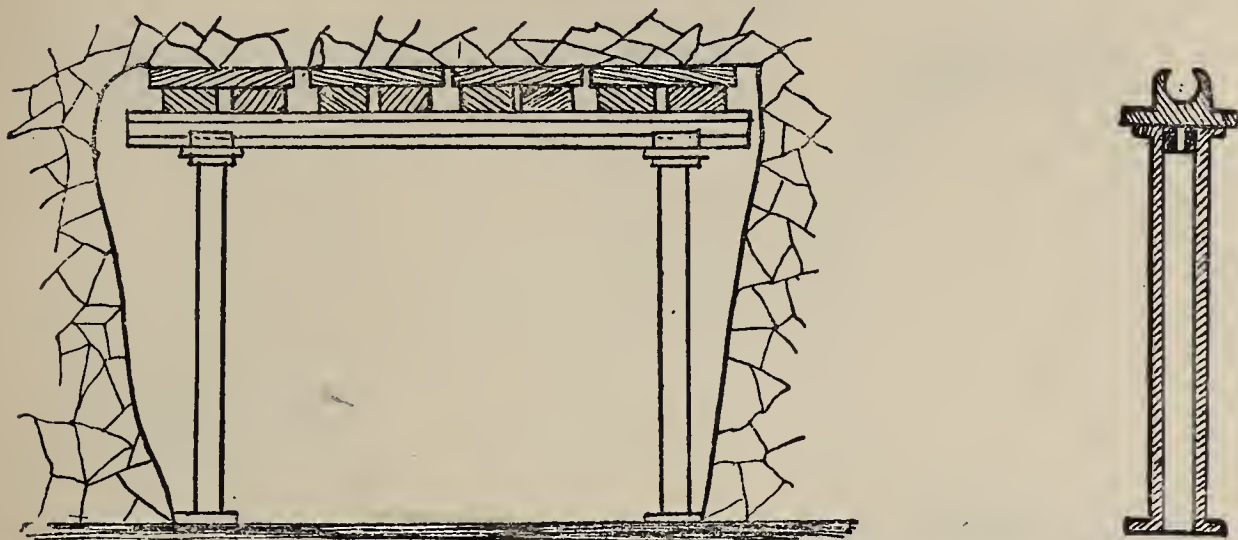


FIG. 260.

the uprights may be fixed to the cross-piece by means of straps or light T-pieces bolted on, as shown in fig. 260.

In some collieries in Staffordshire, hollow cast-iron props are used. These props have a flange 8 in. diameter at top and 9 in. diameter at bottom (see figs. 261, 262). A chair made for the purpose drops into the top of the iron column and receives a reversed iron rail weighing 50 lbs. per yard. These sets are placed 3 ft. apart and are lofted on top with planks or rails, the spaces between these being tightly packed with stones. This method of support makes a capital roadway, but is best suited for roads where no great side pressures exist. The cost of this system is £2, 4s. per lineal yard in an average-sized roadway.

At St Helen's Colliery, Cumberland,* flat-bottomed steel rails were used to support a main haulage road at a depth of 170 fms. Posts of the same material were likewise employed to support the cross-pieces; figs. 263, 264 show how the rails were secured. The crowns were 10 ft. long and the supporting props 6 ft. The latter were set 6 in. off the perpendicular, and were cut and dressed so that



FIGS. 261 and 262.—Steel rails supported on hollow cast-iron props.

the crown would rest evenly on the top. The lower end of the props rested in a cast-iron sole (fig. 263), arranged to give a solid foundation. Near the top of each prop, and also near the ends of each crown, were drilled two holes $\frac{3}{4}$ in. diameter. An angle-iron *b* (fig. 263) was then riveted on the end of the crown so as to fit the upright to which it was bolted, and in this way the legs were fixed and prevented



FIGS. 263 and 264.

from being pushed out. The sets were placed 2 to 3 ft. between centres and lagged on the top with 3 in. planking, and also, when they could be had, with old rails. A piece of wood *A A* was fitted in between the uprights to further strengthen them. The rails used weighed 79 lbs. per yard, and cost 3s. 6d. per cwt. delivered; the cost for steel H-girders, weighing 54 lbs. per yard, would have been 5s. 9d. per cwt. The cost per lineal yard for this method of supporting the roadway, including rails, labour, etc., was £1, 14s. 7d.,

* *Trans. Min. Inst. Scot.*, vol. xiv. pp. 242-249.

while the cost of brick arching, 14 in. thick, for the same road, was estimated to be £2, 5s. 2d. per lineal yard, showing a difference of 10s. 7d. per yard in favour of the rails.

Instead of straight cross-pieces and uprights the sets may be constructed of two pieces, usually H-girders, curved to the form of a horse-shoe to suit the roadway, and joined in the centre by two fish-plates fastened by four bolts, as shown in fig. 265; fig. 267 showing the details of the fish-plate. To strengthen the sets the space between the webs is filled up with planking $1\frac{1}{2}$ to 2 in., forming a neat and strong lining to the road.

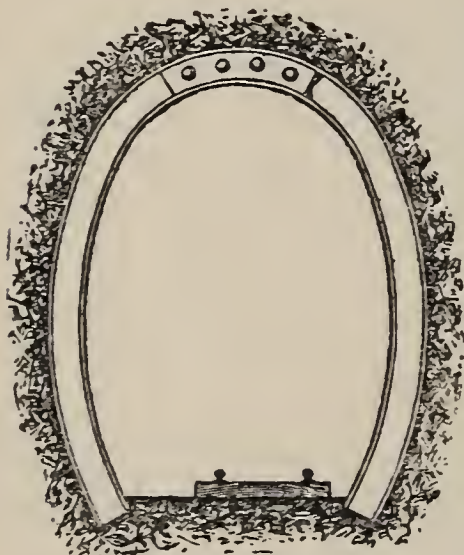


FIG. 265.

In the Government Lead Mines, in the Hartz Mountains, the roadways are supported with flat-bottomed rails, $14\frac{1}{2}$ lbs. per yard section. The ends of the iron arch are lodged in holes drilled in large stones (see fig. 268) set in the floor and fastened by wooden wedges or cement. Between these stones other stone blocks are inserted, in order to keep them apart and thus ensure stability. The lining or lofting is carried out with the same kind of rails, each 19 ft. 8 in. long, arranged longitudinally, the flat bottoms being in contact with the base of the flat rails. The cost of supporting roadways in this manner is about £1, 13s. 2d. per yard. Supporting the same roads with masonry costs £2, 9s. per yard, and with timber £1, 2s. per yard (first cost only).*

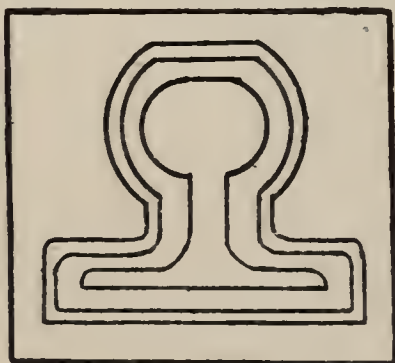


FIG. 266.—Details of sole-piece.

When roof and floor are both soft, and the floor given to creeping, the support is sometimes made in two or three pieces curved to suit the roadway and fastened at the joints by fish-plates and bolts as before (see fig. 269). A cast-iron sleeve is often used instead of the fish-plates. The sleeve is made to slip over the end of the girder, and when the pieces are fitted together it is drawn over the joint

* *Trans. N. Eng. Min. and Mech. Engrs.*, vol. xxxvii. p. 137.

and fastened with wood keys or wedges. Fig. 270 shows the construction of such a sleeve.

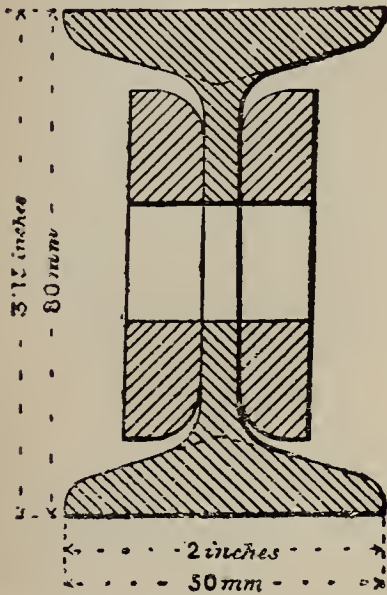


FIG. 267.—Details of fish-plate.

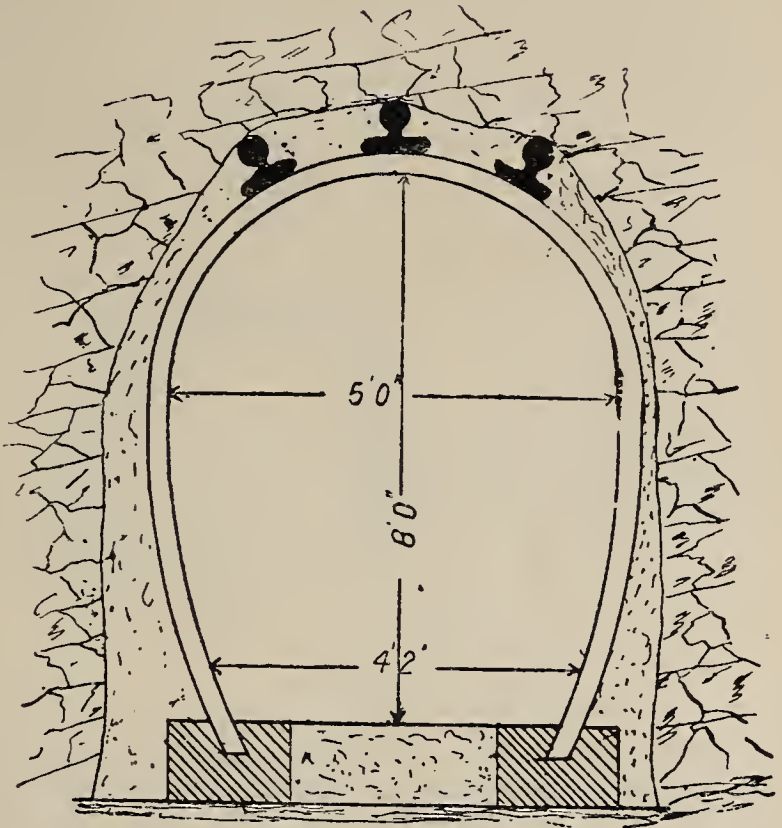


FIG. 268.

At the Altenwald Coal Mines,* near Saarbrücken, iron supports are used in the form of an elliptic arch. To prevent the supports from shifting, horizontal props are inserted from arch to arch at the highest points. The plank covering is of oak, and each plank overlaps the other, to allow some play under heavy pressure. The cost for this kind of support was £2, 18s. 2d. per yard.

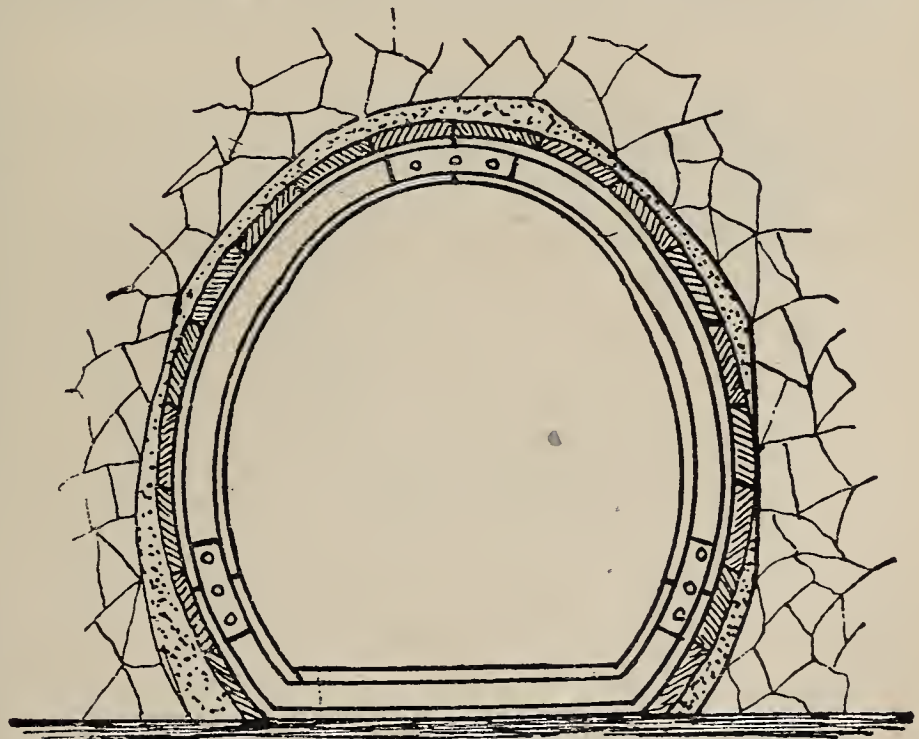


FIG. 269.

Brickwork would not have been applicable in this case, owing to the continuous settling of the floor.

* *Trans. N. Eng. Min. and Mech. Engs.*, vol. xxxvii. p. 138.

At a deep colliery in Fifeshire where the roads were very difficult to keep open, owing to the very bad roof and the great crush, sets like those just described were used with much success. The sets were composed of two steel H-girders, curved at the top to the radius of a circle, but the sides were kept straight. These sets were erected at 2 to $2\frac{1}{2}$ ft. centres, and the space between the webs of the sets

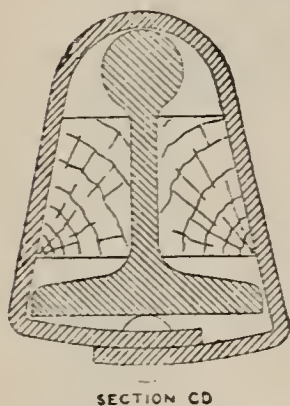


FIG. 270.—Construction of sleeve.

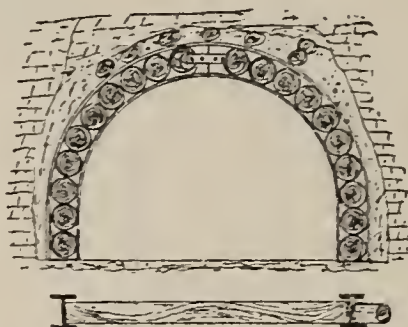


FIG. 271.

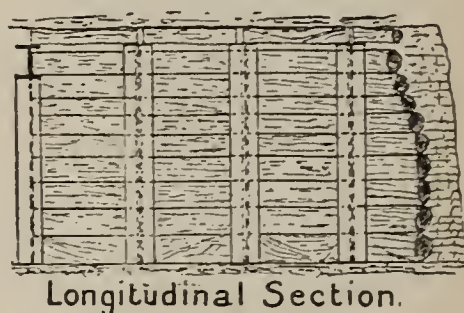
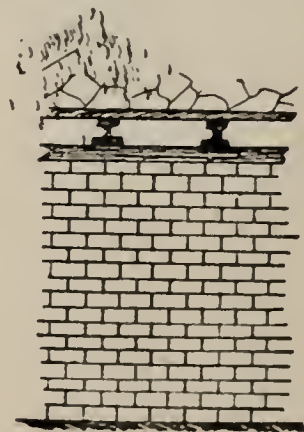
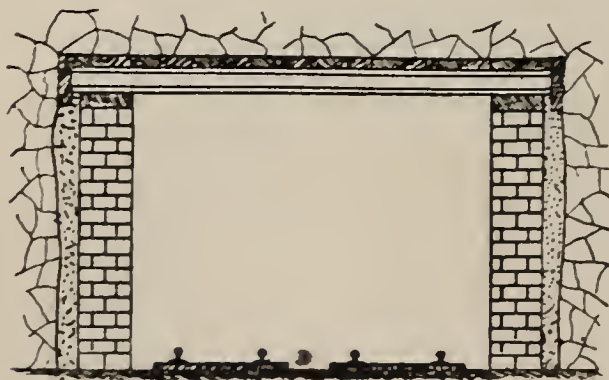


FIG. 272.

packed tight all round with ordinary prop-wood. Figs. 271, 272 illustrate this system, which stood the crush where it was found impossible to maintain brick arching.

Brick Walls and Girders.—This is a system that is very largely employed for supporting haulage roadways in preference to constructing brick arching. Side-walls of brick, 14 in. to 24 in. thick, are



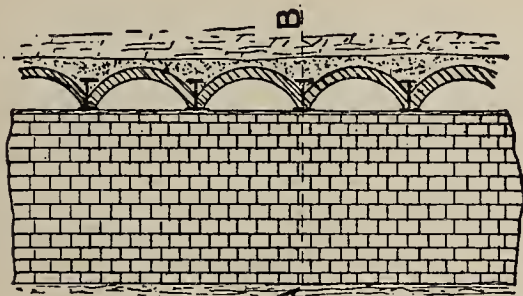
FIGS. 273 and 274.—Brickwork and girders.

built up on each side of the road, and steel H-girders stretched horizontally across them (figs. 273, 274). Along the top of the wall is laid wood planking, 4 in. $\times$ 12 in., on which the girders rest. The wood helps to relieve any sudden pressure to which they may be subjected. The girders should be wedged tight, and a runner or strap of iron fixed between every two sets to prevent them from canting. On the top is placed a lagging of square or round timber laid close together, any space between being carefully packed. At Milnwood Colliery, Bellshill, the wood lagging was replaced by strips

of iron about 3 in. broad and $\frac{5}{8}$ or $\frac{3}{4}$ in. thick. This system is preferable to using wood lagging, as the latter decays and requires frequent renewal. Sometimes sheets of iron, about $\frac{1}{8}$ in. thick, are stretched on the top of the girders, and wood lagging put above that; the sheet iron preventing the wood lagging from dropping on the road when it breaks or decays.

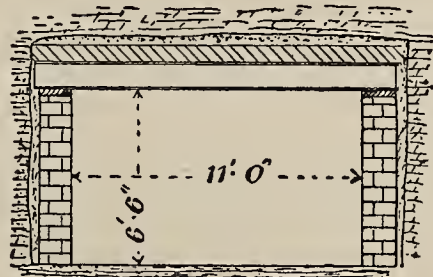
Corrugated sheet iron has also been used for the same purpose, and has given good results. Sheets of iron are best suited for dry roadways, for if there is water given off from the roof, especially water containing acids, chemical action soon occurs, with the result that the iron gets eaten into holes and becomes useless. In such circumstances wood lagging is to be preferred.

In another method of securing roadways by brick walls and girders, the latter, instead of being put in straight, are curved from the side of the wall on either side, which is said to increase their strength. It also gives increased height, but, of course, it is more



Longitudinal Section.

FIG. 275.



Cross Section on A.B.

FIG. 276.

expensive, as more rock requires to be excavated in the roof. The ends of the girders are laid on sheet iron, thus distributing the pressure over a greater area. The cost of this method is £13, 9s. 6d. per yard. The calculated cost for arching the same roadway was £16, 8s. 11d. per yard.

Another system which the writer has seen for supporting haulage roads with double lines of rails is to construct small arches between the rails or girders at right angles to the axis of the roadway (figs. 275, 276). The rails in this instance were put in at 2 ft. centres, and weighed 80 lbs. per yard, the small arches being $4\frac{1}{2}$ and 9 in. thick. Above these small arches the open spaces were packed up to the strata with concrete. This made an excellent lining, and gave good results.

At Lanemark Colliery, New Cumnock, some rather ingenious methods are adopted for supporting the roadways. On each side of the road side walls are built from the rubbish got from the workings, cement mixed with sand, in the proportion of one to four, being used to bind the rubbish together. At intervals along the road brick pillars are built up to the same height as the stone building. Along the top of the side walls planks are laid, and on the top of these

light steel girders are stretched across the road to support the roof. Figs. 277, 278 show this method.

Where the roof is fairly good and does not require any cross-girders, the roadways are secured as shown in figs. 279, 280. In

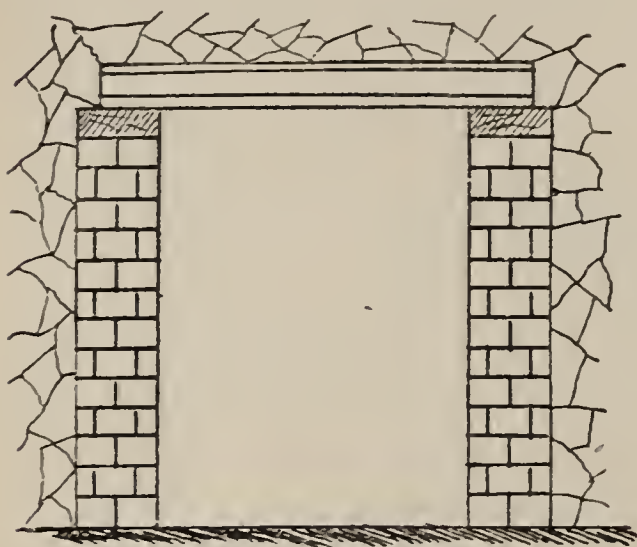


FIG. 277.—Cross-section.

this method the stone and cement walling is carried up to within 4 in. of the roof. In the space thus left, pieces of wood are placed lengthways, being wedged tightly to the roof. Between the stone and cement pillars a pillar of brick is built half-way up, and on the top of this a short prop is fixed between the pillar and the longitudinal planking.

Another variation in this system of supporting roadways is shown in figs. 281, 282, in which a continuous wall of stone

and cement is built up to about three-fourths the height of the road, and on the top of this building short pillars of brick and short props are carried up to support the wood. In both these latter methods no cross supports are used, the roof not requiring them. In other

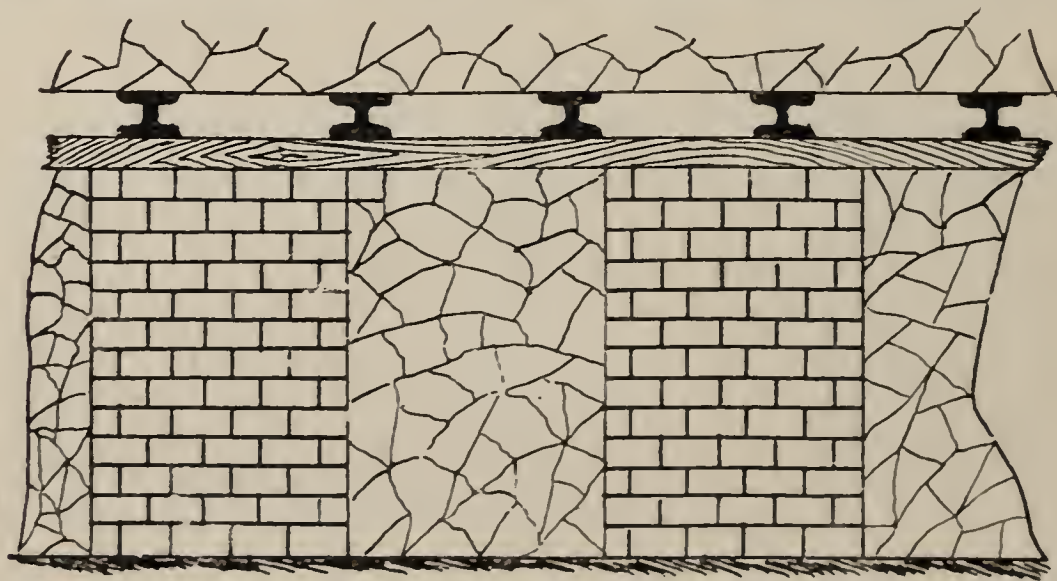


FIG. 278.—Longitudinal section.

parts of the workings the walls are entirely built of rubbish got from the working and cemented.

This kind of wall is found to act better in many respects than brick and lime. It often deflects to a considerable extent before giving way. The advantages claimed for such methods of supporting roadways are that only a very small number of bricks are required,

compared with those in which masonry is used throughout, while walls of this description can be built very cheaply, ordinary workmen being able, without difficulty, to erect them. In pillar and stall working they also prevent slabs of coal from breaking off the pillars and falling on to the roadways—an occurrence which often causes much inconvenience in workings, particularly on main haulage roads.

The advantages of using brick wall and girders for supporting the roadways, instead of brick arching alone, may be stated as follows:—

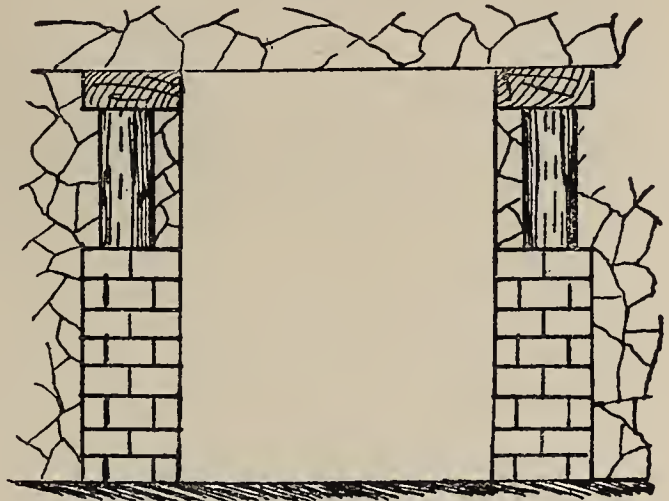


FIG. 279.—Cross-section.

Less space requires to be excavated for a given area, the saving in this respect being nearly 25 per cent.

Less time is required for erection, and hence less cost is incurred for labour.

Where the strata are soft, girders can be placed as the work proceeds, while with brick arching temporary supports would have to be used, thus increasing the cost.

Girders can be easily removed from one part of a mine to another and be used over again, whereas brickwork can seldom be removed.

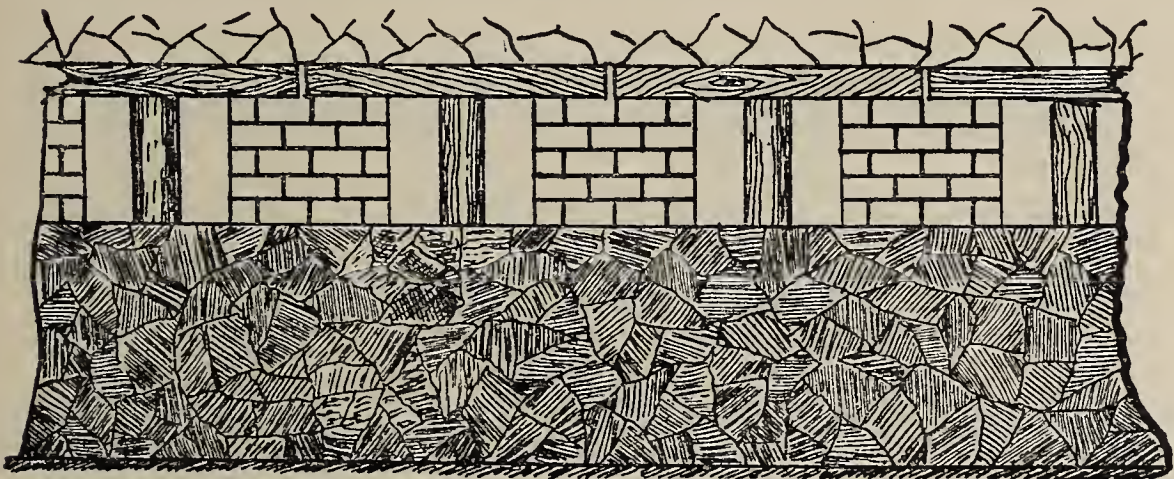


FIG. 280.—Longitudinal section.

The cost of iron or steel girders varies, and will depend to a certain extent on the proximity or otherwise of the colliery to iron-works. In 1904 the cost of girders was about £5, 10s. to £6, 10s. per ton, and for the various sections in use, which are about 50, 66, and 72 lbs. per yard, about 9d., 1s., and 1s. 1d. per foot respectively.*

* *N.B.*—The price of girders varies from time to time according to the market price of iron and steel; at the present time, 1912, the price is £7 to £7, 10s. per ton.

Comparing the cost of supporting roadways with girders with the cost of timber for the same purpose, the first cost for girders will be, approximately, twice as expensive; but, on the other hand, they will last four to six times longer than the best wood, and will, as a rule, give a greater margin of safety.

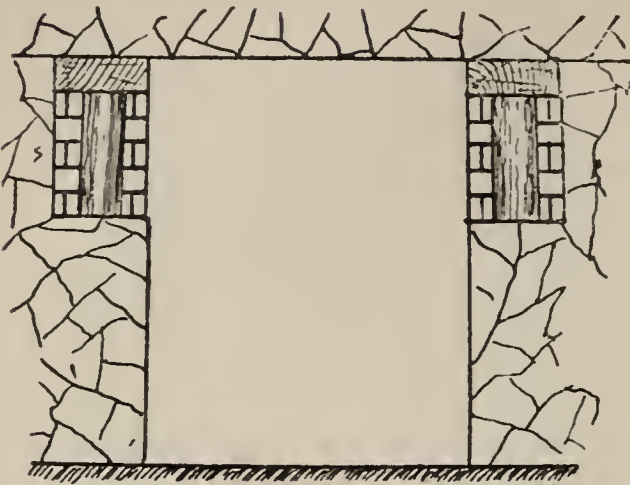


FIG. 281.—Cross-section.

Owing to the varying conditions in different mines, it is impossible to fix any definite weight or size of girder as being suitable for a given span. Under comparatively equal loads,* however, the weights, dimensions, and safe loads for 8-ft. space girders are shown in the following table:—

Depth of Girder.	Width of Flange.	Thickness of Web.	Weight per Foot.	Estimated Safe Dead Distributed Load for 8 Feet Span.
Inches.	Inches.	Inches.	Pounds.	Tons.
5	4	$\frac{3}{8}$	16 $\frac{3}{8}$	7
5	4	$\frac{1}{2}$	22	9
6	4 $\frac{1}{2}$	$\frac{1}{2}$	24	12

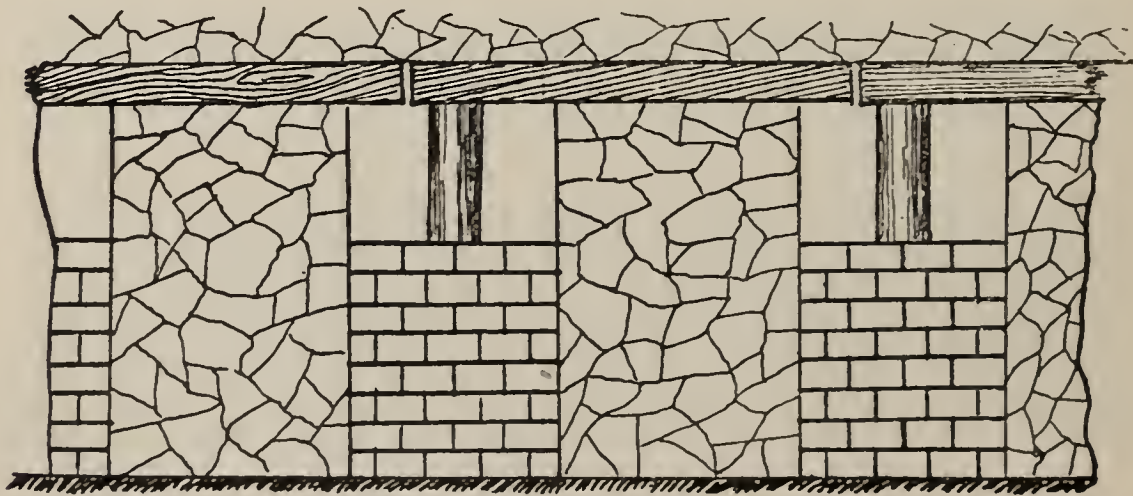


FIG. 282.—Longitudinal section.

The following weights of girders are often used for different spans:—

Girders of 16 $\frac{3}{8}$ lbs. per lineal foot, in spans of 6 to 8 ft.					
„	22	„	„	„	8 to 10 „
„	24	„	„	„	10 to 12 „

The above can, however, only be taken as approximate sizes, and it would be best to err, if anything, on the safe side. In a colliery

* *Trans. Inst. Min. Engs.*, vol. x. p. 274.

where the span was 16 ft., the girders used weighed 42 lbs. per lineal foot and were none too heavy.

Some of the advantages claimed for iron and steel props over timber are—

They are lighter and handier to work with than heavy wooden beams.

Girders give increased space for ventilation compared with timber.

There is no pollution of the air as is the case with decaying timber.

There is no risk of catching fire ; which is so often the cause of underground fires where the timber is in a dry condition.

Brick Arching.—For main roads and pit bottoms masonry is used to a very large extent, and probably no better method of securing a road has been devised. It is very much more expensive than timber, owing mainly to the extra cost for excavation, but where a roadway has to be used for a considerable number of years, it will in the end be the cheapest method. There are two principal methods of supporting roads with masonry, viz. by building perpendicular side walls to a certain height and then springing an arch for the roof, and by building the masonry all round the roof, floor, and sides, no part of the walling being built perpendicular, but each part having a certain curvature. This latter method is undoubtedly the better of the two, and masonry so constructed is very much stronger, but it is also a great deal more expensive, and unless the road has to stand for many years, and a large output is expected from it, it is questionable if the expense of such a system of arching would be justified. For most ordinary purposes, unless the floor is very bad and given to creep, the common method of supporting roads by two short perpendicular walls and a top arch will be found quite efficient.

The brick walling is put in 9 in. to 18 in. or 24 in. thick, and a space should be left between the walling and the strata, which should afterwards be filled in with fine ashes or sand, as this will greatly assist the arching when the pressure comes on it. The method of arching underground roadways is carried out as follows:—Centering pieces are constructed of timbers (*a, b, c*, fig. 283) to suit the circle of the arch to be built. These centres are then placed on short brick pillars *d d*, 12 to 15 in. square, built up without lime at the sides of the roadway, a piece of planking being laid along the top from pillar to pillar, on which the centres rest. Before these pillars and the centering are set in position the main side walls are built up perpendicular to the spring of the arch. After this is completed the centres are set up 5 to 6 ft. apart, and along the top of them a lagging of planking *e* is fixed longitudinally. To keep the structure in position other pieces are placed, also longitudinally, between the centres. When this has been done, and everything made secure, the main part of the brick arch *f* is then proceeded with, the space between the arching and the roof being tightly packed with débris or ashes.

Strength of Timber.—The strength of timber is not always easily determined, and no definite rules can be given as to the size of props

or crown-trees to be used underground. The circumstances prevailing in each colliery as to roof, floor, and sides, combined with everyday experience in practical working, seem about the safest and best guides to depend upon.

Props set in the workings may be said to break in three different ways, viz.: (a) by fracture or 'buckling' alone; (b) by buckling and crushing combined; (c) by crushing alone. In the first case, props generally give way when their length is from twenty to thirty times their diameter, in the second when their length is from ten to twenty diameters, and in the third case when the length is under ten diameters.

From numerous experiments on the strength of pillars (or props) of timber, the following law has been deduced:—

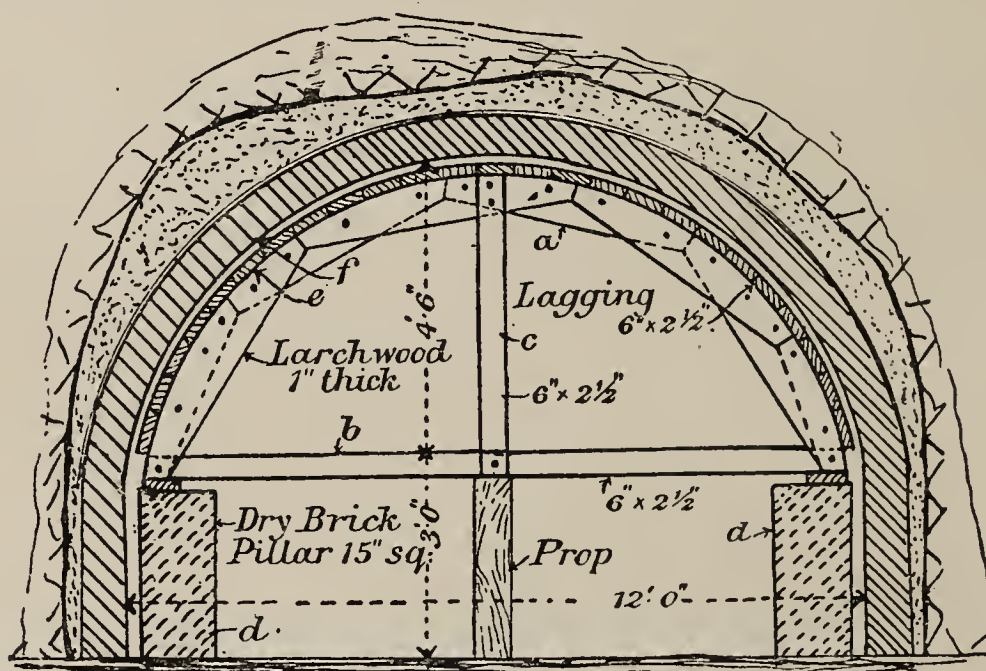


FIG. 283.—Showing method of building brick arching.

“The strength of pillars of timber of equal sectional areas is inversely proportional to the square of the length.”

Thus, with lengths in the ratio of 2, 4, and 8 ft. the strength will be in the ratio of $(\frac{1}{2})^2 : (\frac{1}{4})^2 : (\frac{1}{8})^2$, or $\frac{1}{4}$, $\frac{1}{16}$, and $\frac{1}{64}$.

Taking pillars of the same sectional area, one 2 ft. in length has sixteen times the strength of one 8 ft. in length and four times that of one 4 ft. in length. This is well known in everyday practice, and the props are usually increased in diameter, according as the height of the working increases. For ordinary large props the crushing strain is about $1\frac{1}{2}$ to 2 tons per sq. in., according to the age of the wood and the seasoning it has undergone.*

If timber be cut when green, and allowed to season or dry gradually, it is found to gain in durability, as was proved by the experiments carried out by Professor Louis,† who records a gain of as much as 49 per cent. in the strength due to seasoning in ordinary pit props.

* *Trans I. M. E.*, vol. xv. p. 352.

† *Ibid.*, p. 354.

This fact is fully recognised on the Continent and at many collieries in France and Germany; the props are thoroughly seasoned in specially-constructed drying sheds before they are used underground; and in some cases they are seasoned artificially by an electric process which is said to give good results.

In a number of tests which were recently carried out by the Government colliery officials in the Saar district,* the same results were obtained. The following table shows the effect of seasoning, as ascertained experimentally, on four different kinds of wood:—

Kind of Timber.	Wood shortly after Felling.		Wood Five Months after Felling.		Props artificially dried in a Temperature of 149° Fahr.	
	Resistance to Compression. Lbs. per Sq. Inch.	Specific Gravity.	Resistance to Compression. Lbs. per Sq. Inch.	Specific Gravity.	Resistance to Compression. Lbs. per Sq. Inch.	Specific Gravity.
Beech, with bark .	3243	1084	3570	1094	3627	0915
Fir do. .	2802	0885	3044	0845	3385	0656
Pine do. .	2631	0984	2716	0917	2958	0647
Oak do. .	2475	1235	2133	1050	2958	0825

It must not be forgotten that the different specimens tested were special pieces, free from blemishes, and having little resemblance to the ordinary pit prop.

The following tables† give the crushing and tensile strains of various kinds of wood:—

STRENGTH OF TIMBER TO RESIST CRUSHING-STRAINS IN POUNDS AND TONS PER SQUARE INCH.

Kind of Timber.	Maximum Dry.	Minimum Ordinary State.	Mean.	
	Pounds.	Pounds.	Pounds.	Tons.
Ash	9363	8683	9023	4·03
Beech	9363	7733	8548	3·81
Birch (English)	6402	3297	4850	2·16
Elm	10331	7950	9140	4·08
Fir (spruce)	6819	6499	6659	2·97
Oak (English)	10058	6484	8271	3·69
„ (Quebec)	5982	4231	5106	2·28
Pine (pitch)	6790	6790	6790	3·03
„ (red)	7518	5395	6457	2·88
Larch	5568	3201	4385	1·96

* *Glückauf*, Berginspektor Ch. Düthing, 1898, vol. xxxiv. p. 797.

† *Treatise on the Strength of Materials*, by Mr Thomas Box, 1883, p. 91.

SHOWING STRENGTH OF TIMBER TO RESIST TENSILE STRAIN IN POUNDS AND TONS PER SQUARE INCH.

Kind of Timber.	Maximum.	Minimum.	Mean.	
	Pounds.	Pounds.	Pounds.	Tons.
Ash	17,850	15,784	17,077	7·6
Beech	11,326	11,388	11,467	5·1
Birch	...	...	15,000	6·7
Elm	...	...	13,490	6·0
Fir	13,448	11,000	12,203	5·5
Oak (English) . .	15,500	13,620	14,560	6·5
Pine (Russia) . .	...	...	13,300	5·9
,, (Norway) . .	14,300	12,400	13,350	6·0
Larch	...	...	9,632	4·3

SHOWING SPECIFIC GRAVITY AND WEIGHT OF MATERIALS (WATER AT 62° FAHR. BEING EQUAL TO UNITY).

Material.	Specific Gravity.	Weight of	Measurement
		1 Cubic Foot.	of 1 Ton.
		Pounds.	Cubic Feet.
Wrought iron	7·788	485·30	4·615
Cast iron (British) . .	7·087	441·60	5·070
Oak (seasoned)	0·777	48·42	46·260
Elm	0·588	36·65	61·130
Pine (yellow), seasoned . .	0·483	30·10	74·410

The following rules apply to bars or beams of timber :—

- 1. The strength of bars or crowns of the same sectional areas is in direct proportion to their width. Thus a bar 12 in. wide is twice as strong as one 6 in. wide if both have the same thickness.
- 2. The strength of rectangular beams of the same length and width is directly proportional to the square of their depth ($W \propto d^2$); thus if two bars are of equal width, but one is 6 in. deep and the other 3 in. deep, their strength will be in the proportion of $3^2 : 6^2$ or $9 : 36$ or $1 : 4$, i.e. the prop 6 in. deep will have four times the strength of one 3 in. deep.
- 3. The strength of bars of equal sectional area varies inversely as their lengths ($W \propto \frac{1}{L}$). Thus a bar 12 ft. long will only have half the strength of one 6 ft. long, the sectional area being equal in both.

Formulae for Strength of Beams of Timber.—Different formulæ are given by different authorities for finding the sizes and breaking weights of beams of timber, all of them giving slightly different results. But it must be remembered that most of the formulæ applied in engineering give only approximate results, and are not meant to be absolutely correct, as is the case with the formulæ employed in problems in pure mathematics. In engineering, materials for construction are generally allowed a large marginal factor of

safety, and there is, therefore, not the same necessity for very fine theoretical calculations.

Let L = length of beam or span in inches
 B = breadth
 D = depth
 W = breaking load in tons
 K = coefficient of rupture

The value of K for different materials has been found by experiment, and is given below :—

Wrought iron = 3.40	Beech = 0.65
Cast iron = 2.30	Fir = 0.60
Ash = 0.95	Oak = 0.75
Pitch Pine = 0.75	Larch = 0.75

(1) Both ends *supported* and beam loaded in the centre. $W = 4K \frac{BD^2}{L}$.

(2) Both ends *fixed* and load in the centre. $W = 6K \frac{BD^2}{L}$.

(3) Both ends *supported* and load evenly distributed. $W = 8K \frac{BD^2}{L}$.

(4) Both ends *fixed* and load evenly distributed. $W = 12K \frac{BD^2}{L}$.

In circular beams of radius R , substitute $4.7R^3$ for BD^2 in the above formulæ. These rules are very difficult of application to mine timber, as the load is very rarely either at the centre of the beam or evenly distributed along its length, and there is also the side pressure to take into consideration, which, in underground timbering, is often greater than that from the top, and can never be accurately measured. The pressure per square inch due to the weight of the overlying strata alone would be equal to $D \times .434 \times B$; where D = depth or thickness of overlying strata, B = specific gravity of strata, and $.434$ = a constant number (average specific gravity of strata is about 2.5, taking water as unity or 1).

Strength of steel or wrought-iron girders of H-section :—

When W = breaking load in tons,

A = area of one flange (either top or bottom) in square inches,

f = tensile strength of material in tons per square inch (generally from 22 to 28 for steel and 18 to 20 for wrought iron),

D = depth of girder in inches (including both flanges),

L = length of span in inches.

Then $W = 4f \frac{AD}{L}$.

The safe load is generally taken at $\frac{1}{5}$ th to $\frac{1}{7}$ th of the breaking load for steel girders.

EXAMPLES.

(1) Find the breaking weight at centre of a pitch pine beam, 12 in. deep, 8 in. broad, and 18 ft. between the supports, ends fixed and load in centre; also find

the depth of beam required for a breaking load of 24,000 lbs., if the width of beam is 6 in. and the distance between the supports 12 ft., load uniformly distributed.

Suppose the beam to be fixed at both ends, then

$$(a) \quad W = 6K \frac{BD^2}{L}$$

$$= 6 \times .75 \frac{8 \times 12^2}{18 \times 12}$$

$$= \frac{6 \times .75 \times 8 \times 144}{18 \times 12}$$

$$\therefore W = 24 \text{ tons}$$

$$(b) \quad W = 12K \frac{BD^2}{L}$$

$$\frac{24000}{2240} = 12 \times .75 \frac{6 \times D^2}{12 \times 12}$$

$$\frac{24000}{2240} = \frac{12 \times .75 \times 6 \times D^2}{12 \times 12}$$

$$\therefore 112 \times 1 \times 0.75 \times 1 \times D^2 = 2400 \times 1 \times 1.$$

The breaking strain is, therefore, 24 tons.

$$\therefore D^2 = \frac{.2400 \times 1}{112 \times .75} \quad \therefore D = \sqrt{28.57} = 5.3 \text{ in.}$$

The depth of beam required would be 5.3 in.

(2) If a beam 10 in. broad and 14 in. deep has a breaking strain of 30 tons, what length of span would it support under a maximum load, supposing the beam to be simply supported and the load to be in the centre?

$$\text{Here } W = 4K \frac{BD^2}{L},$$

$$\therefore 30 = 4 \times .75 \frac{10 \times 14^2}{L}, \text{ and } 30 L = 4 \times .75 \times 10 \times 196.$$

$$\therefore L = \frac{4 \times .75 \times 10 \times 196}{30} = 196 \text{ in. or } 16.3 \text{ ft.}$$

(3) What would be the breaking load of a wrought-iron girder of H-section, with top flange $4\frac{1}{2}$ in. broad, depth of girder 6 in., and span between supports 10 ft.?

Taking the tensile strength of wrought iron at 20 tons,

$$W = 4f \frac{AD}{L} = 4 \times 20 \frac{4.5 \times 4.5 \times 6}{10 \times 12} = 81 \text{ tons.}$$

Preservation of Timber.—Timber required for use underground, or, indeed, anywhere, should be felled during the winter when it has but little sap in it, because sap in wood ferments and produces rapid decay. It should also be well seasoned before being used, and if these two points are carefully attended to they are frequently all the timber requires to preserve it. The bark should also be removed before the timber is used underground; if this is done, there is less liability to decay, and when this does set in it is more easily detected.

Various methods of preventing dry rot have been tried. Good ventilation is necessary, as timber decays much faster in foul, hot air than in a pure atmosphere. Water is also a good preservative, and in some places the shaft timber is kept wet for this purpose. The water acts by washing off the spores of the fungi as fast as they are formed.

The various methods of preserving timber :—

By common salt dissolved in water.

By impregnation with metallic salts, such as sulphates of copper and iron, chlorides of zinc or magnesium, etc.

By the use of creosote.

By coating with tar, etc.

Timber is often treated with brine made with common salt, in the proportion of 1 lb. of salt to four or five gallons of water, the timber being allowed to get thoroughly soaked with the solution. It has the advantage of being cheap and easily applied. Sulphate of iron is also much used, and has the recommendation of being effective and economical.

Chlorides of magnesium and zinc are used for preserving timber. In the zinc process, a solution of chloride of zinc is forced under pressure into the timber. The solution consists of one part of liquid chloride of zinc (specific gravity of 1.5) mixed with 35 gallons of water. One gallon of this solution weighs 15 lbs. and contains about $3\frac{1}{2}$ lbs. of metallic zinc. This process is said to make the wood firm, hard, and proof against the attacks of insects and dry rot.

Aitken Process.—In this process the timber is soaked in boiling water containing a strong solution of common salt and chloride of magnesium. The proportion of common salt to the latter should be 7 to 1, and a certain proportion of undissolved salt requires to be kept at the bottom of the tank used for steeping. The timber treated should be free from bark, well seasoned, and thoroughly dry. The plant used at the Niddrie Collieries, near Edinburgh, where this system is in operation, consists of two rectangular iron tanks made of $\frac{1}{2}$ -in. boiler plate, 19 ft. long, 4 ft. wide, and 3 ft. deep, built into a brick seating with a hearth beneath. The boilers are fired with dross, and the tanks have a covering of loose boards.

The props are boiled for forty-eight hours; pitch pine and larch require longer treatment than softer woods. When the timber is removed from the tanks, it is stacked in a covered shed with free access of air, to dry, as it is quite soft and not fit for immediate use.

Cost of using Preparation.—With the above plant 15 tons of timber can be treated weekly at a cost of £2, 12s. 8d., or about 3s. 6d. per ton, which represents about 1s. 5d. per 100 ft. of 6-in. diameter prop wood. The plant itself costs about £100.

The process is said to make the timber brittle, and when it is used as 'sleepers' on roadways the nails do not hold well, owing to the oxidation occasioned by the salts present. To overcome this difficulty galvanised nails should be used. This method of treating timber is employed at a number of collieries in Scotland, among which are the Cadzow Collieries, Hamilton; Auchinraith Colliery, Blantyre; the Leven and Lochore Collieries, Fife; and others.

Creosote Methods.—Impregnating timber with crude creosote, which was first tried in 1842, is one of the best methods of preserving

timber, but such timber has the great disadvantage, particularly for mine work, of being very readily ignited, and is, therefore, less suitable for underground work than for other purposes. For railway sleepers at the surface, and even underground, where no danger of fire exists, creosoting adds greatly to the 'life' of the wood.

The effects of creosote are threefold: (1) it fills the pores and prevents saturation with water; (2) it destroys organic life; (3) the carbonic acid coagulates the albuminoids present in the wood and prevents decay.

Coal Tar.—Painting the timber with liquid tar is sometimes resorted to, but this also confers the disadvantage of being easily ignited.

Painting the props with ordinary whitewash is also a plan adopted, and one which gives fairly good results. While preservatives undoubtedly prolong the life of timber in underground workings, they seem at the same time to decrease its strength to a considerable extent. Professor Louis has made a number of experiments,* which show that timber thoroughly creosoted was diminished in strength to the extent of 8·5 per cent., while woods treated by the Aitken process were weakened from 8 to 20 per cent., according to the kind of timber treated.

The following table† shows the results of tests made at Saint Eloy, on the relative duration of differently preserved woods (unpreserved wood = 30) in France, upon different methods of treating oak, fir, pine, beech, birch, and poplar woods. Two specimens out of every ten experimented on were used in the natural state. The others were treated with solutions of (1) tar, (2) chloride of zinc, (3) sulphate of copper, (4) sulphate of iron, and (5) creosote, respectively.

Name of Preservative.	Name of Wood.					
	Oak.	Fir.	Pine.	Beech.	Birch.	Poplar.
Tar, ...	28·7	263·5	87·5	105·4	26·2	150·5
Chloride of zinc, ...	10·5	50·0	26·3	18·6	52·5	34·7
Sulphate of copper, ...	42·1	12·0	8·0	1·8	2·5	15·5
Sulphate of iron, ...	18·0	12·5	4·2	4·7	3·7	2·9
Creosote, ...	1·7	2·5	4·4	0·6	3·3	1·3

Solutions of molasses have also been used successfully on the Continent and elsewhere.

* *Trans. Inst. M. E.*, vol. xv. p. 352.

† *Comptes-rendus mensuel des Réunions de la Société de l'Industrie Minérale*, 1890, p. 225.

NOTE.—The author is indebted to the proprietors of the *Colliery Guardian*, London, for liberty to reproduce figs. 246–254 in this chapter from their "Report of the Prussian Commission on Falls of Stone and Coal."

CHAPTER X.

WINDING COAL.

Preliminary.—The operation of winding or raising the coal from the underground workings to the surface is one of the most important parts of the daily work of a colliery, for, in many cases, the output is limited only by the means available for raising the coal. When once the winding machinery is erected, it is clear that whatever the demands may become, the quantity of coal raised per day cannot exceed the capabilities of the machinery or the winding power. It follows, therefore, that what may be termed increased cost in the winding gear is of very small importance, when compared to the great advantage that may accrue from having, what may appear at the time superfluous, power which can be employed in case of need and if the output is capable of extension. All other surface arrangements must be subsidiary to the necessity of dealing effectively with the coal when drawn, otherwise much vexatious expense and delay will be entailed.

Pit-head Frames.—Pit-head frames were at one time almost entirely constructed of wood, but of recent years wrought iron and steel have been extensively used in their construction. Where a frame has to stand for thirty to fifty years, or possibly longer, it is a matter of economy to adopt iron or steel structures, as they are more stable and are not liable to decay like wood frames. For high frames, and for the heavy loads now raised at modern collieries, it is almost imperative to build the frames of steel. By the provisions of the Coal Mines Act, 1911 (69 (2)), "Any mine newly opened after the passing of the Act, not being a small mine, no inflammable material likely to cause danger from fire to persons employed below ground shall be used in the construction of the pit-head frame, or of the roof, if any, over the pit-head." This provision seems to provide that pit-head frames erected in future must be of steel or iron, unless for mines in which less than thirty persons are employed. In cases where timber is employed pitch pine is generally selected, the size of the wood depending upon the height of the frame and the load to be raised. The following sizes are often used in practice :

—for frames 20 to 30 ft. high, front stays and main supports 10 to 12 in. square; for frames 30 to 40 ft. high, front stays and main

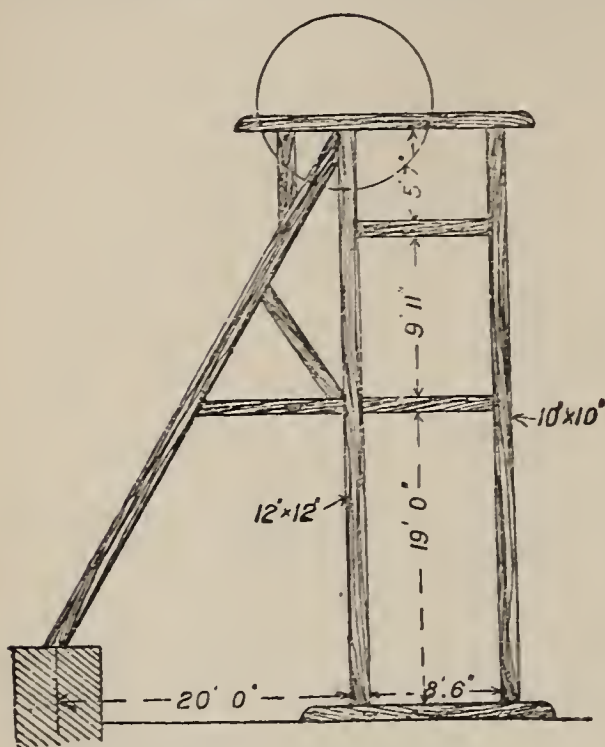


FIG. 284.—Side elevation.

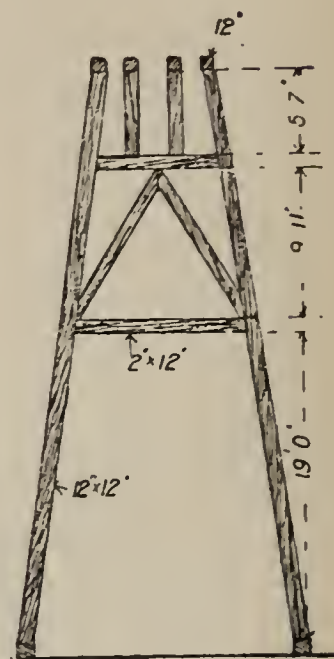


FIG. 285.—Front stays.

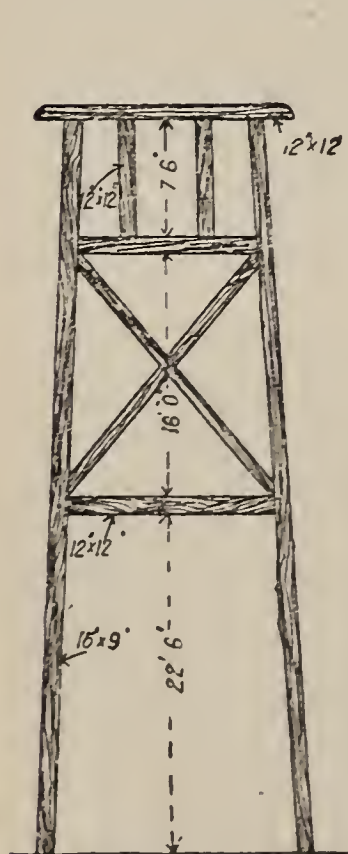


FIG. 286.—Back stays.

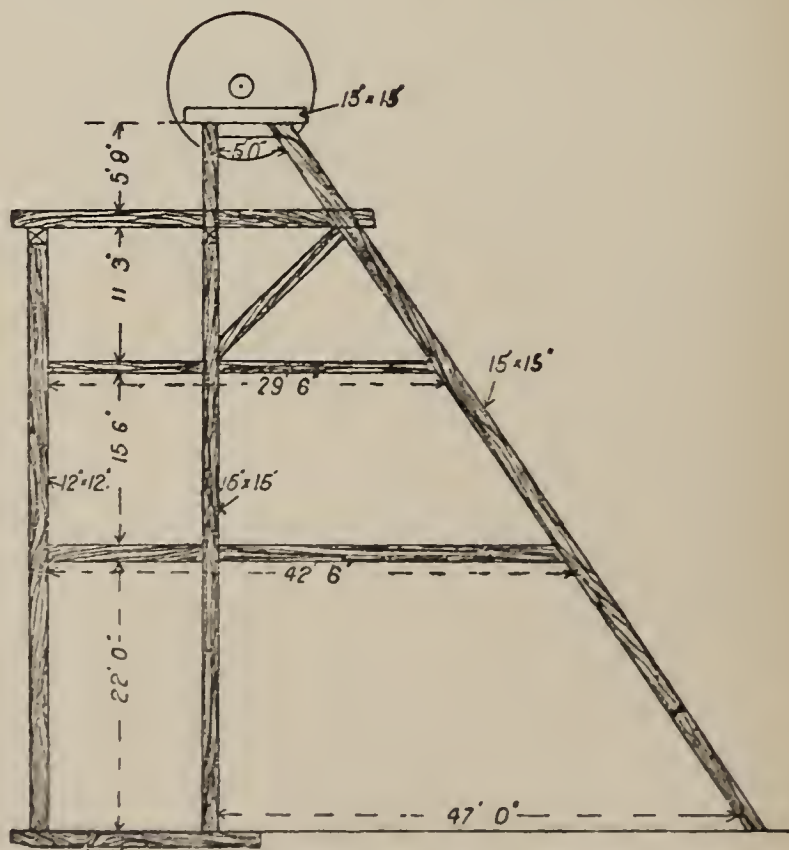


FIG. 287.—Side elevation.

supports 12 to 14 in. square; and for frames 40 to 60 ft. high, the whole of the wood would be 14 in. to 18 in. square.

Pit-head frames are usually of two kinds, single and double, both sorts being largely used, according to the preference of those erecting

them and the class of work for which they are designed. For heavy loads and where pumping is required and tackling has to be fixed

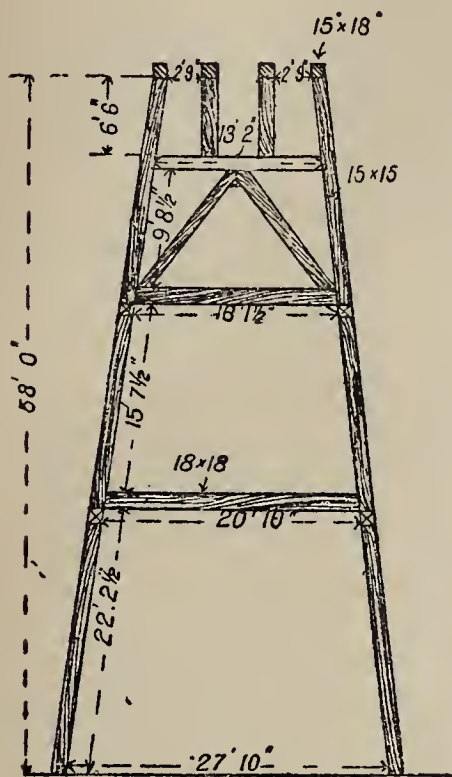


FIG. 288.—Front stays.

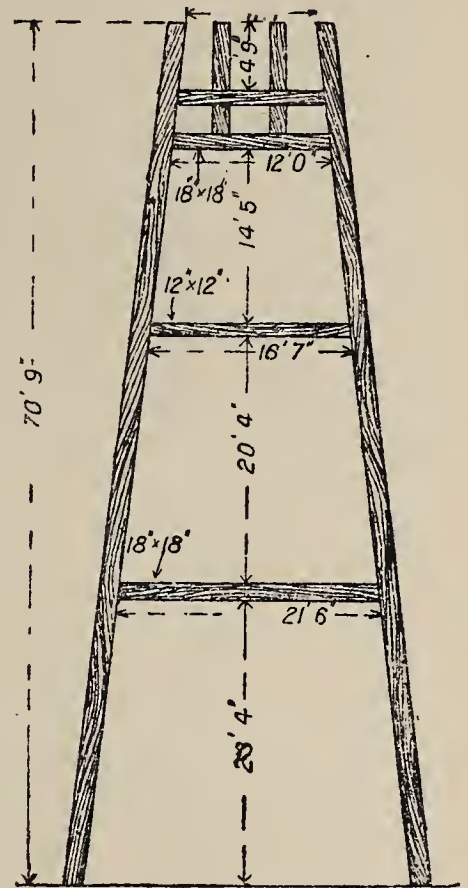
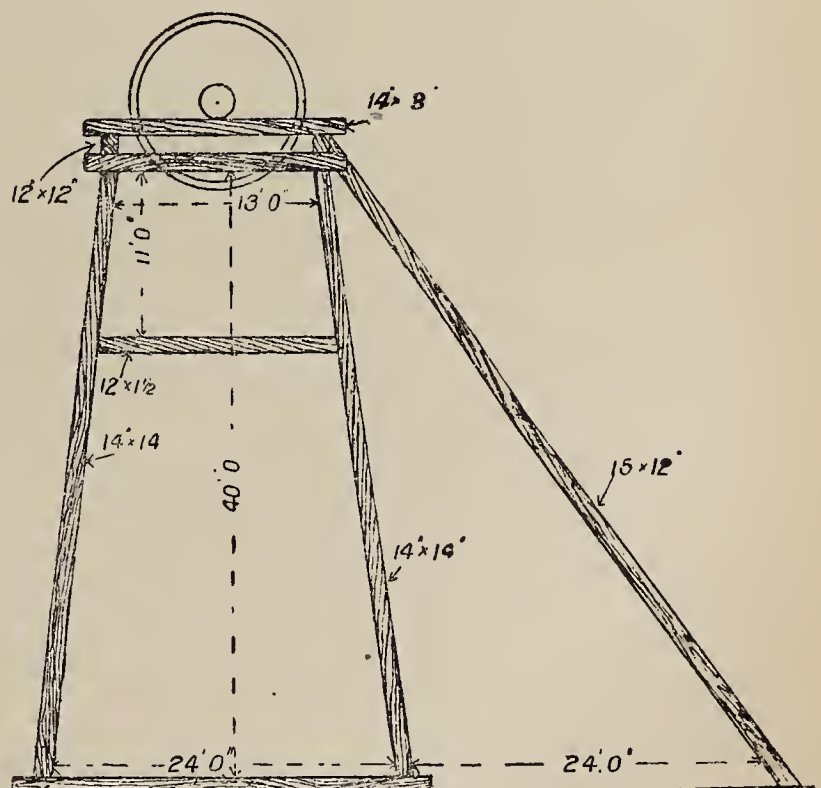
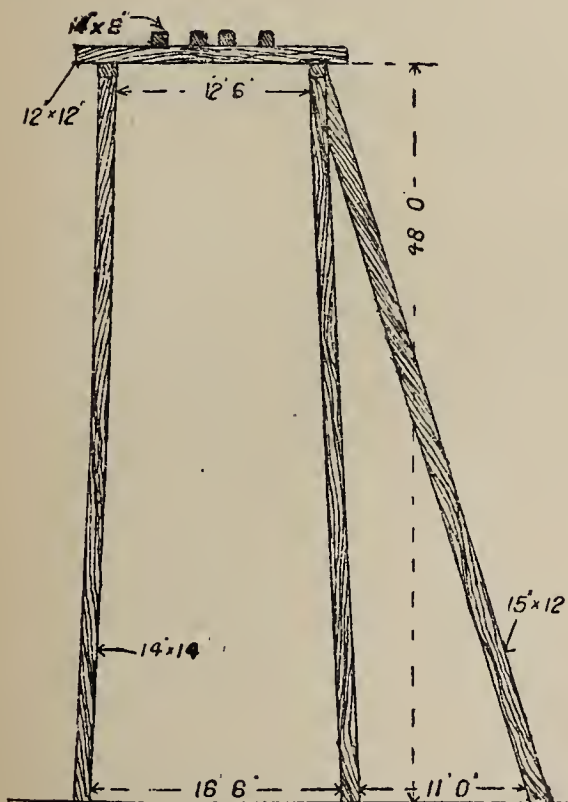


FIG. 289.—Back stays.



FIGS. 290 and 291.—Pit-frames and pit-head gear.

to the frame, the double type of frame is most suitable; a further advantage being that pulleys for haulage ropes can easily be erected

on them, so saving the erection of another frame close to the pit mouth where the room can often be ill spared. Where no pumping is necessary, a good single frame is, on the other hand, just as good for winding, is neater, more easily erected, and is less expensive than a double one.

Figs. 284–289 show two types of double wood frames largely used. Figs. 290, 291 show a type much in use at large collieries in Scotland, and known locally as a ‘table frame.’ Where much pumping is done, and block-and-tackle pulleys have often to be suspended from the frame, or, as is the practice at some collieries, for the pulleys for haulage ropes, as well as the winding pulleys, to be placed on the frame, then this type of table frame is possibly the best form that can be adopted.

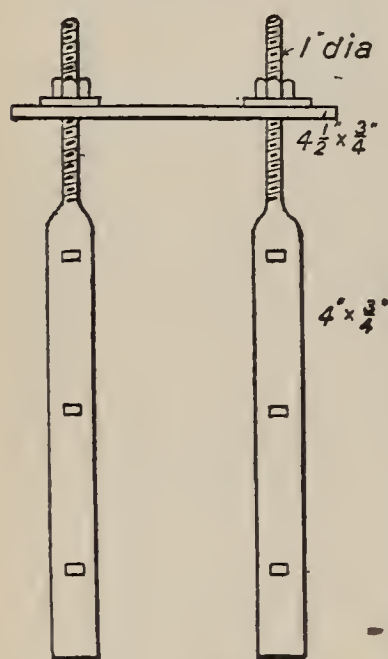


FIG. 292.—Manner of fixing glands.

The whole construction should be firmly and carefully put together by careful and experienced workmen, and the parts fitted together previous to its erection, so that every part fits exactly. The cross-stays should be morticed into the uprights, about 3 in. being generally allowed, all the parts being well bound to each other by good, strong wrought-iron glands and plates. Fig. 292 shows the details of these glands and the manner of fixing. The wood in the frame should be smoothly planed, and two or three good coats of paint should be applied to preserve it from the weather. It should also be repainted every second year at least; this will prevent decay setting in.

Generally, the back and front stays are fixed at the bottom on sole-pieces running across the front of the pit and at right angles to the end of the back stays. These sole-pieces should rest on a good foundation of brick or concrete above the surface of the ground, to prevent moist earth from coming in contact with the wood, which will help greatly to prevent decay. Probably the best way is, however, to omit these sole-pieces and to fix the ends of both back and front stays into cast-iron shoes which rest on, and are firmly bolted to, pillars of masonry or concrete. The seam at the top of the shoe should be well filled with putty to prevent water lodging, otherwise this method of fixing is of little advantage so far as the prevention of decay is concerned.

The position of the back stays in regard to those in front is a very important consideration, as it is on this part of the frame the tension due to the winding ropes exerts itself. The back stays ought to be put up with a fairly large angle, otherwise the frame is liable to be drawn over by the tension or pull on the ropes; on the other

hand, they ought not to be erected with too large an angle, otherwise their own weight will exert a pressure on the front stays and tend to push the latter out of position.

A good plan is to make the distance between the centre of the shaft and the foot of the back stays about equal in length to the height of the frame, or even longer; or else the distance should equal the height of the frame multiplied by the sine of the angle made by the ropes with the pulley and drum. Suppose the pit-head frame is 60 ft. high, and that the angle that the ropes make between the drum and pulley is 50° , then $60 \times \sin 50^\circ$ (0.7660) $= 45.96$ ft., the distance the back stay ought to be from the centre

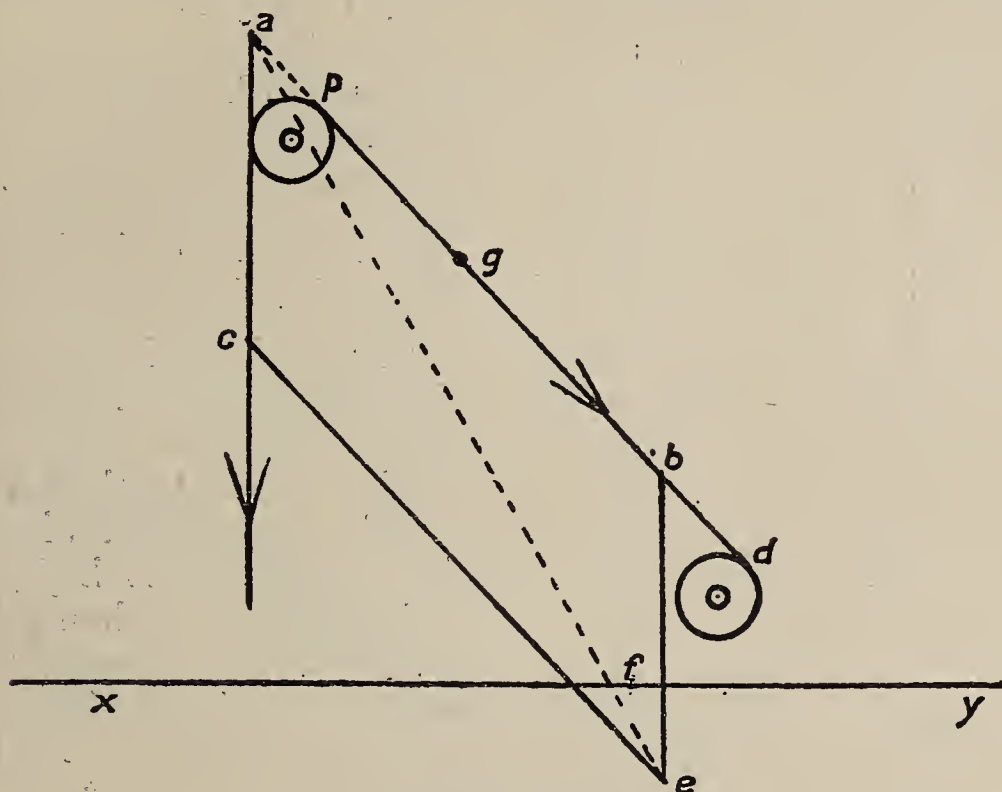
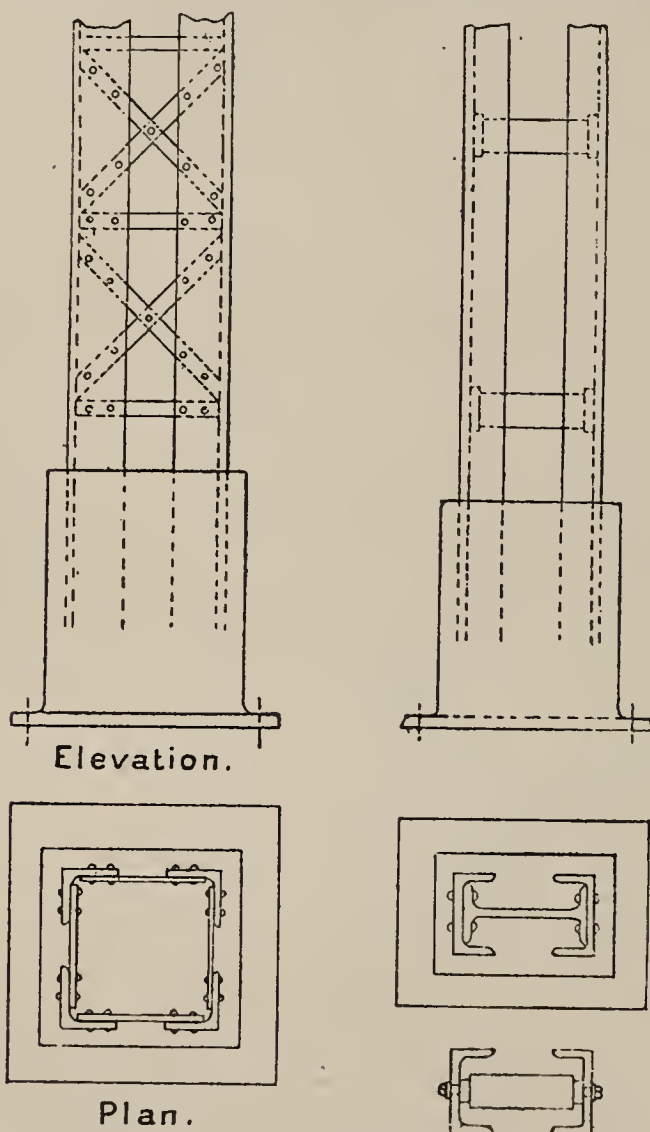


FIG. 293.

of the shaft. The position for the back stays may also be found graphically by employing the principle of the parallelogram of forces. Let xy be the ground line (fig. 293), d the position of the drum, and p the position of the pit-head pulley. Draw a line ad between these two, and another line ac representing the part of the rope hanging in the shaft to which the load is attached. Ascertain what the total load to be raised amounts to. Now, along the lines ad and ac , two forces which are equal and opposite to each other will be acting, the force along ad being that due to the pull of the engine required to raise the load, while the force along the line ac will be exerted by the load itself pulling in a downward direction. With any suitable scale, say 1 in. to represent 1 ton of load, lay off the distance ac equal to the total load, and along the line ad lay off the same distance ag ; but to allow for contingencies, such as undue strain due to an over-wind, wind pressure,

etc., it is better to make the distance ab along the line ad equal to twice ac . From the point b draw a line be parallel to ac , and another line ce parallel to ab ; a line joining the points ae represents the resultant of the forces ac and ab , and the point f where it cuts the ground line will be the position for the back stay.

Iron or Steel Frames.—As already stated, pit-head frames are often constructed of iron or steel. On the Continent the frames are sometimes made of tubular material, but this type of frame has



FIGS. 294, 295, and 296.

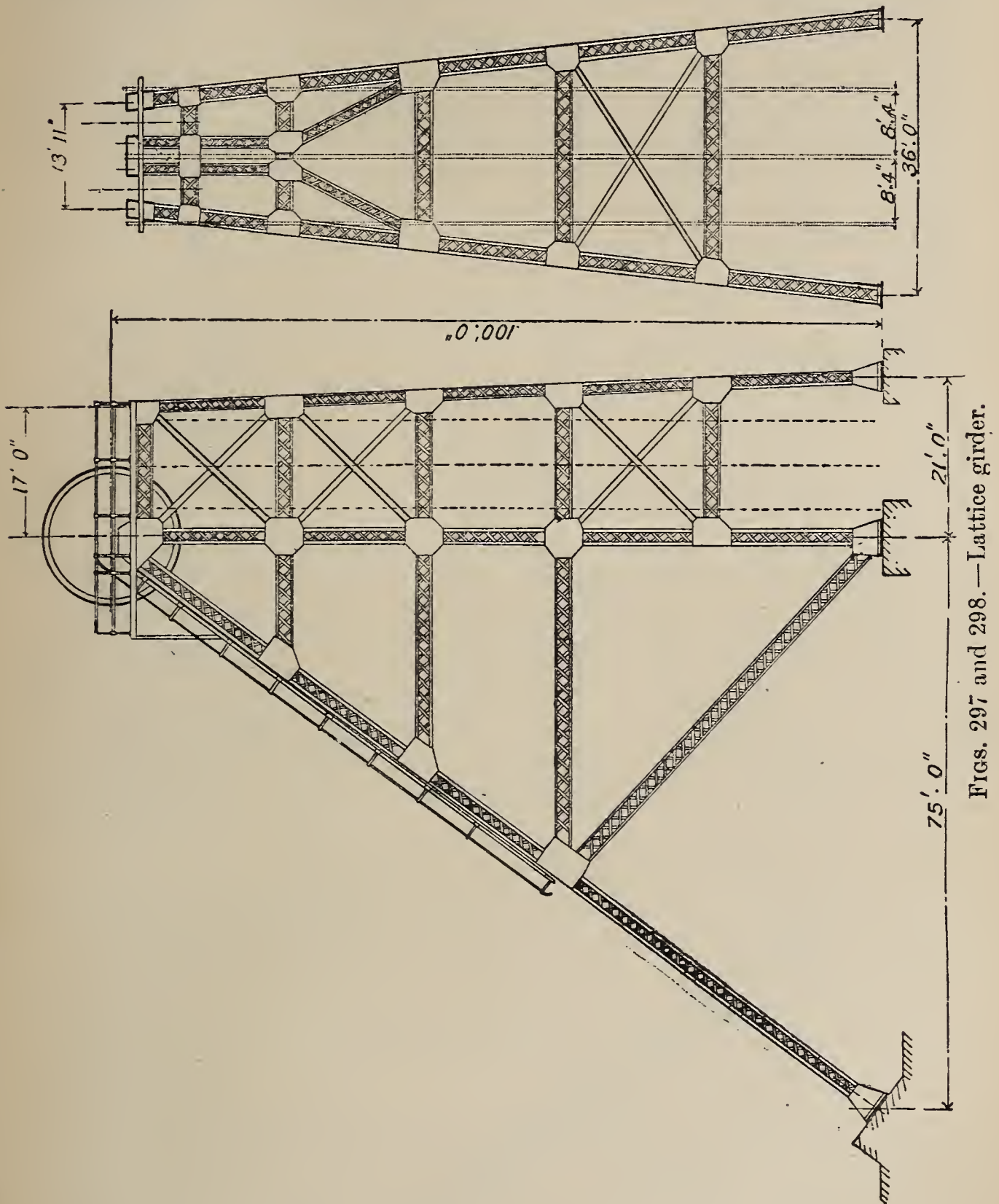
never come much into use in Britain. Those most generally employed are constructed either of (a) L-irons with binding pieces (fig. 294); (b) channel-sections with binding centre or distance pieces (fig. 295); or (c) H-sections with binders. For large frames the first method is frequently used, as it makes a strong structure and is easily built up. Figs. 297, 298 show the construction of a frame of this description.

Figs. 299, 300 show a frame mainly made up of H-sections and binders, 70 ft. high, which is less expensive than a lattice girder frame. The cost of such a frame would be about £350, including erection. At Palace Colliery and Bent Colliery, Hamilton, the principal parts of the frames are constructed of ordinary railway rails and lattice work on the back stays. Figs. 301–303A show this class of frame, which

makes both a neat and strong erection.

Height of Frame.—It is important that sufficient height be allowed so that the lower rim of the winding pulley will not be too close to the cage connections when the loaded cage is brought to the surface. The height between the drawing-off level at the pit-head and the pulleys will depend on the type of cage used; *i.e.* whether it has one, two, or three decks, and also the length of the chain attachments. When the cage is at rest on the keps a minimum distance of 12 to 15 ft. should be allowed between the top of the rope capping and the lower rim of the pulley. It would be better, however, to allow more than this, say a distance equal to the circumference of

the drum, between the drawing-off level and the pulley. Thus, if the drum was 18 ft. diameter, the distance from the pit-head level to the pulley would be $56\frac{1}{2}$ ft., and as the height between the rail level



Figs. 297 and 298. — Lattice girder.

on the sidings and the pit-head is 21 to 25 ft., this would require a pit-head frame approximately 77 to 81 ft. high. On large shafts the frames are frequently 90 to 100 ft. in height.

Winding Engines.—Winding engines may be divided into two classes, viz. : (1) Direct-acting coupled engines ; (2) non-direct-acting

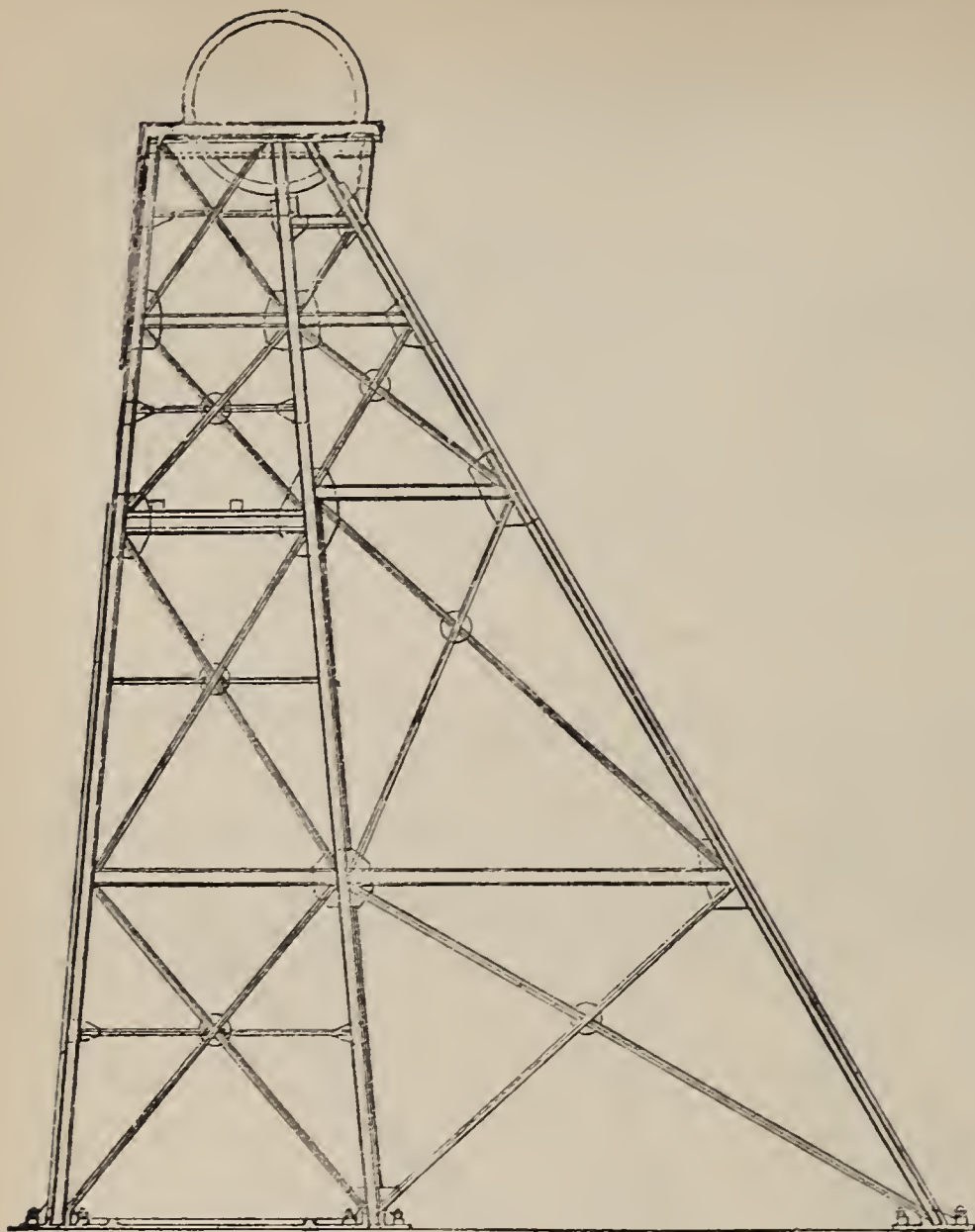


FIG. 299.—Steel frame (side elevation).

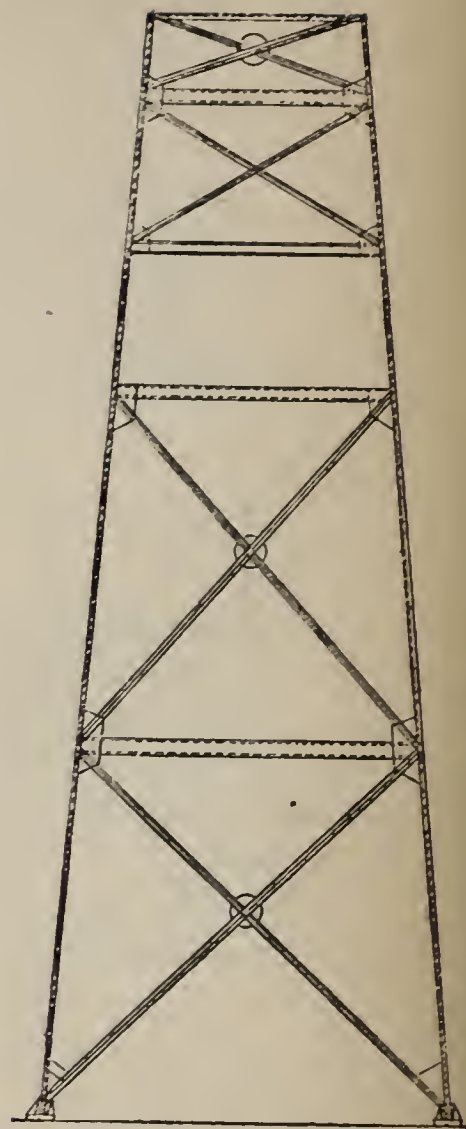


FIG. 300.—Steel frame (back stays).

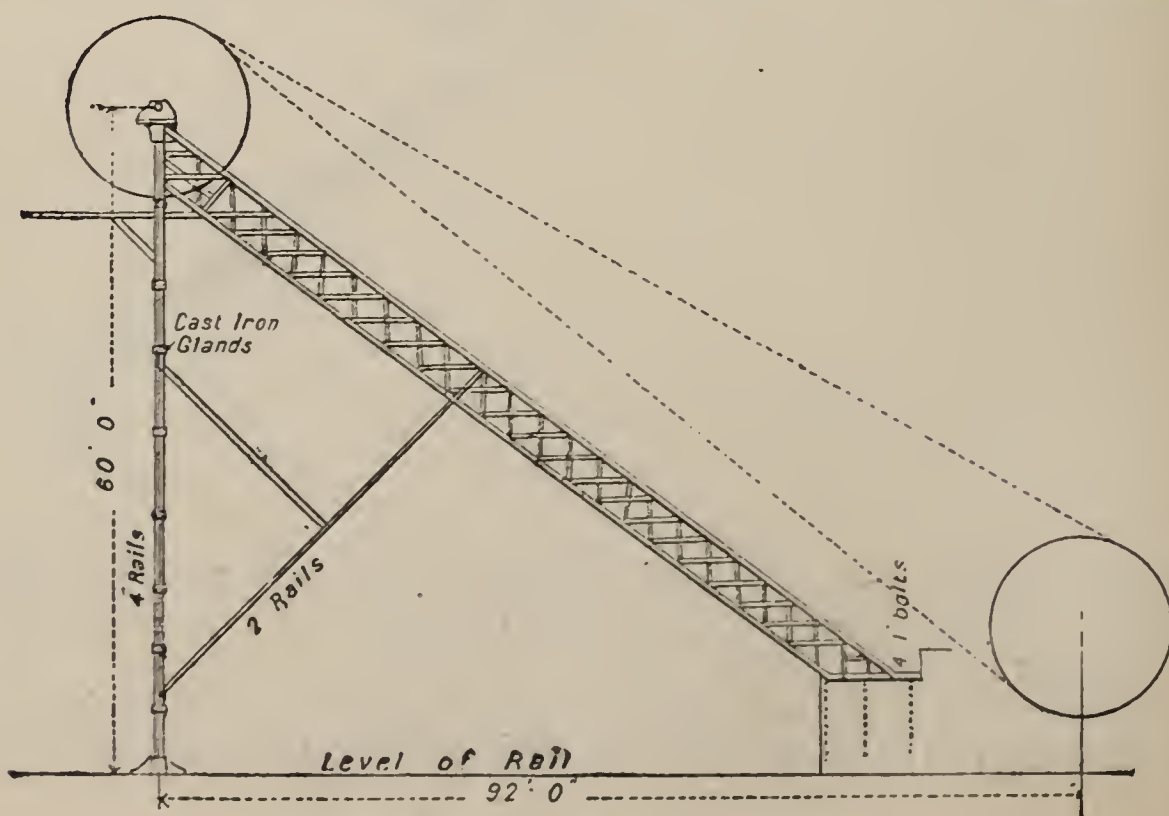


FIG. 301.—Side elevation.

geared engines, either of which may be horizontal or vertical. The best type of winding machinery is a pair of coupled direct-acting engines placed horizontally, as they are efficient, compact, easily cleaned and repaired, and well in view of the engineman. Figs. 304, 305 show the general arrangement of a pair of horizontal direct-acting engines. Figs. 306, 307 show a large pair of modern horizontal engines fitted with a scroll drum. These engines were built by Messrs Andrew Barclay, Sons, & Co., Ltd., Kilmarnock, for a new colliery at Sydney Harbour, New South Wales, and are designed

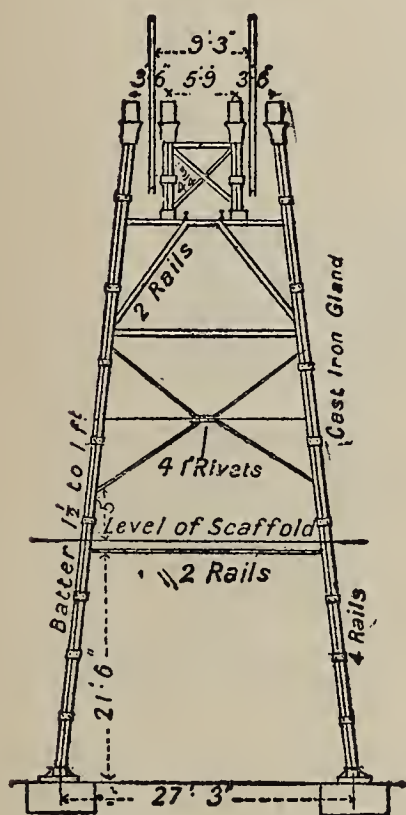


FIG. 302.—Front stays.

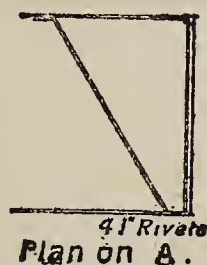


FIG. 303A.

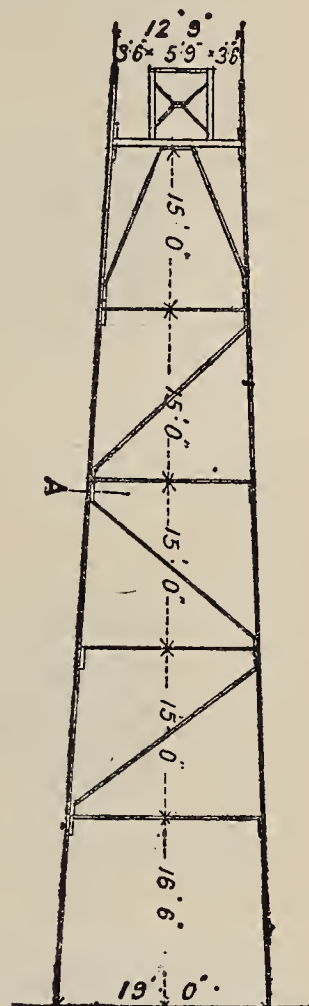


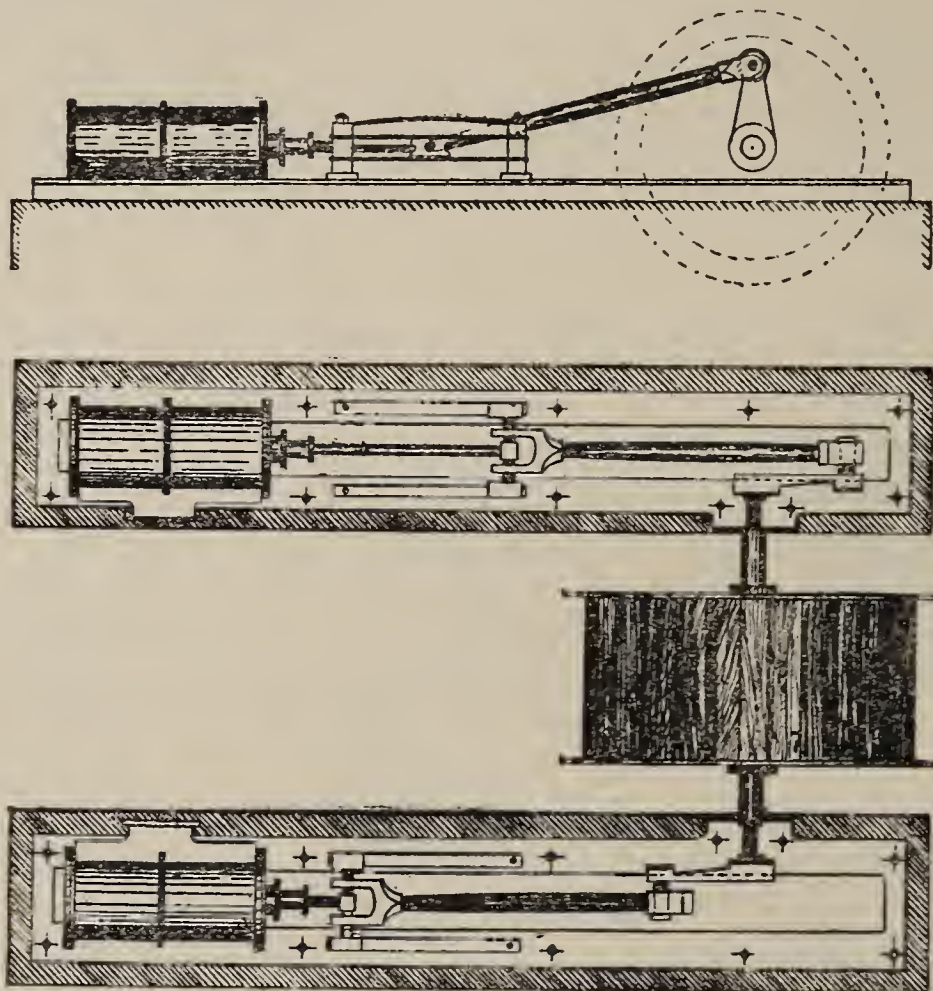
FIG. 303.—Back stays.

to wind coal from a depth of 800 yds. The cylinders are 40 in. diameter $\times$ $6\frac{1}{2}$ ft. stroke, fitted with steam and hand brakes and cut-off gear.

Condensing and expansion forms of winding engines have not been much used owing to their difficulty of application for colliery work, the rapid winding and frequent starting and stopping being against their working economically. At a few collieries, however, they have been employed with fairly good results.

Coupled engines working at high pressure and provided with automatic cut-off valves are possibly the most efficient and economical type of machine for winding coal, as their working parts are few and not so complicated as in compound condensing engines.

Single-acting engines may be employed for winding small outputs from shallow shafts, if geared and fitted with a heavy fly-wheel. Such engines are not, however, to be recommended, as they are very unsteady in their motion, and occasion delay and annoyance when they stop on a 'dead centre.'



FIGS. 304 and 305.—Horizontal engines, with both cranks shown in position at end of stroke.

The following conditions should be satisfied in a good winding engine:—

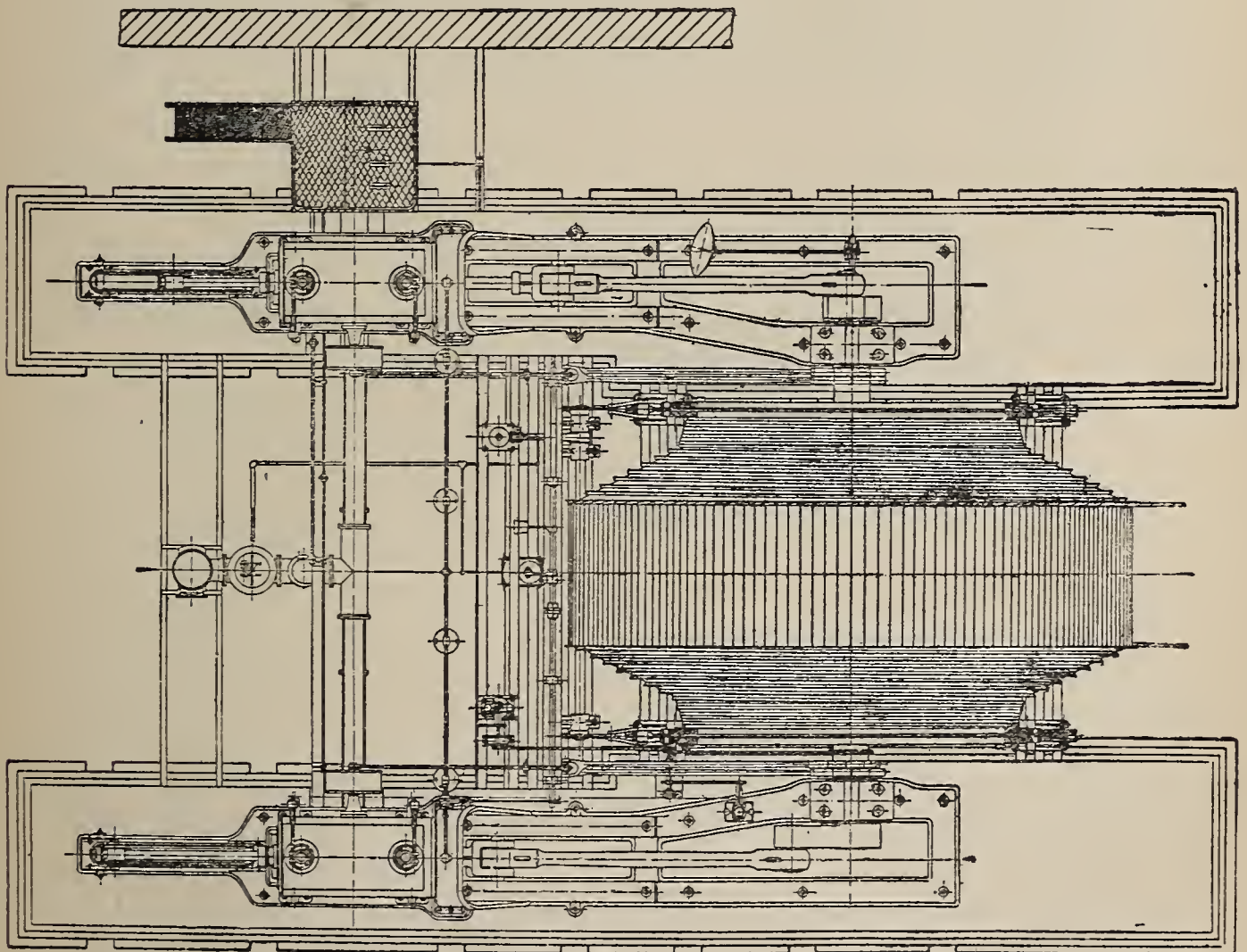
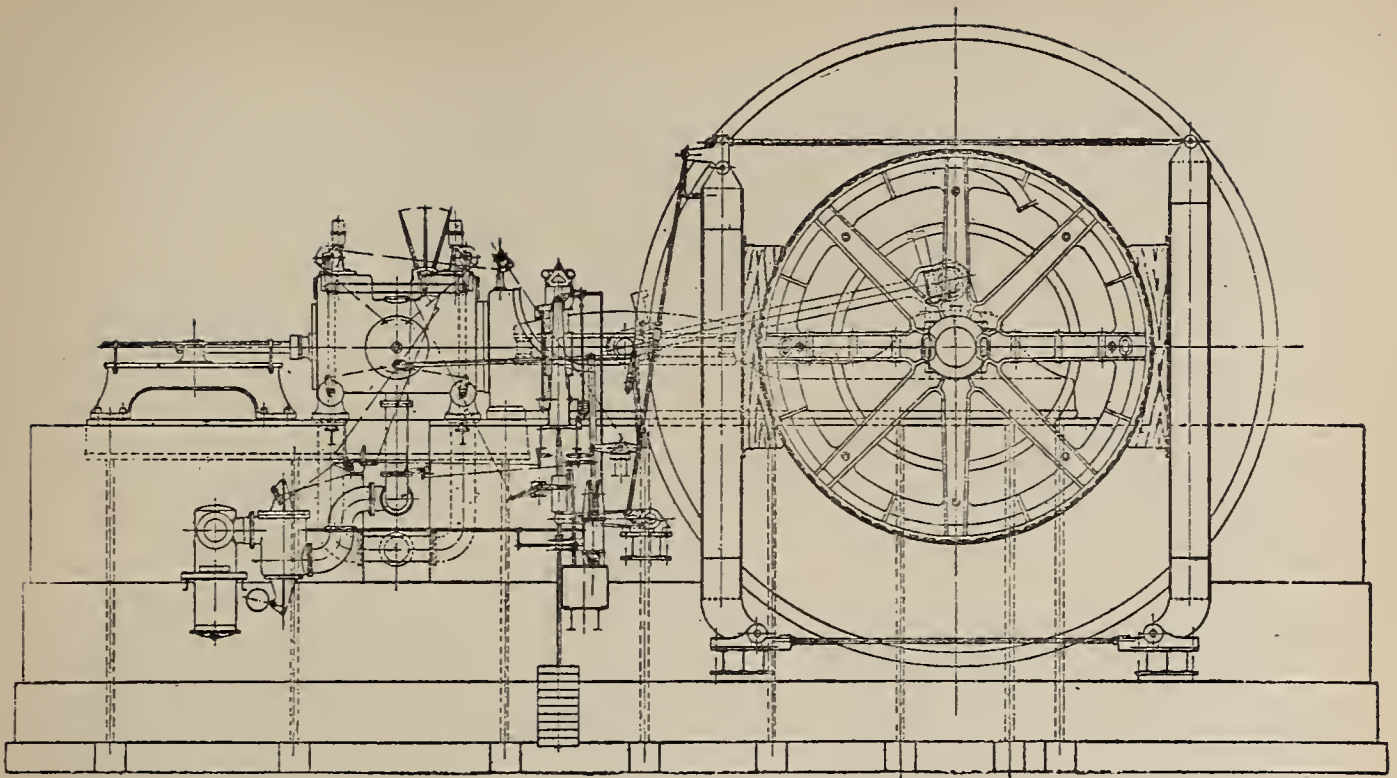
It should be as direct-acting as possible, *i.e.* the connecting parts between the piston and the crank shaft should be few in number, as each part entails a waste of power.

The moving parts should be strong to resist stresses, and at the same time light enough to offer no undue resistance to motion. Parts moving upon each other should be carefully and smoothly machined in order to reduce friction to a minimum.

The steam should reach the cylinder easily at the proper time, and should also be able to leave the cylinder as easily.

The engine should be capable of being readily and immediately stopped, started, or reversed.

Speed of Engine.—The speed of winding engines varies according to the size and class of engine, but 250 to 500 ft. per minute, as the rate of piston travel, is considered good speed for winding engines.



FIGS. 306 and 307.

Position of Winding Engine.—The laying down of the winding engine in a proper position is a very important matter. The exact site will, of course, depend on the position of the winding drum, as the engines should be placed at a suitable distance back from the shaft to afford sufficient inclination for the ropes from the pit-head pulleys to the drum.

The distance that the drum should be from the centre of the shaft is equal to about 1 to $1\frac{1}{2}$ times the height of the pit-head frame,

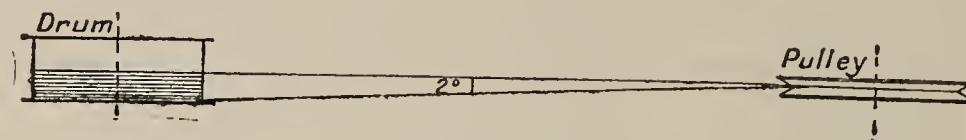


FIG. 308.

which will give a fairly good angle for the ropes to work at. Sometimes the position of the drum is fixed so as to give the winding ropes an inclination of 45° with the pulleys. Another rule which is recommended is to place the drum at such a distance from the shaft that the angle of lead of the rope will not exceed 2° when the cage is at the bottom; that is approximately that the rope should have a lead of 30 ft. for every foot of travel across the drum (fig. 308). This rule would have to be modified slightly for very shallow pits having fair-sized drums, and for very deep pits. If the angle

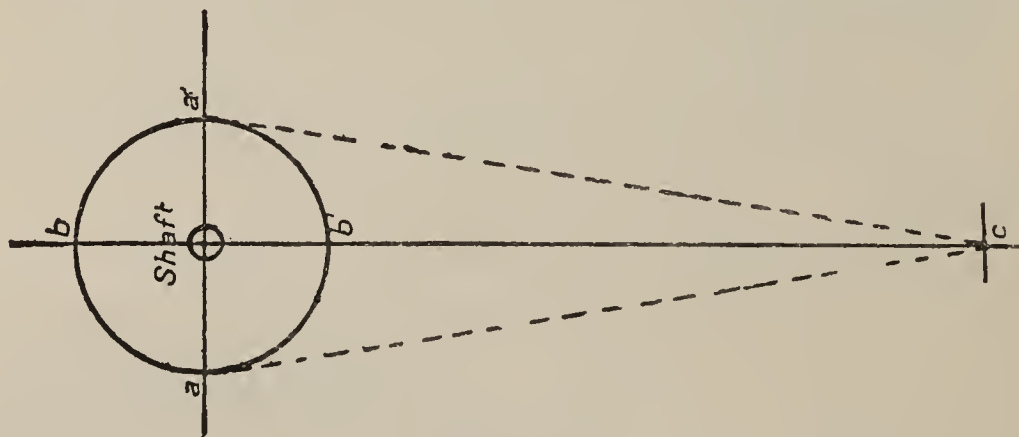


FIG. 309.

of lead is greater than 2° the side friction and side wear of the ropes will be excessive.

Having determined the exact position of the drum in relation to the shaft, it must be set off with great accuracy, either by measurement alone or with the aid of the theodolite.

The method of procedure is to get the exact centre of the shaft, or centre of barring on either side of the pit, by means of two cords, $a a'$ and $b b'$ (fig. 309), stretched across the pit at right angles to each other. From the centre of these two cords take another cord $o c$, equal in length to the distance the drum has to be placed from the centre of the shaft, and drive a stake at c . If the cord is in a

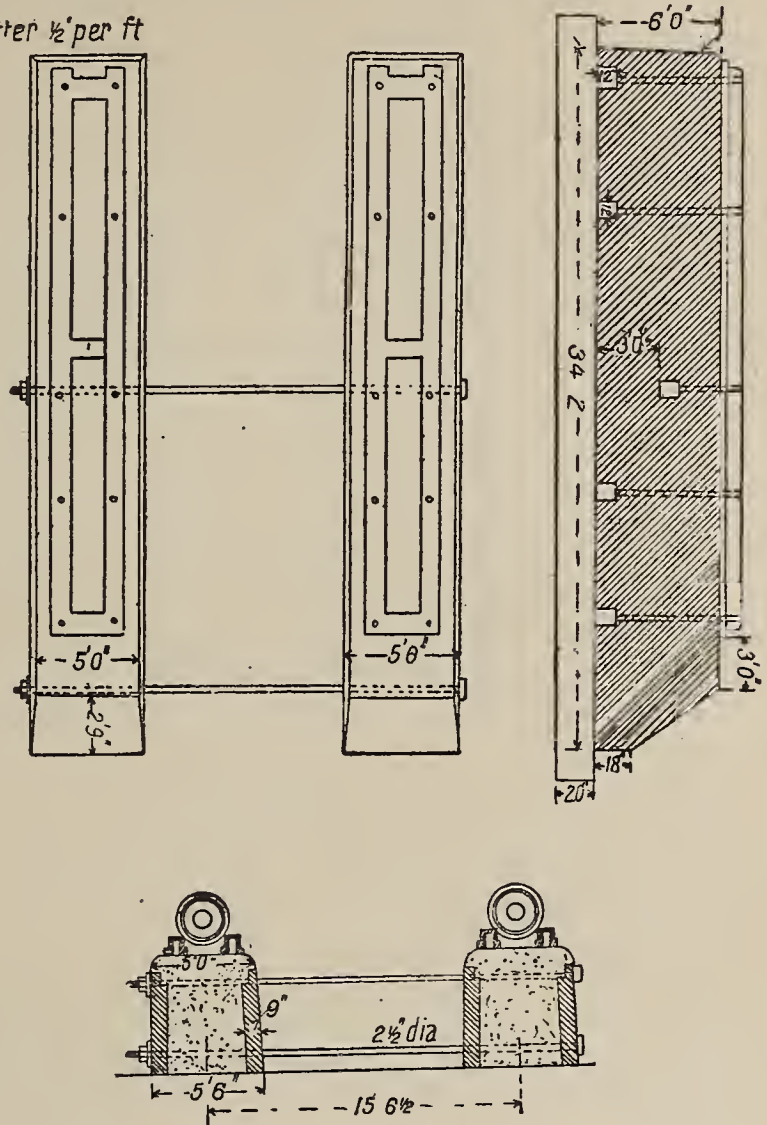
straight line, the point c will be the centre line for the drum, but it should be tested by two side cords, ac and $a'c$, the exact lengths of which can be calculated, since $ac^2 = oc^2 + oa^2$, and likewise $a'c^2 = oc^2 + oa'^2$. Both these cords should then be taken, and if the point c has been properly fixed, the ends of ac and $a'c$ will also coincide with this point and give the centre line of the drum.

Engine Seats.—Winding engines are secured in their position by seats of wood, brick, or concrete, or a combination of the two latter. For small single cylinder engines wood seats may be used, as they are easily put into position and are cheap at first cost; but they have the disadvantage of being easily set on fire and are not so stable as brick or concrete. The wood may be either pitch pine or oak logs—generally the former is used—the principal beams being 18 in. to 24 in. square and the others 12 in. to 16 in. square. For an engine with cylinder 18 in. diameter, sixteen or eighteen beams would be required, and, as pitch pine costs 2s. 6d. per cubic foot, the cost of the engine seat would be from £35 to £50.

Timber seats are now rarely used unless at very small mines, as they have nothing to recommend them except that they are easily constructed.

Brick or concrete seats are, however, preferable to those made of wood, as they give the engines a firmer foundation and a more solid bed, and are not susceptible to fire. Figs. 310–312 show the construction of such a seat composed of brick and concrete for a pair of horizontal winding engines, with cylinders 25 in. diameter. To fix the binding bolts, wood rhones 4 in. square are generally built in at the exact position for each bolt, and a set of 'pigeon-holes' left along the foot of the seat to fix the washers and cotter on the binding bolt.

Batter $\frac{1}{2}$ ' per ft



FIGS. 310–312.—Plan, end and side elevation of brick and concrete seat.

Brickwork laid in lime mortar for engine seats costs about 18s. to 20s. per cubic yard, including labour, etc.; cement alone 18s. per cubic yard; and concrete alone, 14s. 6d. to 16s. 6d. per cubic yard.

Winding Ropes.—The different forms of winding ropes used are: (1) flat, (2) circular, (3) tapered; and the materials used in their construction are: (a) hemp or other fibres, (b) iron, (c) steel. Hemp ropes may be conveniently used for shallow pits and light loads, because of the facility with which they can be made to coil round small drums. They are also much used on winches for other colliery work. The great objection to their use under other conditions is their weight; the weight of hemp rope for a breaking load of 20 tons would be about 20 lbs. per fathom, while the weight of a steel rope for the same breaking load (20 tons) would only be about 6 lbs. per fathom. On the Continent of Europe ropes made of Manilla and of aloe fibres are greatly in favour, even for very heavy loads and deep pits, and seem to be preferred to steel ropes, and they are said to give good results both as to wear and safety.

Iron Ropes are still a good deal used, and are recommended by some as superior to steel, both as regards wear and in affording better indications before breaking, besides being more pliable. But with the different qualities of steel now in use, these advantages over steel ropes no longer hold good, as varieties of the latter can now be had suitable for work under almost any circumstances.

There are four qualities of steel wire used for making winding ropes, viz. :—

				per sq. in. sectional area.
Plough-steel wire with breaking strain of 95 to 120 tons.				
Patent	„	„	„	65 to 90 „
Bessemer	„	„	„	45 to 60 „
Iron	„	„	„	30 to 40 „

For shallow pits where the load is light, it is found that Bessemer steel ropes are the most economical, because the first cost is considerably lower than that for those made from higher qualities of steel.

Flat Steel Ropes.—This kind of rope is not much used for winding, nor is it to be recommended for such purposes. It is more difficult to get a perfect flat rope than a circular one; the latter throughout being made by machinery, whereas the stitching of the flat rope is done by hand. The disadvantages of the flat ropes are :—

Greater first cost, such ropes being 30 per cent. to 50 per cent. more costly than circular ones.

Very much shorter life.

Greater liability to failure.

Against this, the only advantage is the greater diameter of the drum of the descending rope which assists to lift the load at the beginning of the wind.

Circular Ropes.—These ropes are most largely used and are made of from four to seven strands, each strand consisting of five to thirty-seven wires, and for some purposes even more. The different types of circular wire ropes for use in mines are generally constructed of:—

(a)	6	strands of 7	wires each	twisted on a	hempen centre.
(b)	6	„	12	„	„
(c)	6	„	19	„	„
(d)	6	„	24	„	„
(e)	6	„	37	„	„

The construction is sometimes varied; for instance, in the nineteen-wire strand thirteen larger wires may be coiled round six smaller wires, but the general construction is the same. This is called a compound rope. Again, the centre core, or centre, instead of being hemp, is sometimes made of a softer quality of steel than the wires. The different types of ropes are selected in accordance with the different conditions under which they have to work. For instance, type (a) should only be used with large drums and wheels and where easy curves can be employed, such as in winding from shafts or for haulage band ropes. Generally, the individual wires in all ropes should be of as large a diameter as possible, consistent with the size of drums and wheels. Compound ropes usually give better wear than those in which all the wires in the strands are of a uniform size. To obtain good results, a wire should not be bent over a wheel or drum 1000 times its diameter.

The ratio of the diameter of the individual wires to the diameter of the rope should be in some definite proportion. For the types (a), (b), and (c) above, the ratio should be $\frac{1}{9}$, $\frac{1}{12}$, and $\frac{1}{15}$ respectively, and from these the gauge of wire required to construct a rope can be got to a close approximation. The strands are usually laid round a hemp core, made of long fibre Russian hemp, or where clips are used, as in haulage, of Manilla hemp, which has a harder fibre and is less liable to deteriorate. This hemp core should be carefully treated with linseed oil or other preservative, to prevent wasting from internal corrosion. From an external examination a rope may appear quite sound, but if a piece is cut off and examined internally it will probably be found, after a certain period of work, that corrosion has set in, attacking the wires round the core, and seriously reducing the strength of the rope.

Wire of a soft quality steel is preferable for haulage, especially where clips are used, and because it bends round quick curves with ease, and winds round small pulleys without injury. The essential features in a good winding rope are flexibility and strength, and it is desirable to obtain these qualities with the least possible weight. The weight of a winding rope is a very important matter; the *dead* weight to be lifted by the engine should be as little as possible. Another point to be considered is that the strength of the rope is in

some degree dependent on its own weight, as the weight of the portion suspended in the shaft must be subtracted from that of the safe load.

Life or Durability of Ropes.—This will in a great measure depend on the treatment they receive and the work performed daily. Mr W. E. Hipkins states that the life of a rope will depend on the following points:—

The quality and temper of the wire, having regard to the stresses the rope has to bear, and the conditions under which it has to work.

Its construction as regards number of wires, strands, and nature of core.

The ratio of the lay of its wires to that of its strands, and their proportions to the diameter of the drum or pulley over which it works.

The nature of the dressing with which it is lubricated, and the mode and frequency of its application.

The number and angle of the turns it is required to make in working

All ropes ought to be well tested at stated intervals, by taking a piece nearest the cage and applying tensile and torsion tests to each individual wire. The tensile test consists in fracturing the wire by direct stress. The torsion test means that the wire must stand a certain number of twists in a length of 8 in. without cracking. The bending test is sometimes used at collieries, and consists in fixing a single wire in a vice, and then bending it at right angles a given number of times to see whether the wire shows signs of failure.

Winding ropes should also be recapped at least every six months, as this gives an opportunity of examining the inside wires, and also changes the lifting point of the rope on the pit-head pulleys. Ropes should also be well dressed or lubricated, the lubricant to be applied with a stiff brush. Wherever practicable, the rope should be allowed to run through a trough, having brushes filled with the lubricant fixed on either side. Ropes treated with a good lubricant last from 25 per cent. to 50 per cent. longer. The dressing should be applied at least once a week. A good lubricant is made from the following ingredients: tar, summer oil, mica and axle grease, in varying proportions to suit varying conditions. The tar and oil must be free from acids. This combination is said to thoroughly penetrate the wires and prevents rust and so fills the cable as to give it an appearance of solidity; it resists water successfully, and does not strip. It is stated to cost only about one-eighth as much as ordinary lubricants, and to give better results.

Care and Management of Ropes.—Wire ropes ought to be carefully stored, and should on no account be placed on the ground, but upon sound planks raised several inches from the soil, so that they may be kept free from damp; they should also be covered over with tarpaulin and inspected frequently, besides having a coating of some lubricant at regular intervals. Care should be taken in uncoiling wire ropes to prevent 'kinking'; they should, during the process, be

placed on a reel or drum when being 'paid' out. During working, the greatest stress on a rope being at the moment of starting, every care should be taken to ensure perfect steadiness, as jerking is very bad for ropes. Jerking at the starting of the load may be greatly reduced if the ropes and chains are kept of equal length. It is stated that with 6 in. of slack, the tension put on the rope, due to jerks, is twice that due to the dead load, which means, in other words, that the factor of safety of ten, which is usually employed, may be reduced to five or less, according to the amount of slack allowed when the cage is resting in the pit-bottom. Another cause of damage, and sometimes breakage, of ropes, is the sudden checking of the engine (retardation) after allowing it to run at an excessive speed.

The tension on a rope at the lifting of the cage may be found from the formula

$$T = W \left(1 + 0.5 \sqrt{1 + \frac{ah}{16.1b}} \right)$$

where T = total tension at starting.

a = acceleration at beginning of wind.

h = slack, in inches.

b = elongation of rope due to load W .

W = weight: cage, rope, tubs, and coal.

The value of a may be taken at 7.5 to 10 ft. per second. The value of b is more difficult to ascertain, but in a deep shaft, 400 to 600 yds., it may amount to 2 to 3 ft. or more, according to load, especially with new ropes. No rope should ever be changed from a larger to a smaller drum, but it will do no harm to change it from a smaller to a larger.

If the following precautions are taken and carefully carried out, few accidents will occur to winding ropes:—

Choose a good quality of rope from a maker of good repute, and pay a fair price for it.

Make minute examinations of the rope at frequent intervals.

Protect the rope as far as possible from the action of the atmosphere and from water by frequently lubricating it.

Recap the rope and reverse it every six months. It is now compulsory for winding ropes to be recapped every six months.

Test portions at regular intervals.

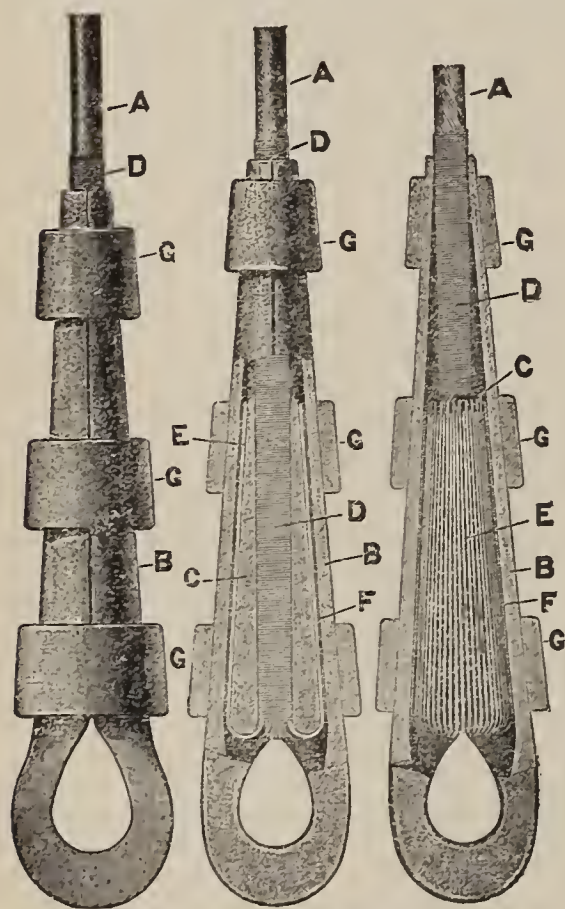
Discard the rope after it has been in use a certain fixed time, even if it is apparently sound, so far as outside examination shows. The Coal Mines Act, 1911 (40 (5)) enacts that winding ropes, used for lowering or raising persons, must be discarded after a period not exceeding three and a half years. This practically means that all winding ropes must be renewed every three and a half years, as at some periods persons will be raised or lowered in every shaft, as the shaft and its fittings must be periodically inspected.

A careful record ought to be kept of all ropes, showing the length of time at work and the quantity of mineral raised by them, and also

the speed at which they worked, as it is only by doing so that a fair comparison can be instituted between different ropes.

Rope Cappings or Attachments.—The proper capping of winding ropes is of great importance, for it is at such attachments that the rope wears quickest, and consequently where it will be most likely to give way. The cappings are fixed on the ropes in a variety of ways.

In the old method, which is still used to a considerable extent, the capping is formed by two semicircular sockets which nearly surround the rope, thickened out at the bottom end and formed into a link for attaching to the cage chains (fig. 319). The rope is fixed



FIGS. 313-315.

- A = wire rope.
- B = socket or capping.
- C = hollow conical plug.
- D = wire lapping on rope.
- E = ends of wire of the rope turned back over the cone.
- F = wire binding.
- G = loose rings.

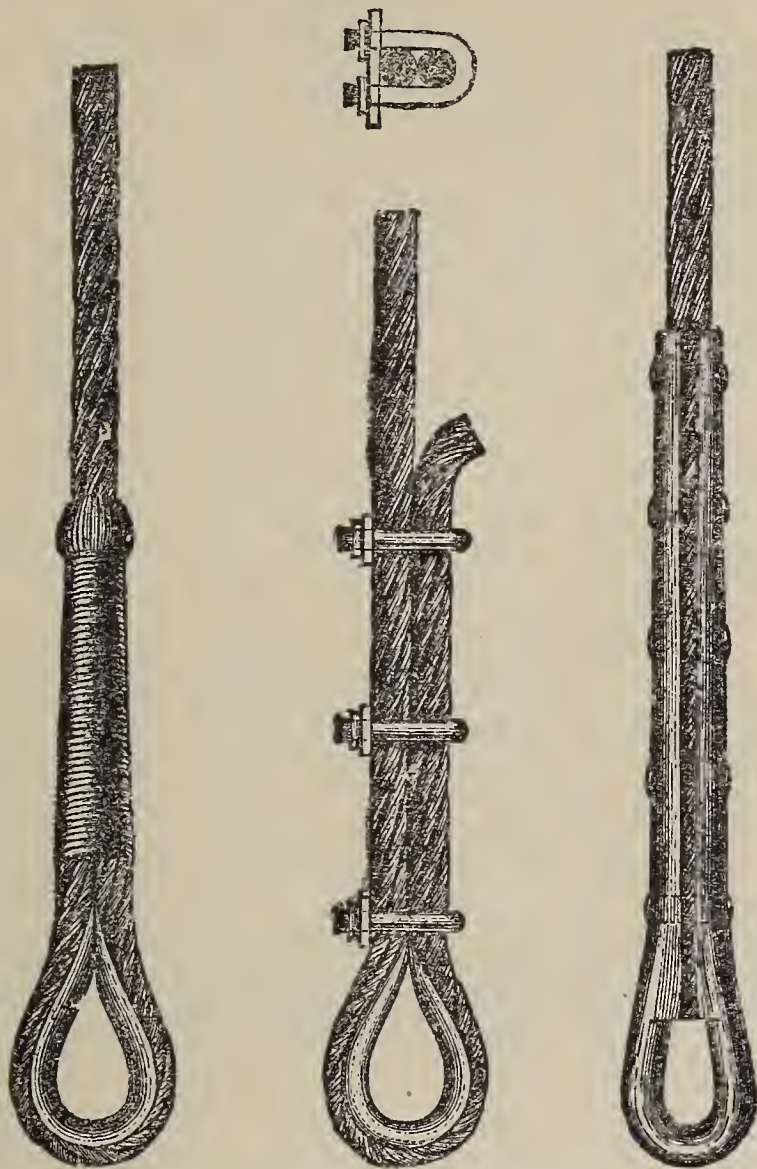


FIG. 316.

in this cap by rivets which pass through the capping and rope. To secure the rope properly, a part of the end is taken and the wire strands frayed out and bent back on themselves, the part of the rope to which this is done being firmly wound with tarred cord and tapered upward, to suit the shape of the capping or socket. When this is finished the socket is fitted on, and holes are carefully made through the rope with a marlinspike, to coincide with the rivet holes in the capping; as each hole is made, a rivet is driven through and well hammered when in position. This method gives fairly good results if the riveting is done carefully, but there is always the possibility of damaging the wires when the rivet holes are being made.

Another method is to use a socket with hoops, as in figs. 313-315. The rope is treated as already described, and drawn into the socket,

and the rings then hammered firmly down into position. A third method is to use a solid conical socket or capping, which requires neither rings nor rivets (see fig. 316). The capping is made with a conical opening, and through this opening, and for some feet beyond, the end of the rope is drawn. The strands are now opened up as before, and laid back over themselves, some of the wires being cut off, and the rest carefully wound with copper wire



FIGS. 317, 318, and 319.

until the end of the rope itself assumes a conical form; it is now drawn into the socket and is ready for use. Except under very exceptional circumstances, it will be impossible to draw the thick end of the rope through the small end of the socket unless the capping were to split, which rarely happens. For additional security where heavy loads require to be raised, a collar is shrunk on. Figs. 318, 319 show the method of attaching flat and circular ropes respectively with capels and clamps.

A method of capping which has been much used in recent years is to run in white metal to form a solid core at the end of the rope.

This system is carried out in several ways, but that which is most commonly used, and which gives the best results, is as follows:—The strands at the end of the rope are untwisted and left straight—not bent back as in other systems—and cut to an equal surface. The length of the cap is measured off, and the rope tightly tied at the proper distance to prevent the strands unlaying when the wires are being opened out. The capping rings are now passed on to the rope in their proper order, and the rope pushed through the opening in the cap, and the rings driven down over the cap. The strands are now opened out and the wires thoroughly cleaned, and then draw the rope level with the opening in the cap. An iron spike, shaped to a cone, is now driven up the centre to form a cavity into which the white metal is poured to form a solid core. The metal is allowed to cool, and the rings caulked down to their proper position, and the white metal at the top of the socket dressed off with a file. Before the metal is run in, the apertures between the wires should be filled with clay or asbestos to prevent the molten metal running out. Differences of opinion exist as to the efficacy of this system of capping, and quite a number of cases have been recorded where the rope broke inside the capping with a strain much below the breaking strain of the rope. Like other systems of capping it must be very carefully done to give good results.

Strength of Ropes.—The strength of ropes naturally varies according to the quality of the material of which they are constructed. In winding ropes a large margin of strength should be allowed, and the gross load, including the weight of the rope between the pit-head pulley and the cage at the commencement of the lift, should never—except in rare cases—exceed one-tenth of the breaking strain.

The following formulæ are often used for finding the size of ropes for a given load, or to calculate the breaking load for a given size of rope. These formulæ give only approximate results, and are not strictly correct in every case.

SIZE OF ROPES.

Let W = breaking load in tons = safe working load $\times$ factor of safety (M).

C = circumference of rope in inches.

M = factor of safety (10 for winding and 6 for haulage ropes).

w = safe working load in tons = coal + cage + tubs.

$$A \left\{ \begin{array}{l} (1) \text{ Then } W = C^2 \times .25 \therefore C = \sqrt{\frac{W}{.25}} \text{ or } C = \sqrt{\frac{w \times 10}{.25}} \text{ for hemp ropes.} \\ (2) \quad W = C^2 \times 1.50 \therefore C = \sqrt{\frac{W}{1.5}} \text{ or } C = \sqrt{\frac{w \times 10}{1.5}} \text{ for iron ropes} \\ (3) \quad W = C^2 \times 3.00 \therefore C = \sqrt{\frac{W}{3}} \text{ or } C = \sqrt{\frac{w \times 10}{3}} \text{ for patent steel ropes.} \\ (4) \quad W = C^2 \times 4.00 \therefore C = \sqrt{\frac{W}{4}} \text{ or } C = \sqrt{\frac{w \times 10}{4}} \text{ for plough-steel ropes.} \end{array} \right.$$

These formulæ do not allow for the weight of rope hanging in the shaft, and to correct this, a second formula may be employed.

Let L = lead of full cage in tons.

d = depth of pit in fathoms.

M = factor of safety.

C = circumference of rope in inches.

$$B \left\{ \begin{array}{l} (1) \text{ Then } C = \sqrt{\frac{L}{\frac{1.5}{M} - \frac{d}{1.1 \times 2240}}} \quad . \quad . \quad . \quad . \quad \text{for iron ropes.} \\ (2) \quad C = \sqrt{\frac{L}{\frac{3}{M} - \frac{d}{1.1 \times 2240}}} \quad . \quad . \quad . \quad . \quad \text{for steel ropes.} \end{array} \right.$$

Examples.—Find the circumference of (a) a hemp, (b) iron, (c) plough-steel rope for a safe working load of 4.5 tons.

$$(a) \ C = \sqrt{\frac{4.5 \times 10}{.25}} = \sqrt{180} = 13.4 \text{ inches for a hemp rope.}$$

$$(b) \ C = \sqrt{\frac{4.5 \times 10}{1.5}} = \sqrt{30} = 5.47 \quad ,, \quad ,, \text{ an iron rope.}$$

$$(c) \ C = \sqrt{\frac{4.5 \times 10}{4}} = \sqrt{11.25} = 3.35 \quad ,, \quad ,, \text{ a plough-steel rope.}$$

Or, by second formula, supposing depth of pit was 150 fms. :—

$$\text{Then } C = \sqrt{\frac{4.5}{\frac{4}{10} - \frac{150}{1.1 \times 2240}}} = \sqrt{\frac{4.5}{.4 - .06}} = \sqrt{\frac{4.5}{.34}} = 3.63 \text{ in. for a plough-steel rope.}$$

American mining engineers state that the size of rope required for a given load should have some relation to the number of wires of which the strands are composed, and consequently they use different formulæ to the above. The following are examples of the American formulæ :—

Six-strand (9 wires rope).

$$\text{Iron } d = \sqrt{\frac{fL}{17}}.$$

$$\text{Cast-steel } d = \sqrt{\frac{fL}{34}}.$$

$$\left. \begin{array}{l} \text{Crucible} \\ \text{cast-steel} \end{array} \right\} d = \sqrt{\frac{fL}{39}}.$$

$$\text{Plough-steel } d = \sqrt{\frac{fL}{44}}.$$

Six-strand (7 wires).

$$d = \sqrt{\frac{fL}{16}}.$$

$$d = \sqrt{\frac{fL}{32}}.$$

$$d = \sqrt{\frac{fL}{37}}.$$

$$d = \sqrt{\frac{fL}{42}}.$$

where L = working load in tons (ton = 2000 lbs.).

f = factor of safety.

d = diameter of rope in inches.

raised to the top of the staple ready for another wind. Thus during the first half of the wind, and when the load is greatest, the engine is assisted; in the latter half, when the load is a diminishing quantity, the engine is retarded and brought more easily to a stop. The disadvantage of this method is, that it often requires a staple pit 40 to 50 yds. deep, involving considerable expense.

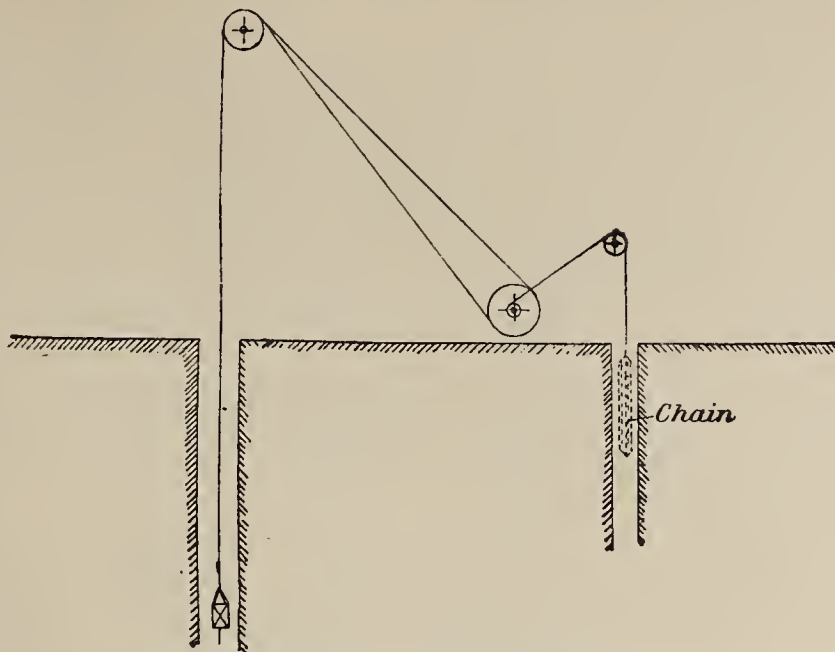


FIG. 320.—Chain and staple balance.

Tail Rope Method.

—In this system of balancing, a ‘tail rope,’ of the same weight as the winding ropes, is

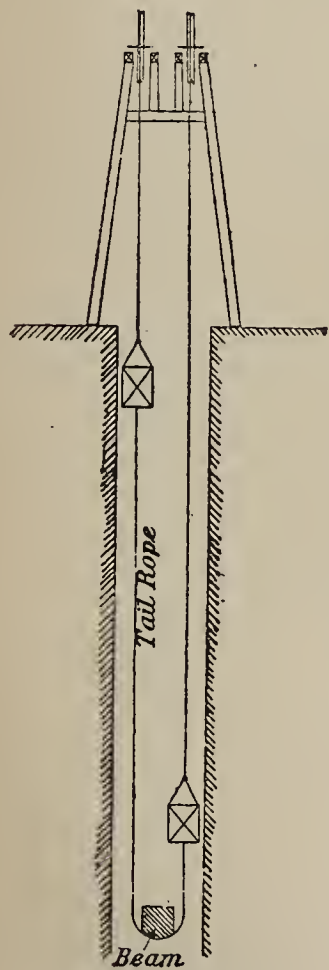


FIG. 321.—Tail rope method.

attached to the bottom of each cage, and passes round a beam, placed across the shaft, and below the level of the pit-bottom (see fig. 321). A pulley working in a sliding frame was at first used in this method instead of a beam, but it does not work so well, as the rope is apt to get off the pulley, or to pull it out of position. When the pit is deep no pulley or beam is required, the weight of the rope causing it to form a natural loop of itself. This system of counterbalancing has been found to give good results, but it is most suitable for shafts that are comparatively free from cross buntions, pump rods, pipes, and other apparatus, although the writer has seen it successfully applied in rectangular shafts, where cross buntions are necessary. The use of the tail rope is not only of advantage in balancing the engine, and assisting it at the beginning of the wind, but it also ensures much steadier running of the cages in the shaft. For this advantage alone its use is recommended. The tail rope should be attached to a bar of iron laid across the bottom of the cage, the strength of the bar being less than that of the rope, so that in case of accident the latter may give way rather than the rope.

Conical or Spiral Drums.—This method of counterbalancing

would be one of the best that could be adopted, but to obtain perfect balance, and at the same time to get a satisfactory degree of wear out of the ropes, the drums would often require to be 25 or 30 ft. in diameter, which would make them very heavy and expensive. Hence such drums are not much used, and where they were employed, they have been discarded owing to accidents taking place by the rope slipping on the drum. To prevent slipping, a spiral groove is sometimes turned on the drum. In many cases the spiral is put on the drum by riveting an angle iron on shell plates or an openwork frame, the angle iron forming a continuous spiral from end to end of the sloping portion of the drum (fig. 322). By this method it is almost impossible for the rope to slip. The drum is usually so arranged that several coils of the rope wind round

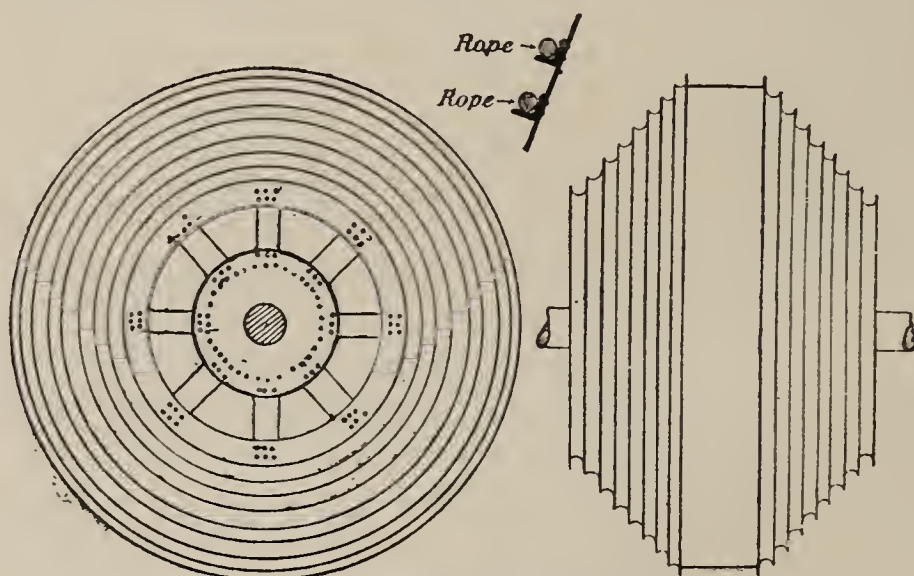


FIG. 322.—Conical or spiral drums.

the flat centre part of the drum. This enables the latter to be made smaller at the large diameter than would otherwise be the case.

To find the ratio between the large and small diameters of such drums the following rule may be used:—

Let W_1 = weight of 1 winding rope in lbs.

W_2 = „ 1 cage, tackle, and tubs in lbs.

L = net unbalanced load (coal) in lbs.

R = maximum radius of drums in ft.

r = minimum „ „

$$\text{then } \frac{R}{r} = \frac{L + 2W_1 + 2W_2}{L + 2W_1},$$

$$\text{or } R = \frac{r(L + 2W_1 + 2W_2)}{L + 2W_1},$$

$$\therefore \text{ maximum diameter of drum} = 14.55 \times 2 = 29.10 \text{ ft.}$$

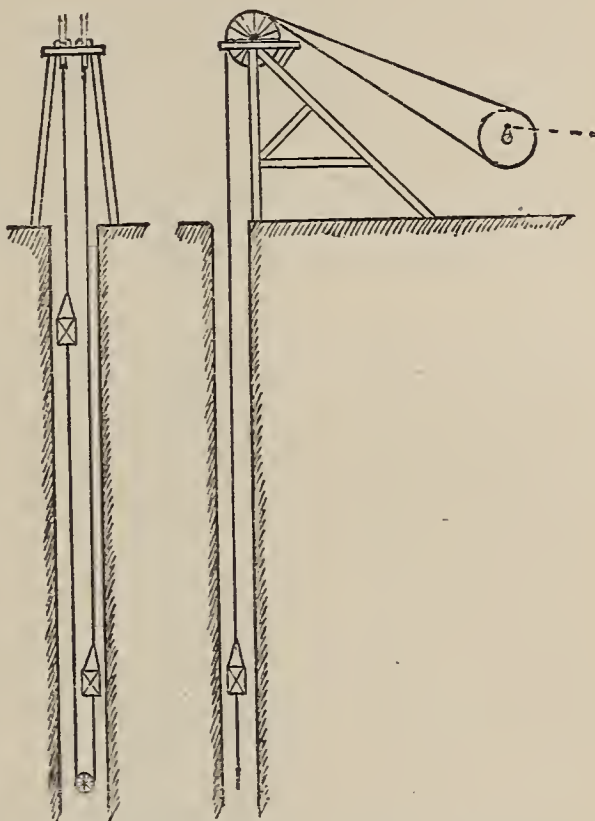
Example. -- Suppose weight of one rope (W_1) = 3770 lbs. ; weight of cage, tackle, and tubs (W_2) = 11,760 lbs. ; net unbalanced load of coal (L) = 8960 lbs. ; and minimum radius of drum (r) = 6 ft.,

$$\text{then } R = \frac{6 \{ 8960 + (2 \times 3770) + (2 \times 11,760) \}}{8960 + (2 \times 3770)} = \frac{240,120}{16,500} = 14.55 \text{ ft.}$$

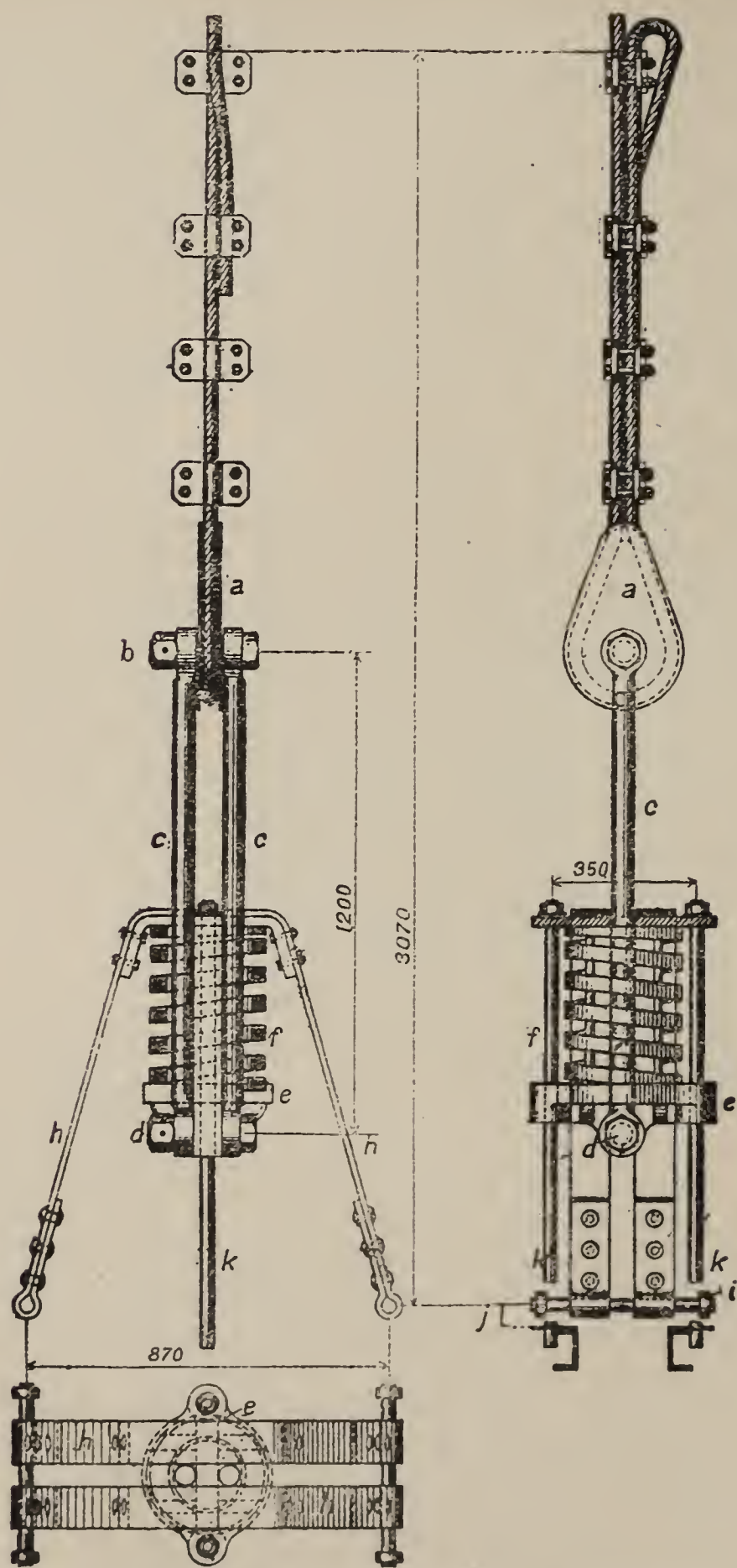
Koepé's System of Winding.—This system of winding was invented to secure a properly balanced load. Instead of a drum, as in ordinary winding arrangements, a large pulley is used, and only a single winding rope is required. The same rope extends all the way from one cage up over a pulley on the head-gear, round the engine pulley, back over the second pulley on the head-gear, down to the other cage, then to the bottom of the shaft, where it forms a loop, and lastly up to the cage from which it started. The friction between the rope and the engine pulley is sufficient to raise the useful load, which represents all the work required to be done by the engine, since the rope is balanced in every position of the cages.

This system has never had any extended application for winding in Britain. At the few places where it was adopted, it has since been discarded for the ordinary arrangement. In working, it was found that the winding pulley did not give sufficient gripping power, and the rope was apt to slip, and allow the cages to run back. Another danger was that if the rope broke, both cages would fall to the bottom. To overcome these difficulties, a modification of this system has been adopted, in which two additional pulleys are used below those of the head-gear, and at right angles to them, over each of which a rope extends from one cage to the other. These safety ropes are meant to prevent the cages from falling away in the event of the main winding rope breaking. A balance rope is also used, which is attached to the bottoms of the cages and passes round a beam in the pit-bottom.

Reducing the Strain on Winding Ropes.—As the greatest strain on a winding rope is at the moment of starting, various appliances have been introduced to reduce this as much as possible. One of the first methods was to put pieces of india-rubber below the 'mingles,' or plummer blocks, to give some spring to the pulley when the load is lifted. But rubber is a bad substance for this purpose, as it possesses a very small amount of elasticity, and soon loses the little it has, especially when exposed to the weather. Placing the pedestal on springs, like an ordinary waggon spring reversed, has been tried with much success for reducing the strain on the rope at starting to wind.



FIGS. 323 and 324.—Koepé system of winding.



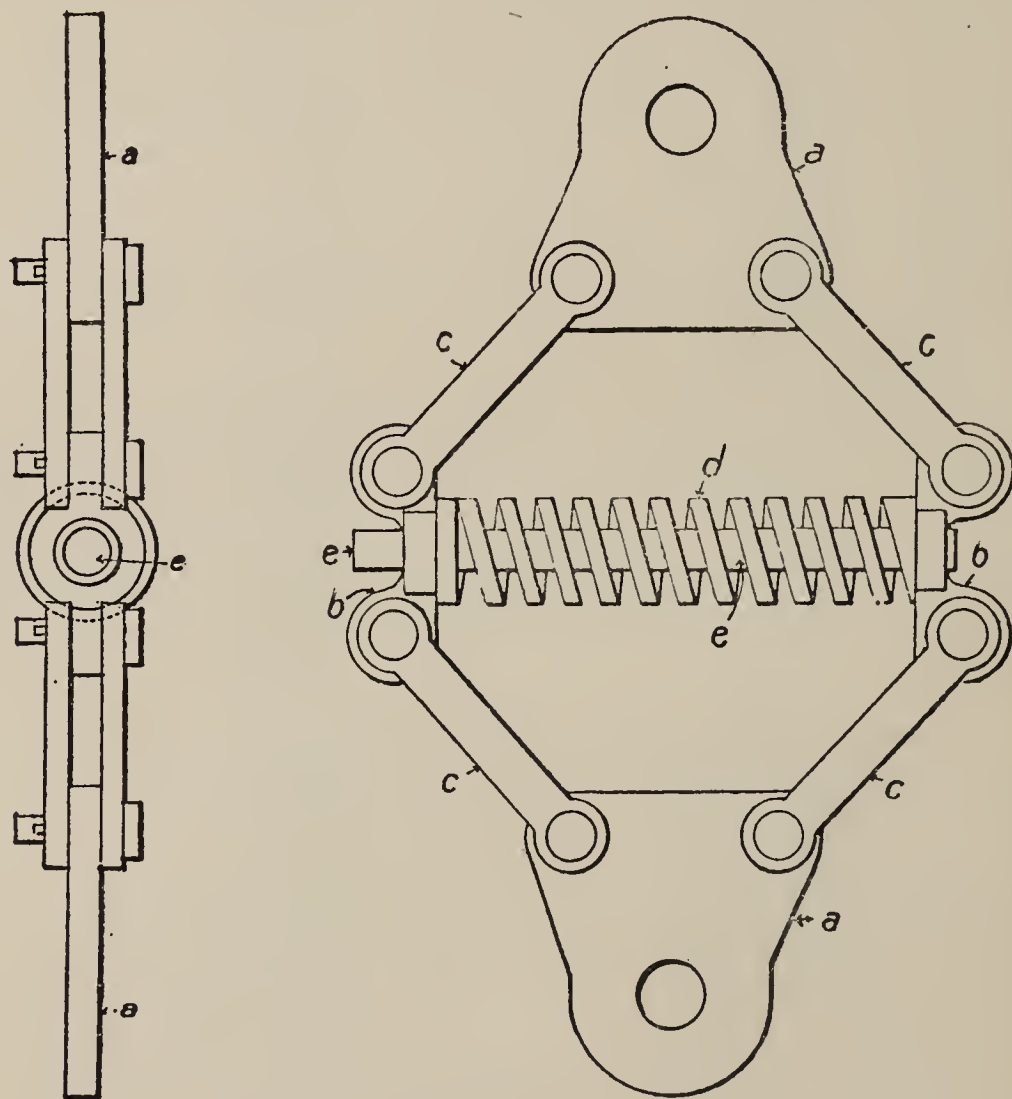
FIGS. 325 and 326. —Spring attachment for winding ropes.

Spring Attachment.—At the Gerhard Colliery, near Saarbrücken, in Germany, a spiral arrangement is used for attaching the rope to the cage. The former is placed round a sheave *a* (figs. 325, 326), and the loose end is fixed to the rope by means of four clips. A bolt *b*, $2\frac{1}{2}$ in. diameter, runs through the sheave, to which two round wrought-iron rods *c* of a diameter of 2 in. are attached at their lower ends. These rods are connected at the bottom by another strong bolt *d*. Upon this last bolt rests a plate of cast steel *e*, which receives the powerful spiral spring *f*. The upper end of the spring acts upon the wrought-iron plate, above which two hoops *h* are screwed. The ends of these hoops are formed into eyes, which by means of the bolt *i*, $1\frac{5}{8}$ in. diameter, are connected with the links riveted on the cages. The round rods *k* serve as guides to the plate *e*. When the cage is raised, the spiral spring presses against the iron plate under the hoops, and the cage is lifted gently. When the rope above the cage is loose, the rods *cc* descend perpendicularly on to the plate, and thus all jerking of the rope is avoided.

Spring Coupling.—Another apparatus used for the same purpose is that known as the 'spring coupling.' Figs. 327, 328 show this apparatus. It consists of two plates *aa*, working on pivots, and two end plates *bb*, connected by links *cc*, which are pivoted to both of these plates. The side plates are held apart by a spring *d*, through the centre of which passes a loose rod *e*, one end of which is fixed rigidly. When a sudden strain is put on the end plates *aa*, the side plates *bb* approach each other and compress the spring. As the strain increases, the resistance of the spring to compression also becomes greater and the compressing power of the links becomes less, so that a condition of equilibrium is attained.

Cage Guides or Conductors.—*Wood Guides.*—The cage is guided in the shaft by guides or conductors, which may be made by either (a) wood, (b) iron or steel, (c) wire rope or circular rods of iron. Wood guides may be made of oak, pitch pine, or fir, but the most suitable timber is pitch pine, as long lengths of this can be readily obtained free from knots and blemishes. Oak would give greater wear owing to its hardness, but it is expensive, and is difficult to obtain in long lengths free from faults, and it is, therefore, seldom used for guides in British mines. Fir is far too soft a wood for guides, and although low in first cost is expensive in upkeep. The size of guides required depends on the loads raised. For cages holding one tub the guides would be 4 in. $\times$ 3 in. or 4 in. $\times$ 4 in., and for double cages with two tubs, 5 in. $\times$ 4 in. or 5 in. $\times$ 5 in. For larger cages and heavier loads the size of guide may be 6 in. $\times$ 6 in., and in some of the German mines larger sizes are used, sometimes 8 in. $\times$ 7 in. The guides are cut into lengths of 12 to 30 ft., 18 and 24 ft. being lengths commonly used. They are fixed to the cross buntons or bearers in the shaft, and joined together as

shown in figs. 329, 330, 331. Fig. 329 shows a common method of joining them by an ordinary 'butt' joint, with a piece of wood *a*, 3 or 4 in. thick, and the same width as the guide to stiffen them, fixed at the joint by means of bolts having their heads counter sunk in the face of the guide so as not to catch the shoe, attached to the cage, in passing; or very often an iron plate about $\frac{5}{8}$ in. $\times$ $3\frac{1}{2}$ in. is used instead of *a*, the wood backing piece (see fig. 330). A 'scarf' joint is also used and an iron plate *b* fixed as before, with bolts

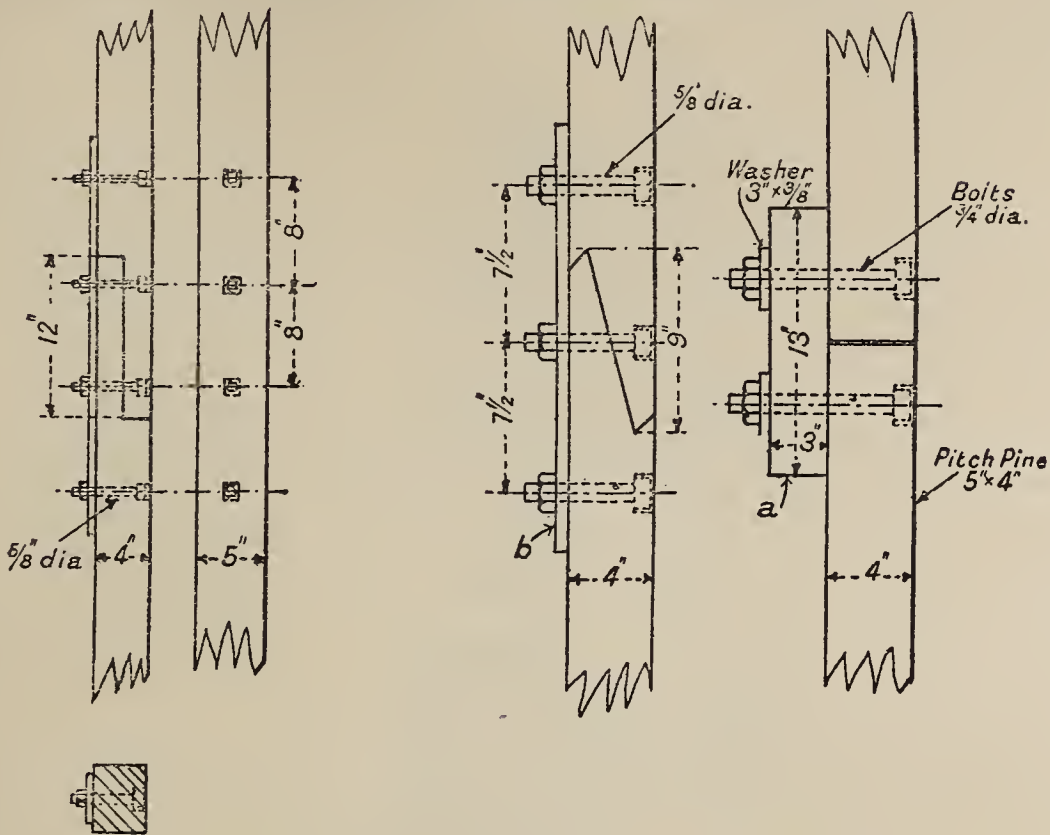


FIGS. 327 and 328.—Spring coupling.

having their heads counter-sunk (see fig. 330). Another form of joint frequently used is that shown in fig. 331, which gives good results if carefully fixed. Wood guides are most suitable for rectangular shafts in which cross buntons are necessary for their construction; they are easily fixed, cheap at first cost (about 2s. per cubic foot), but on becoming worn require frequent repairs and are very liable to cause accidents.

Iron and Steel Guides.—These guides are now very largely used. They are made somewhat in the style of an ordinary rail for surface use, and weigh from 40 to 60 lbs. per yard. Great care is required in fixing these guides to fit them to the proper gauge, and to have

all joints even and perfectly vertical, because if they are not well-fitted to begin with, they give a great deal of trouble.



FIGS. 329, 330, and 331.—Wooden guides.

Fig. 332 shows the ordinary method of fixing them; each 'chair' should carry the weight of the guide above it, as they should not

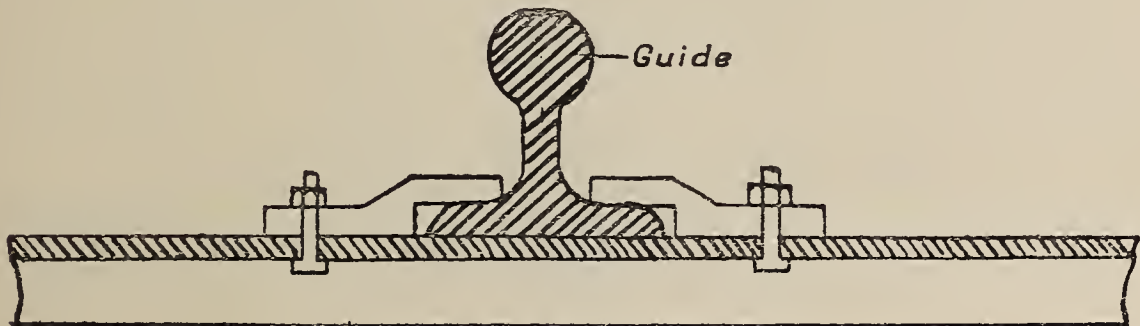


FIG. 332.—Iron guides.

rest on one another, but a small space should be left at each joint to allow for contraction and expansion. The space for admitting the guide into the chair is made a little smaller for the top side, so that a small portion requires to be taken out of the guide to admit it, the projecting parts acting as a support for it.

These rail guides should be made of steel, as iron, having very much less elasticity, causes a greater degree of vibration in the cage, which is a matter of great importance in rapid winding.

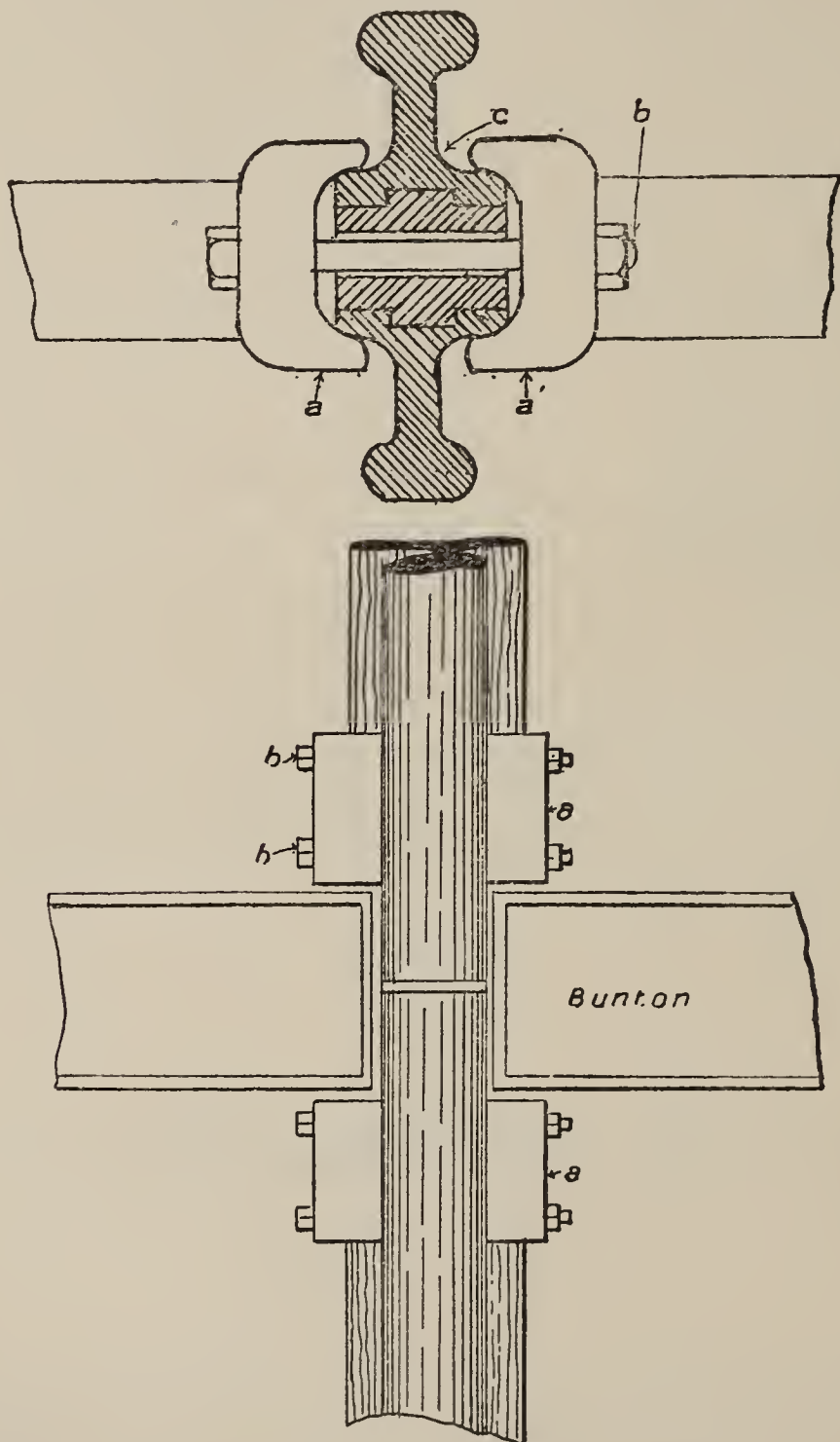
Where large cages and heavy loads are in use rail guides weighing



FIG. 333.—Shoe for steel guides.

80 to 90 lbs. per yard should be employed to give sufficient stability ; for lighter loads sections weighing 40 to 50 lbs. per yard may be used.

Briart's Method of Fixing Guides.—In this method of fixing steel

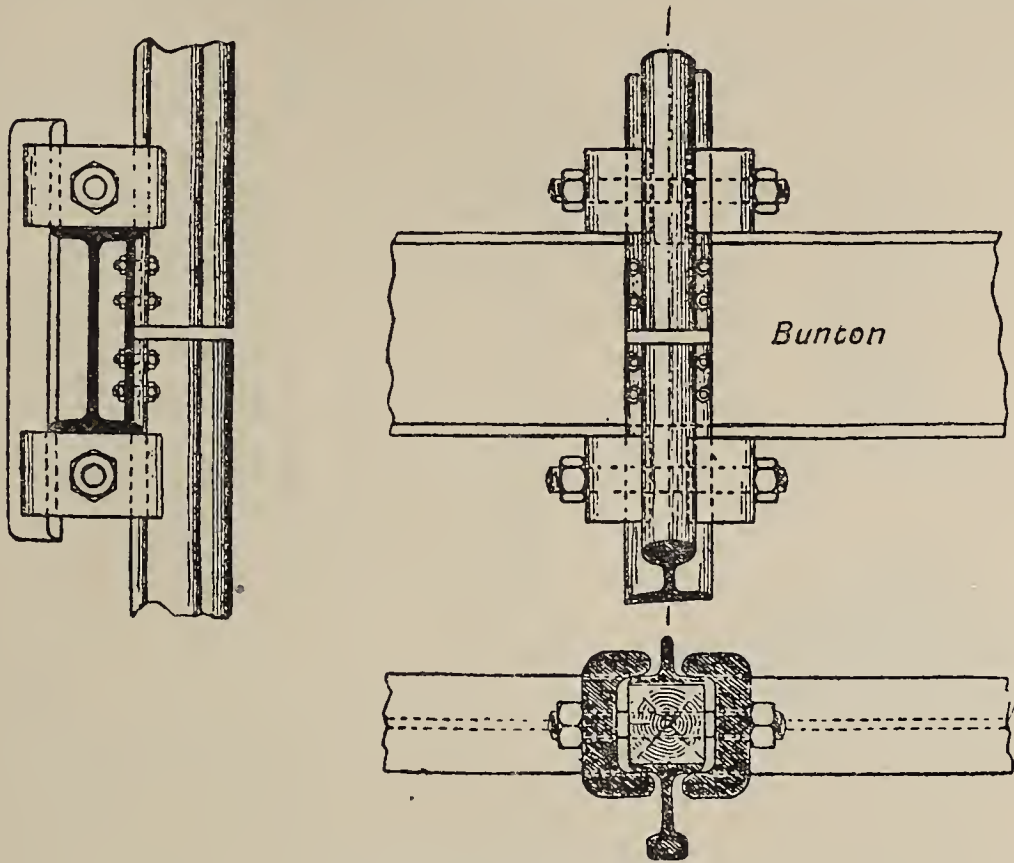


FIGS. 334 and 335.—Briart's method.

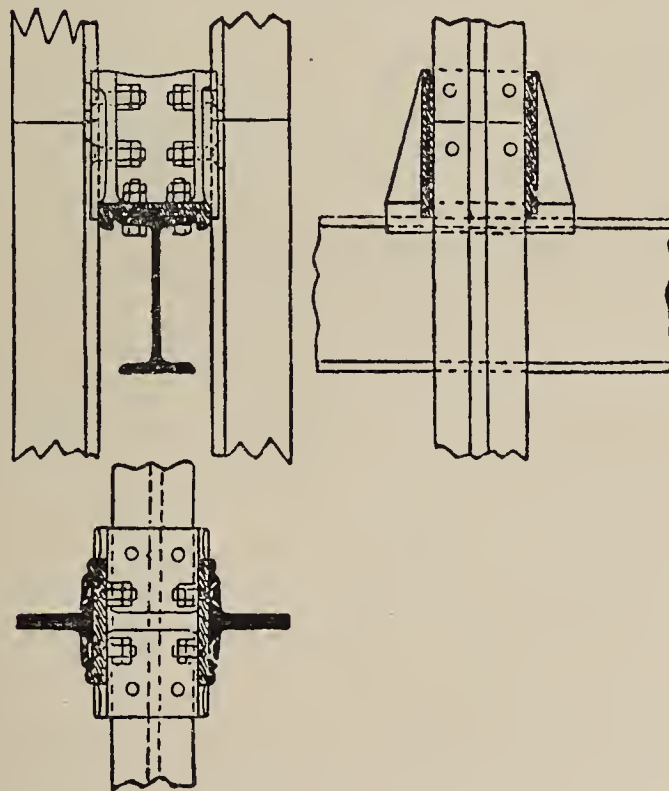
guides, which is so common on the Continent, a single series of **H** girders, 9 ft. 10 in. apart, divide the shaft.

Each guide is 19·66 ft. long, which allows a slight play between the joints. Previous to being fixed, the buntions are carefully notched to receive the foot of the rail to a depth of 0·39 in. and to a width of 4·33 in. Figs. 334, 335 show the method of fixing the guides. Two steel glands *a a* secure the rail *c* to the buntions,

one above and another below, and are made to grip the guide firmly by a pair of bolts *b b* passing through them. To prevent any move-



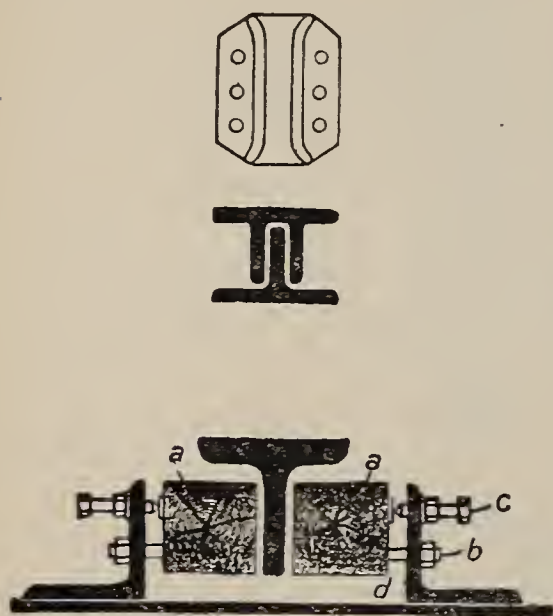
FIGS. 336, 337, and 338.



FIGS. 339, 340, and 341.

ment, or the rails from getting twisted by the pressure of the glands, a block of cast iron, through which the bolts pass, is placed between the rails, and is furnished with a slight projection, which

fits into a corresponding groove in the flange of the guide. This arrangement has been found to act very effectively. Instead of using this iron block the guide rails may be cast smooth at the flat end, and a block of wood inserted, which gives equally good results, and is cheaper in first cost. Figs. 336, 337, 338 illustrate this method.



FIGS. 342 and 343.

In some of the German mines plain T-sections $4\frac{3}{8}$ in. $\times$ $4\frac{3}{4}$ in. $\times$ $\frac{3}{4}$ in. web are used as guides, fitted as shown in figs. 339–341, and with a guide shoe made to suit (fig. 342). It was found, however, that with this arrangement the guide webs wore very rapidly, often to a knife edge, which set up excessive vibration and caused severe shock when the cage was running at a high speed. To overcome this difficulty wooden lining pieces, *aa*, fig. 343, were bolted to the cage guide *d* by a bolt *b*, and also fitted with an adjustable pin *c*, for adjusting the blocks as required. This greatly

lessened the shock, and decreased the wear of the guides. In passing through tubing, the buntons are carried in boxes or shoes bolted to the internal flanges of the tubing, the girders being wedged in position with wood keys.

Iron or steel guides are suitable for either circular or rectangular shafts, and always require buntons for fixing them; they are much more durable than wood guides, but more expensive at first cost and difficult to repair. When they get unevenly worn they can no longer be repaired, and there is also a tendency for the circular head of the guide to get worn off if the cage shoe grips it too tightly. If properly fitted at first, they will often last for years without giving any trouble, and require little repair.

Rope or Rod Guides.—These conductors are very extensively used in circular shafts where no buntons are required in their construction, and there can be no doubt as to their suitability. It must, however, be remembered that there should be at least 15 in. to 18 in. clearance between the corners of the cage and the walling to ensure safety in winding. Cold-drawn steel rods twisted together are often used instead of the ordinary rope guides; they give greatly increased strength, and wear much better. They are made up usually of seven rods, with a total circumference of $2\frac{1}{4}$ to $4\frac{3}{4}$ in., and weigh from $7\frac{3}{4}$ to 31 lbs. per fathom. The size used will

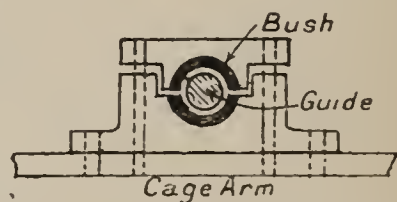
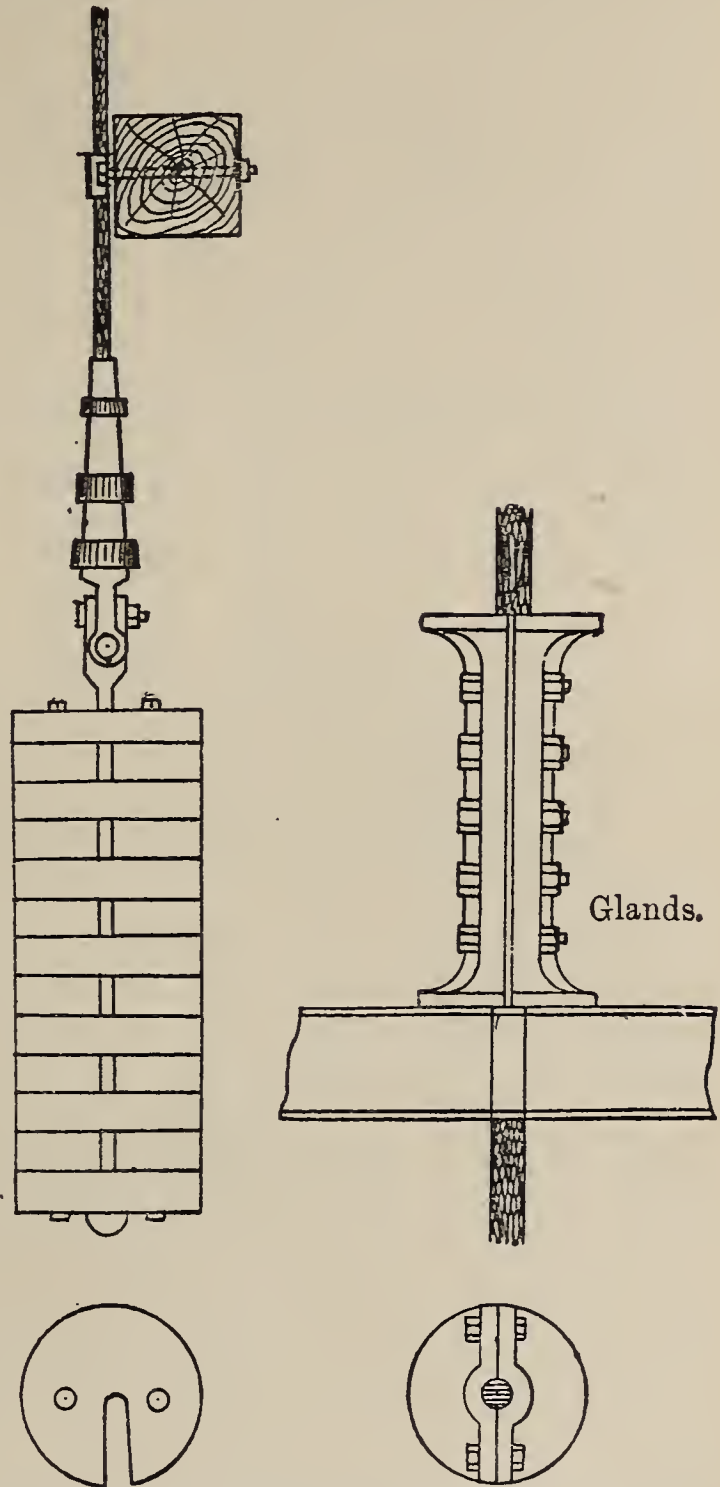


FIG. 344.—Shoe for rope guides.

depend on the load and strength required. The conductors are fastened to a strong beam at the pit-head by means of five or six glands to keep them from slipping (figs. 347, 348), or they may be fastened by a capping and strong eye-bolts fastened with a nut and washer.

At the pit-bottom they are kept tight by attaching weights to them (fig. 345). This is preferable to fixing them rigidly, as it allows for expansion and contraction, which is often of considerable amount, the weight used being about 1 ton for 250 yards of conductor, although the exact amount required can only be ascertained by experiment in each individual case. The weight should not be in the form of a single solid block, but in the form of segments or 'cheese weights,' and used in much the same way as weights are used on a 'dead-weight' safety valve. Some prefer to fix the guides at the pit-bottom and pass them over pulleys at the surface, attaching weights to the loose end. At some collieries they are fixed by a strong spring, to allow of the necessary expansion and contraction.

The number of conductors used will depend on the load and upon the speed of winding, but in ordinary cases two guides are often used, one on each side of the cage, while for heavy loads and quick winding it is best to have at least four guides, one at each corner of the cage. Some engineers prefer to have all the guides, three for long cages, at one side, the outer side of the cage, which is said to cause less vibration and less



FIGS. 345, 346, 347, and 348.—Rope guides.

liability for the cages to strike each other and on the sides of the shaft.

To prevent the cages from catching each other in passing, two or three unconnected conductors should be suspended between them; the space between the cages may then be as little as 6 in., but 12 to 15 in. is a better allowance. On the sides of the cages next the unconnected ropes, flat rubbing pieces of oak are fixed, to keep the cages at a specified distance apart, and to prevent them from striking each other at the meeting place in mid shaft. For heavy cages and high winding speeds, the rubbing ropes are made heavier than the guide ropes, frequently 6 to 7 in. circumference, and weighing 50 to 60 lbs. per fathom. Flat rubbing ropes have also been successfully used, but the more general custom is to employ circular section ropes. With rope guides it is a great advantage to use a balance rope under the cages to ensure steady running and decrease of vibration.

The advantages of using rope guides are:—

- (1) The first cost is cheap.
- (2) They are easily fixed and require few repairs and little attention, except oiling regularly.
- (3) They last much longer than wood or iron guides.
- (4) No buntons are required for fixing, and they occupy but little space in the shaft.
- (5) They are more flexible, and contract and expand more readily than rigid wooden or rail conductors.

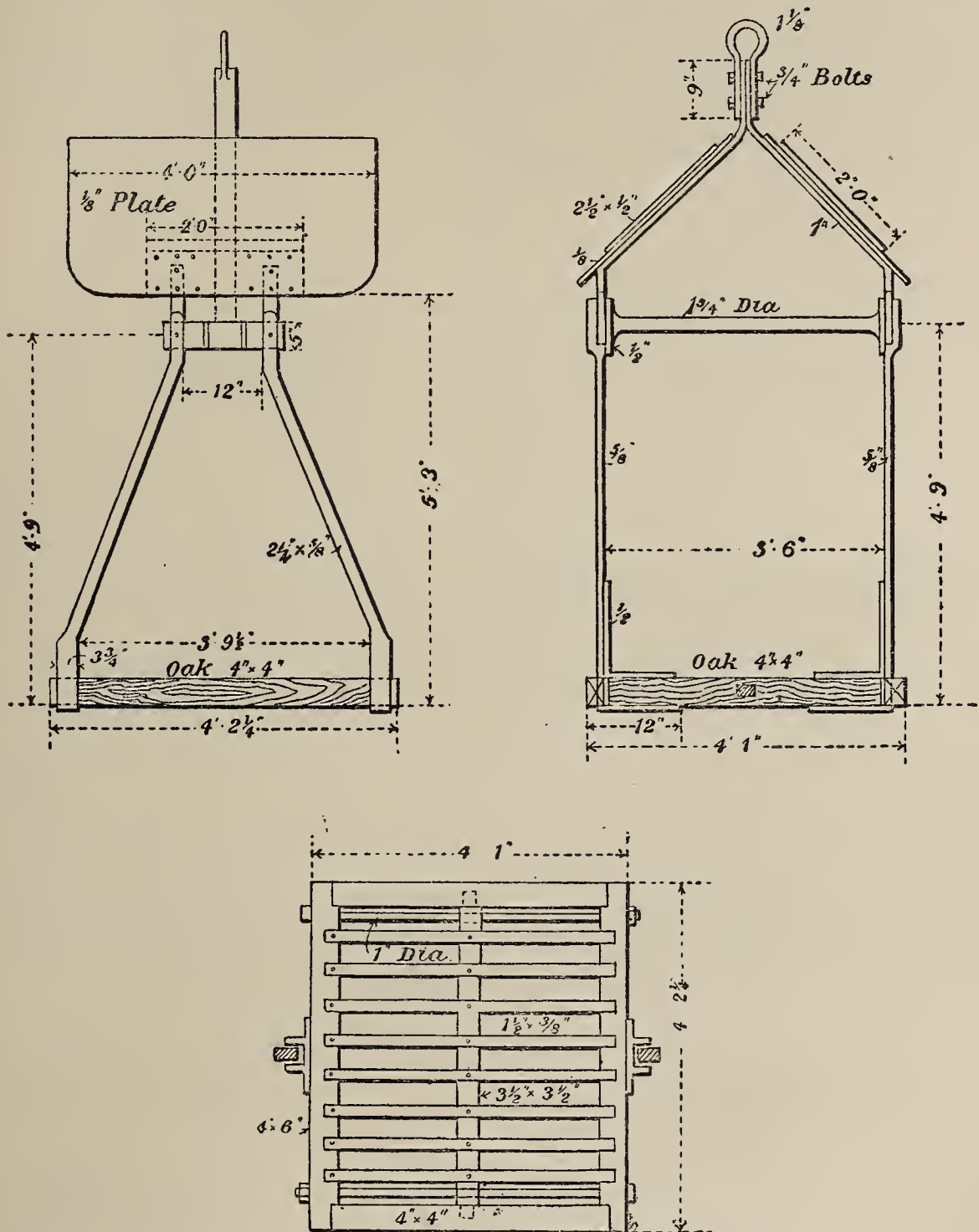
Cages.—These are made of different sizes according to the output required, the size of shaft, and that of the tubs used. The material employed in their construction is either wrought iron or steel, the bottom being often constructed of oak, but for large cages it is best to dispense with wood altogether, and to make them entirely of steel or wrought iron, as by this means their weight will be much reduced, which is an important point to keep in view. A good cage made of steel throughout should not weigh more than about two-thirds the weight of the coal raised, while a wrought-iron one should weigh about three-fourths of weight of coal raised per wind.

A cage should be so constructed as to allow the greatest carrying capacity with the least possible weight; the form selected should be such that the tubs can be readily placed in it and easily removed, while the whole construction should run easily in the shaft.

Cages cost from £30 to £35 per ton if made of wrought iron, and from £40 to £45 per ton if constructed of steel. Before cages are used they should have two coats of good paint, which will help to preserve them from corrosion in wet shafts, and particularly in pits where the water contains acid. Figs. 349, 350, 351 show the construction of cages for holding a single tub, this form shown being frequently used in Scotland, as they are easily and cheaply made.

Figs. 352, 353 show a form of single-decked cage for two tubs placed abreast, a type much used in rectangular shafts.

Figs. 354–357 show two types of double-decked cages, in one of which the tubs are placed abreast, and in the other, which is usually



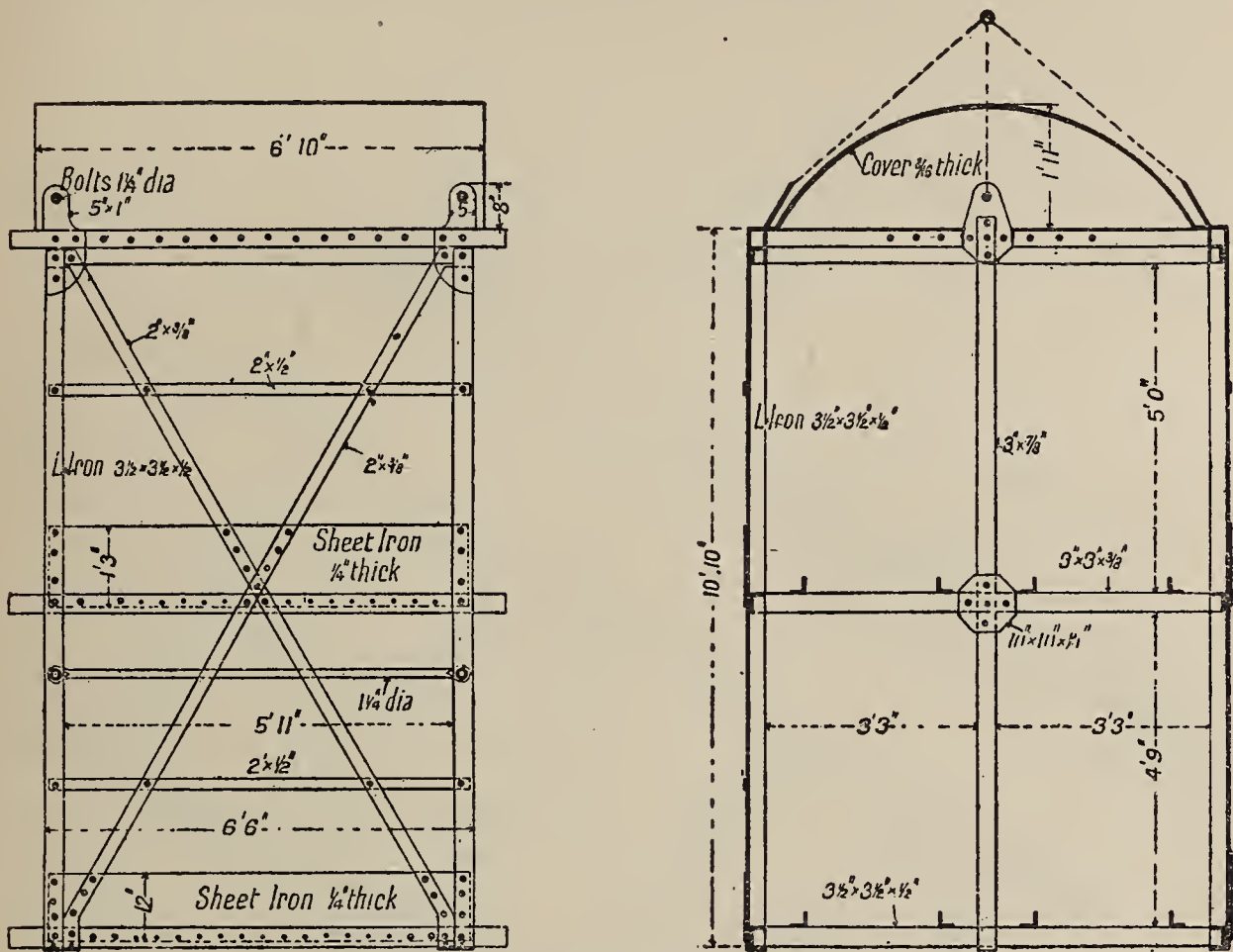
FIGS. 349, 350, and 351.—Single cage.

more suitable in circular shafts, end to end. Cages with the tubs placed abreast are generally more suitable for rectangular shafts, where the space is narrow compared with its length. Figs. 358, 359 show the detailed construction of a cage for drawing mineral on an inclined shaft.

Cage Shoes.—In order to keep the cages running in the con-

ductors while being wound up the shaft it is necessary to fix what are termed shoes on to the cage. The type of shoe required will depend on the kind of conductors with which the shaft is fitted. For small single-deck cages usually four shoes are required, but for large cages, with tubs put on tandem, the number is generally eight, two at each corner. Of course, if ropes are used for conductors, and only three are employed, then six shoes will be the number required.

When wood conductors are in use the type of shoe generally



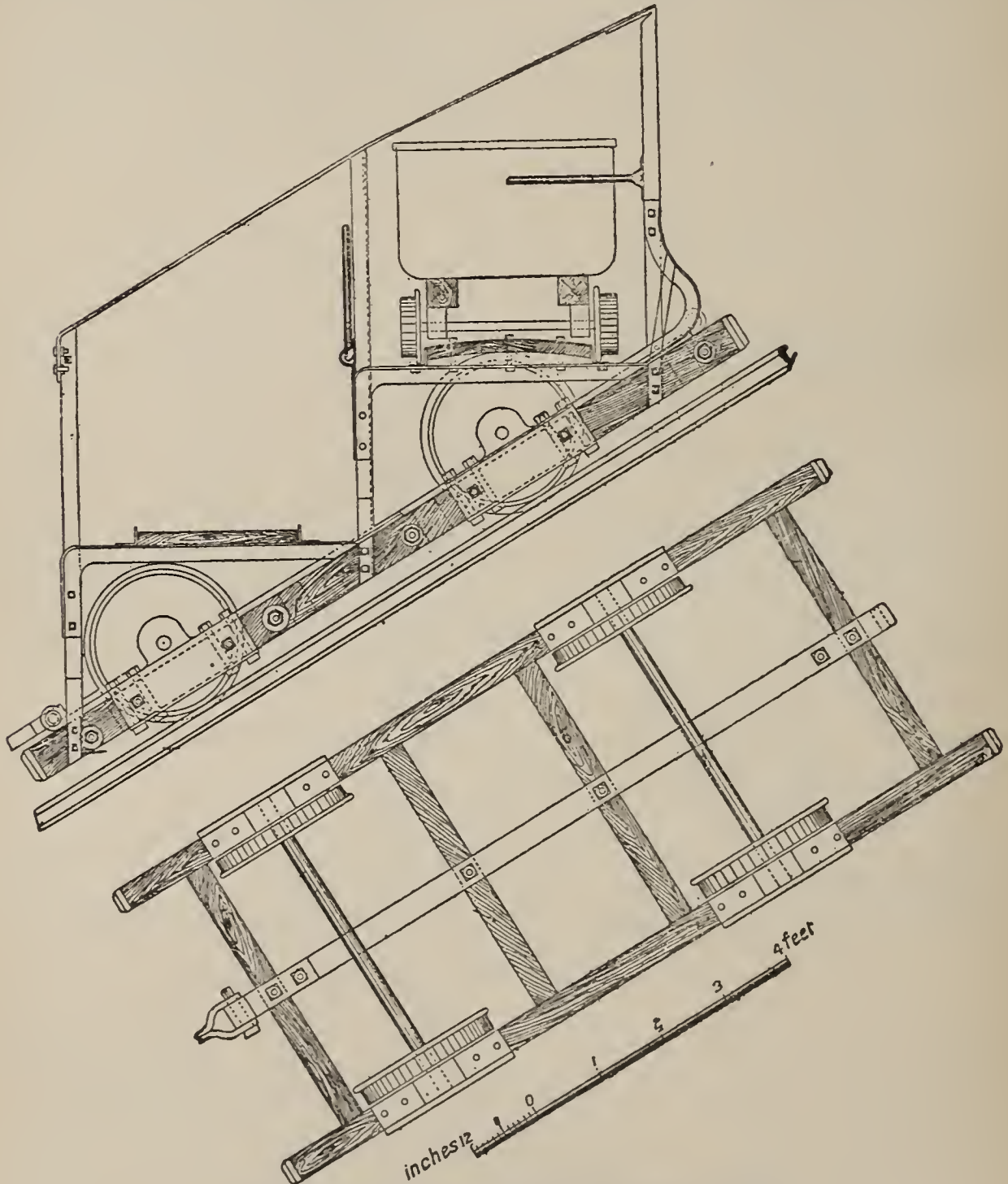
FIGS. 356 and 357.—Double-decked cage for four tubs.

employed is one similar to that illustrated in fig. 359A. Sometimes instead of using this type the shoe is constructed of channel irons made the full length of the cage. The idea of constructing them in this way is to give a longer bearing surface and ensure steadier running of the cages.

When iron guides are employed the type of guide is somewhat similar to a 'chair' used on a surface railway for holding the rails in position (see fig. 333). With rope guides a different type of guide is required, as it must wholly surround the guide. The guide is frequently made in two halves, with liners or bushes in the centre, which can be renewed when worn. The bushes are sometimes made of brass, but this material is expensive, is too soft, and requires frequent renewal. There is little advantage in using brass, and, instead, ordinary cast-iron or gun-metal bushes are largely used and

are found to give good results. The construction of the shoe is shown in fig. 344.

To prevent the swaying or oscillation of the cage when it arrives at the pit-head, the guides are sometimes placed in long iron troughs



FIGS. 358 and 359. — Cage for inclined shaft.

into which the cage shoes fit, or sometimes a set of four heavy wood guides are fitted at the surface and angle-irons fixed on to the cages to grip these.

Cage Speeds.—The speed of the cage in the shaft may be anything between 15 and 50 ft. per second. The following are average speeds : —depth 50 to 100 fms., speed, 15 to 20 ft. per second ; 100 to 150 fms., speed 20 to 25 ft. per second ; 150 to 200 fms., 25 to 35 ft.

per second; and for depths above 200 fms., from 35 to 50 ft. per second.

Fences or Gates for Cages.—The Coal Mines Act, 1911, provides that cages are “to be closed in at the two sides in a sufficient manner to prevent persons or things from projecting beyond the sides, and shall be provided with suitable gates or other rigid fences” at the open ends. The sides of the cage may be closed with either light plates of sheet-iron or by sheets of strong wire mesh. Sheet-iron plates are largely used for this purpose, but the objection to them is that they completely hide the conductors in the shaft, so that even for a cursory examination of these the plates have to be removed from the cages. With large cages the plates are necessarily heavy, and it requires some labour and time to have them removed. They ought to be put on in such a way, such as by hanging pins, so that they can be readily removed and replaced. Where the cages are constructed with a flat cover, the examination of the conductors in the shaft can be made from it, as it enables a workman to stand on it securely. The other method is to fix sheets of strong wire netting, preferably with a diamond-shaped mesh, and so arranged that they can be fixed on to hangers fixed to the arms of the cage. These sheets are lighter and handier to take off and replace when required, and, from practical experience, we prefer them to the sheet-iron. They have also the advantage that the conductors can be seen through them, but, of course, for a proper examination or repairs they require to be removed.

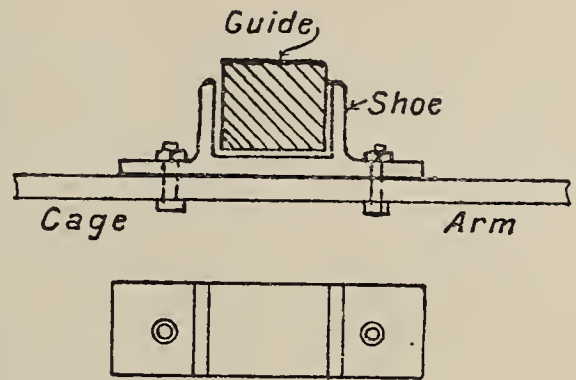


FIG. 359A.

The fences or gates used for the open ends of the cage are of great variety, depending upon the type of cage in use and the amount of space available for such appliances. In some cases two or three light circular rods of iron are used, which may be put on or taken off readily when men are about to descend or ascend the shaft. Sometimes two or three lengths of chain are used instead of the rods. In preference to either of these, some form of gate is employed. The variety of gate used for this purpose is very varied, and almost every colliery have a design of gate to suit their own requirements. Many of them are made on the telescopic principle, in much the same style as those used on hoists or elevators in buildings. A gate which is simple in construction and is easily opened and closed is shown in fig. 360. It is made in two halves, so that each half can be folded back against the sides of the cage when the winding of coal is being done. It is constructed of light flat iron bars, two cross-pieces, and three uprights to each half. Of course, a gate like

this can only be used when there is sufficient clearance space between the tubs and the side of the cage to allow of it being folded back. Whatever type of gate is adopted, it must be made so that it can be readily opened or shut, as otherwise it means considerable loss of time where a large number of men have to be lowered and raised in a certain time, a period that is now fixed by the Eight Hours Act, and has to be adhered to.

Changing Tubs on Multiple-Deck Cages.—When the cages have two or more decks the placing of the tubs in them at the surface and pit-bottom becomes a matter of considerable importance. The

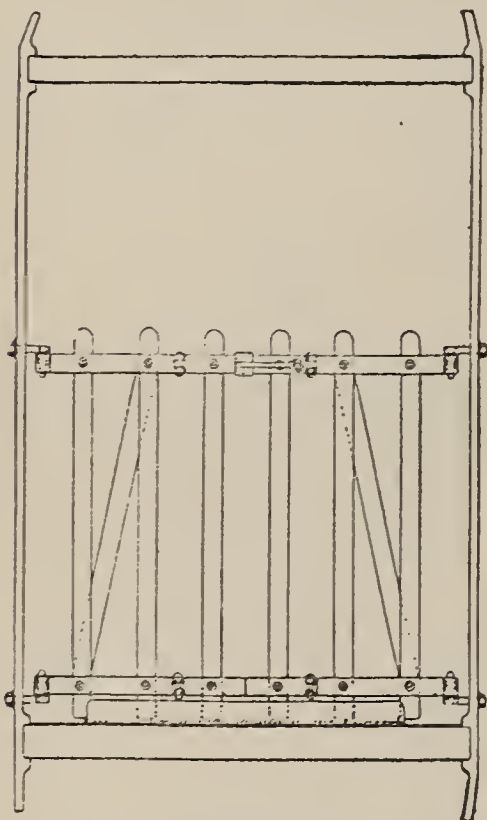


FIG. 360.—Front elevation of gate on cage.

changing of the tubs at top and bottom may be carried out in several ways.

At one time the most common method was to bring the several decks of the cage successively to the hanging-on levels by raising or lowering the cage with the engine. This system is tedious, especially when three or four decks have to be changed, as it wastes considerable time in moving the cages, it increases the consumpt of steam, and it requires great care on the part of the engineman to bring the cage always to the proper level. It is also unsuitable if conical or spiral drums are employed, as the cages at top and bottom will not be moved to an equal extent owing to the unequal diameters of the drum.

A second plan is to have two or more fixed stagings or floors all gravitating to a common level, and from these stagings the decking of the cages is carried out simultaneously. Where this method can be carried out it is a quick and easy way of doing the work, but it requires a good deal of space, especially if the cages have more than two decks, which is not always convenient, and specially at the pit-bottom.

Another method is to have the fixed stagings as described above, but, instead of allowing the tubs to gravitate to a common level, each of the stagings are provided with an endless creeper chain for raising the load and lowering the empty tubs. Probably the most common system of decking cages is to use balance cages or hoists either operated by hydraulic pressure, electricity, or made self-acting by balancing weights.

In Fowler's system of decking two sets of auxiliary cages are employed, one set for the empties and another for the loads. These cages are raised by pistons operated by hydraulic pressure, and

there are also two or more horizontal rams or pistons, according to the number of decks. When the cage lands at the surface the

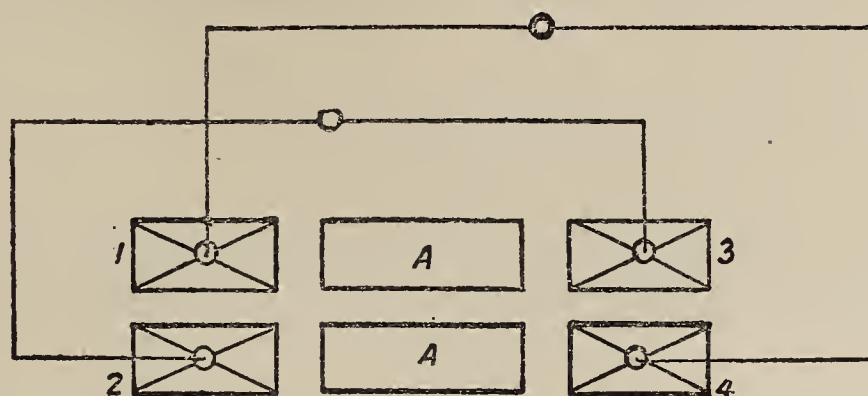


FIG. 361.

1, 2, 3, 4 = Supplementary cages.
A, A = Shaft cages.

horizontal rams push the empty tubs out of the auxiliary cage into the winding cage and at the same time displace the load tubs.

In Tomson's system * a somewhat similar arrangement is provided, as shown in figs. 361, 362. The four supplementary cages 1, 2, 3, 4 are connected by a chain or rope and rest on hydraulic rams. The cylinder of 1 is connected with 4, and similarly 2 and 3 are connected. When the full tubs are on the auxiliary cage 1, their weight in descending raises 4, which is filled with empties on each deck as fast as 1 is discharged. On cage A coming to the bank the full tubs are discharged into 2, which raises 3, and the empties from 4 take their place, and so on. A regulator enables the rate of descent to be controlled and the cages stopped in any position, and also admits water to the hydraulic cylinders in the event of the weight of the full cage being insufficient to raise the corresponding empty one.

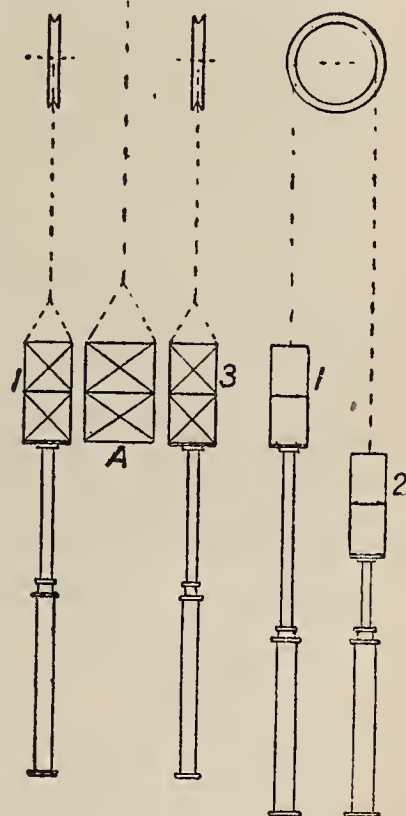
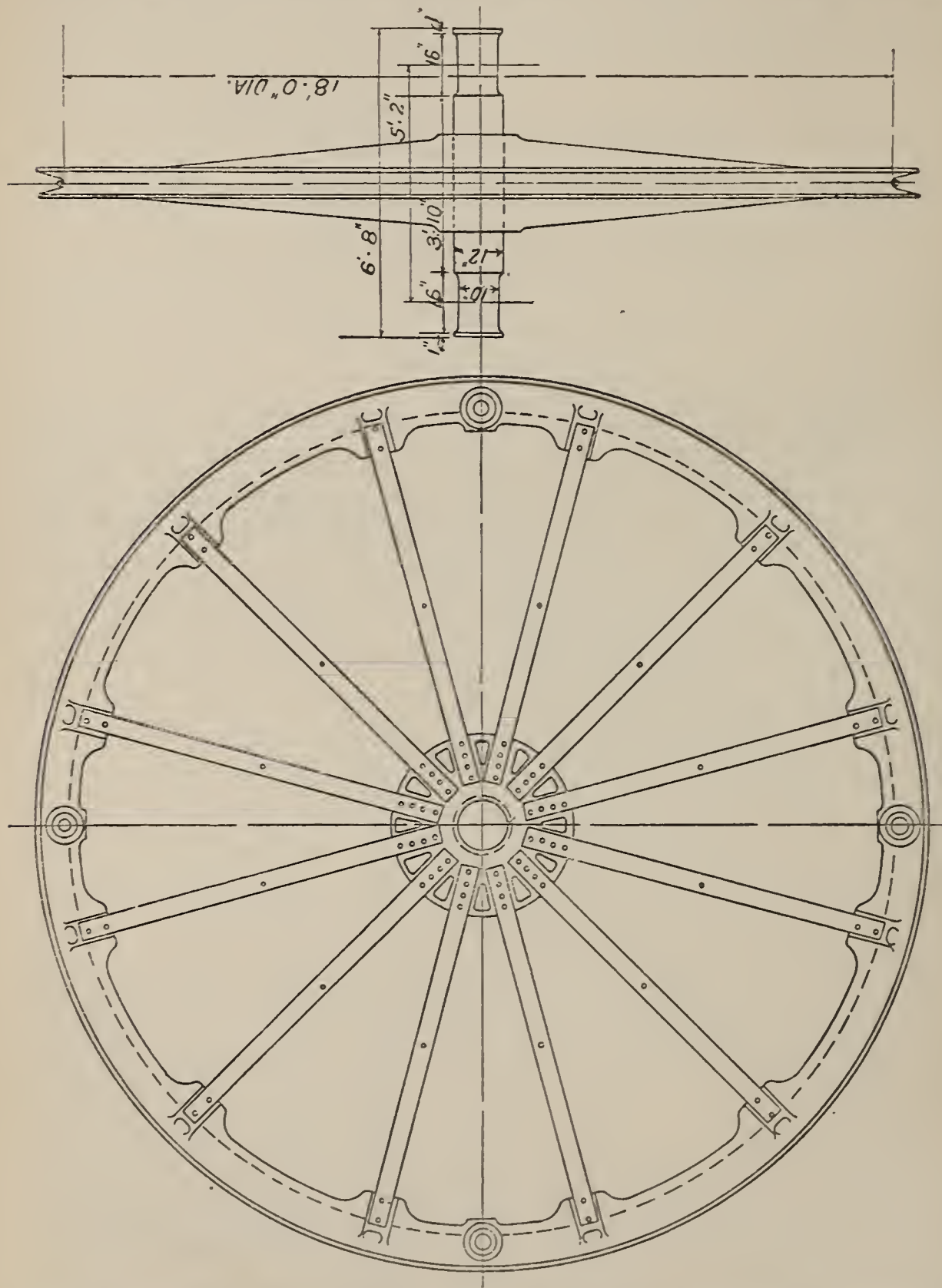


FIG. 362.—Tomson's system of decking cages.

Pit-head Pulleys.—Pulleys used on pit-head frames are usually made either of cast or wrought iron, or a combination of both, the rim and bosses being cast and the spokes constructed of wrought iron. The shape of the groove varies with the shape of the rope used, and the groove should be quite smooth and of such a section that one-third the circumference of the rope is supported on the tread. The flanges, however, should be set at such an angle as will pre-

* *Haulage and Winding Appliances used in Mines*, by Carl Volk, p. 44.

vent contact between them and the rope at all parts of the wind and during the travel of the rope across the drum. The pulleys should be as accurately set as possible to give the rope a straight



Figs. 363 and 364. — Pit-head pulley.

lead to the drum and to obviate as much as possible side friction. The pulleys should also be made as light as possible, compatible with strength, for if they are too heavy much frictional wear takes place both on rope and pulley, due to overcoming the inertia of the pulley when at rest and checking the momentum when completing

the wind. In view of this it is not desirable to have pulley more than 16 to 18 ft. diameter under any circumstances. If the pulleys are too heavy, it is not uncommon for them to continue revolving after the completion of the wind, causing severe and unnecessary frictional wear of ropes and pulleys. The size of pulleys will vary between 6 and 18 ft. in diameter, according to the size of rope and drum used. Where wire ropes are used they should be as large as possible to avoid straining the rope by too sharp a bend, a common size being between 12 to 16 ft. in diameter. The diameter should however, be in proportion to that of the rope used.

For a steel winding rope 1 in. diameter, a pulley not less than 10 ft. in diameter is required, and for ropes from $1\frac{1}{4}$ in. to $1\frac{3}{4}$ in., pulleys should be 15 to 18 ft. in diameter, but, as a rule, they ought to be about the same size as the winding drum.

Drums.—Winding drums are either cylindrical, conical, or spiral in shape, the first-named being the most common, and being usually constructed of two cast-iron cheeks fitted or keyed on to a wrought-iron shaft, and lagged between the two cheeks with strong wood deals for the rope to coil on. Sometimes they are wholly constructed of wrought iron or steel, but it is a mistake to have ropes coiling on such a hard surface as iron or steel, as it sets up excessive friction and wear. At a colliery in Scotland two winding ropes were used on a steel-clad drum; one rope gave fifty-two weeks' life and the other sixty-six weeks. Wood lagging was put on the drums, and the ropes then lasted upwards of five years. From an economical point alone, wood-clad drums are to be preferred. Conical drums should have a spiral groove running round them, from the shorter to the longer diameter, in order to keep the rope from slipping. A spiral groove should also be used on an ordinary cylindrical drum to prevent excessive side friction between the coils of rope when winding, which will increase its life (see fig. 365). The pitch of the groove should be such as to have a space of $\frac{3}{32}$ to $\frac{1}{8}$ in. between the coils of the rope so as to prevent lateral friction. Drums ought to be made as light as possible so as to minimise the inertia at starting. Good, strong, and at the same time light drums are best made of steel. Where high winding speed is necessary, the winding

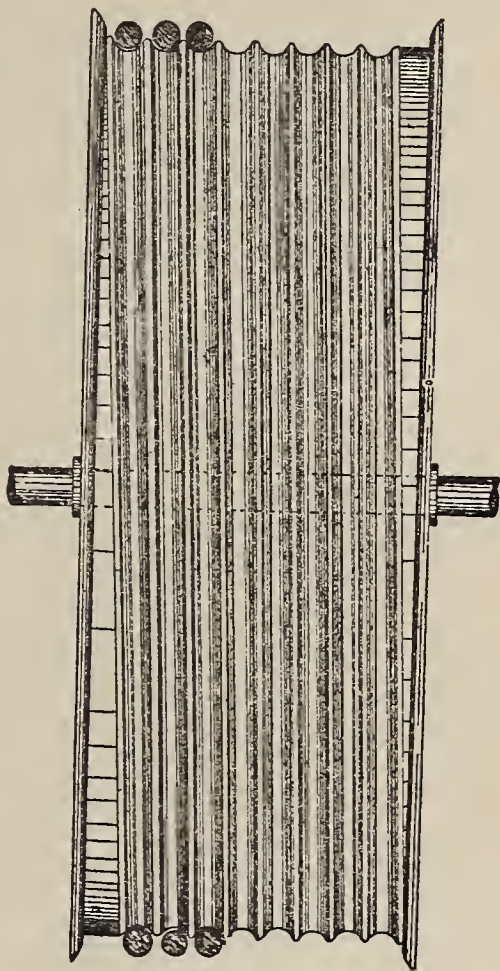


FIG. 365.—Drum.

drum should be made as light as its mechanical strength will permit.

The size of drum used will greatly depend on the size of engine erected for winding and upon the size of the rope, but it is a mistake to use very large drums, as large diameters decrease the efficiency of the engine. In winding, speed requires to be got up quickly each wind, and with large drums this will not be so easily attained as with drums of smaller diameter.

Various rules are given for the size of the drum, but a minimum diameter is 100 times the diameter of the rope or 1000 times the diameter of the individual wires in the rope. The following sizes are frequently used for winding drums :—

- 120 to 140 times diameter of rope (patent steel) = diameter of drum.
- 140 to 160 times diameter of rope (plough steel) = diameter of drum.

The size of the drum should, however, have some relation to the number of wires in the rope or in the strand. One maker of ropes gives the following :—

- 6 strands of 7 wires each, rope 1 in. diameter, drum 150 in. diameter ;
- 6 strands of 12 wires each, rope 1 in. diameter, drum 115 in. diameter ;
- 6 strands of 19 wires each, rope 1 in. diameter, drum 90 in. diameter.

The following rule may also be taken as a fairly approximate guide :—

Diameter of Rope in Inches.	Weight of Rope in lbs. per Fathom.	Diameter of Drum in Feet.
$1\frac{1}{8}$	10	12
$1\frac{3}{8}$	12	13
$1\frac{5}{8}$	14	14
$1\frac{7}{8}$ (fully)	16	15

Or the size of drum may approximately be found from the formula

$$D = 10 + (c - 1)2.$$

Where C = circumference of rope in inches,
D = diameter of drum in feet.

Size of Conical Drums.—It is difficult to construct this type of drum the right size to obtain the proper balance ; in fact, to construct a properly balanced conical drum is difficult mechanically and uneconomical. The dimensions for such drums will require to be calculated to suit the conditions of load, depth, etc., of each individual case.

Drum Brakes.—It is desirable that a good reliable and efficient brake should be attached to the drum of all powerful high-speed winding engines. The power should be applied as near to the centre of the drum as possible, and a large leverage, from 150 to 200 to 1, given. The ordinary block brake is very efficient, and is largely used (fig. 368). In this brake two blocks of wood, generally elm, are brought into contact with an iron ring fixed either on the centre or side of the drum, and the power applied by an arrangement of levers that can be worked either by hand or foot by the engineman.

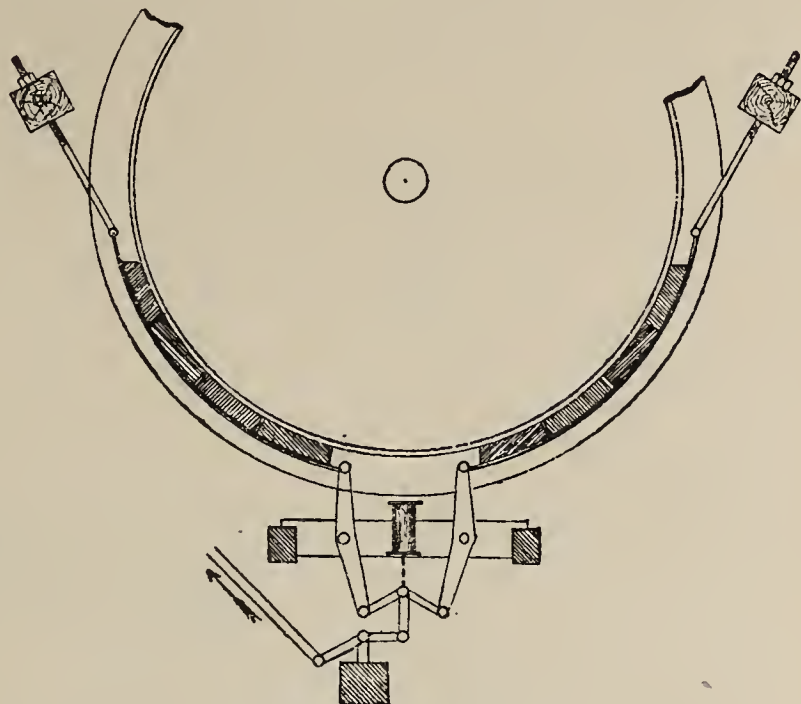


FIG. 366.—Brake device.

The type of brake shown in fig. 366 is much used in some districts of England. It is one which is arranged underneath the drum, and there is little friction when the engine is at work, as when released it immediately frees itself of all contact with the brake 'ring.'

This brake is applied by a toggle-joint arrangement, and is arranged so that it can be worked either by hand or steam power.

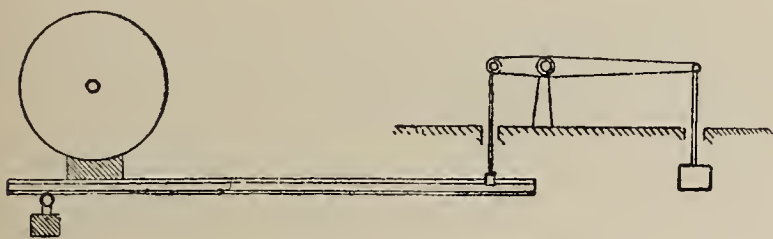


FIG. 367.—Burns's brake.

the advantage of being simple in construction, is shown in fig. 367. In this brake a single block of wood is fixed to a long lever and applied to the bottom of the drum, the leverage being in the ratio of 200 to 1. In the block holes are sometimes bored and filled with sand, which renders the brake much more effective.

Whitmore Automatic Adjustment Brake.—This brake can be applied to any existing brake, either of the post or band type. It reduces to a minimum all undue strains on the winding engine, and any load can be applied ranging from zero to full load, with the greatest certainty. It is designed to be powerful enough to hold the winding engine against full steam on the piston when the throttle

It should be made from well-seasoned blocks of elm or oak wood.

Another form of brake—known as Burns's brake—one that gives good results, is very powerful, and has also

is open, and arranged so that any desired braking force can be applied, and so dispense with the necessity for having two brakes, as are often supplied with large winding engines, one being a foot brake for applying ordinary light loads for pulling up the engine at the end of each wind, the other a steam brake applied in case of emergency.

The advantages claimed for it are that any brake load can be applied between maximum and minimum with certainty, and the load always corresponds to the position of the hand or foot lever, *i.e.* if half-load is applied when the lever is half-way over, it practically remains so always. This important advantage is arrived at owing to the wear on the brakes being taken up automatically. This taking up the wear is an improvement in itself, the take-up continuing until the blocks are worn entirely out.

Figs. 368, 369 show the usual type of post brakes with Whitmore Brake Engine and Gear applied. The brake engine A consists of a steam cylinder and cataract cylinder, this engine being fitted with a floating lever gear by which the position of the piston corresponds exactly to the position of the hand lever.

Weights B are suspended by rod C from the crosshead of the brake engine. These weights put on the brake; the steam applied under the piston raises the weights and takes off the brakes. With this arrangement, should any accident occur in the steam main or connections to steam cylinder or brake engine, the brake would go on, *i.e.* it does not depend on steam to put on the brakes.

Rod C is threaded through a spring box D which is free to slip up and down. This spring box bears upon brake lever E, and a spring, or springs, F, contained in this box, are compressed between the bottom of the box and plate G.

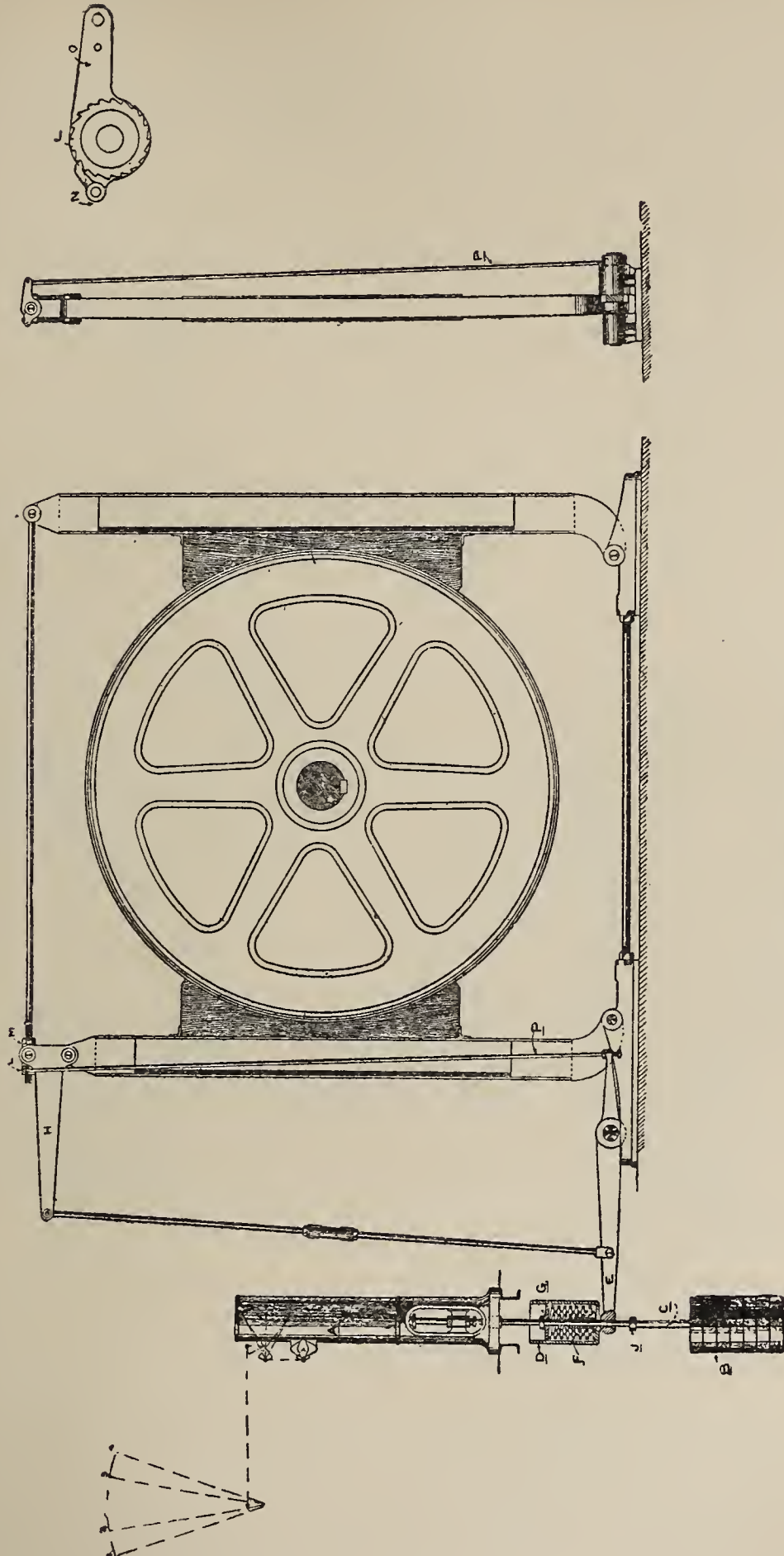
Fig. 368 shows the brake with about half-load applied. The brake lever E is connected to the top ends of the brake posts, drawing them together. The varying load is applied by the compression of the springs, the weight bringing down with them the top plate G, the position of each being controlled by the hand lever. The further the hand lever goes over, the further do the weights and plate move downwards, and so compress the spring more.

The maximum load is applied when the plate G is touching the sleeve distance piece in the spring box, and the minimum load is applied when the collar J is touching the underside of brake lever E, the springs being then fully extended.

The position of the hand lever is then at about point 2, and in bringing this back to point 1, the piston travels up still further, carrying with it the weights and fixed collar J and, consequently, the brake lever E sufficiently far to give clearance between drum and posts.

At the end of rod K there is fitted a ratchet nut L, fig. 370, mounted in the crosshead M of upper brake lever H. This

ratchet nut L has fitted to it a pawl N mounted in a lever O.



Figs. 368, 369, and 370.

This lever O is connected by rod P to an extended arm of the brake lever E, and comprises the take-up gear, which acts as

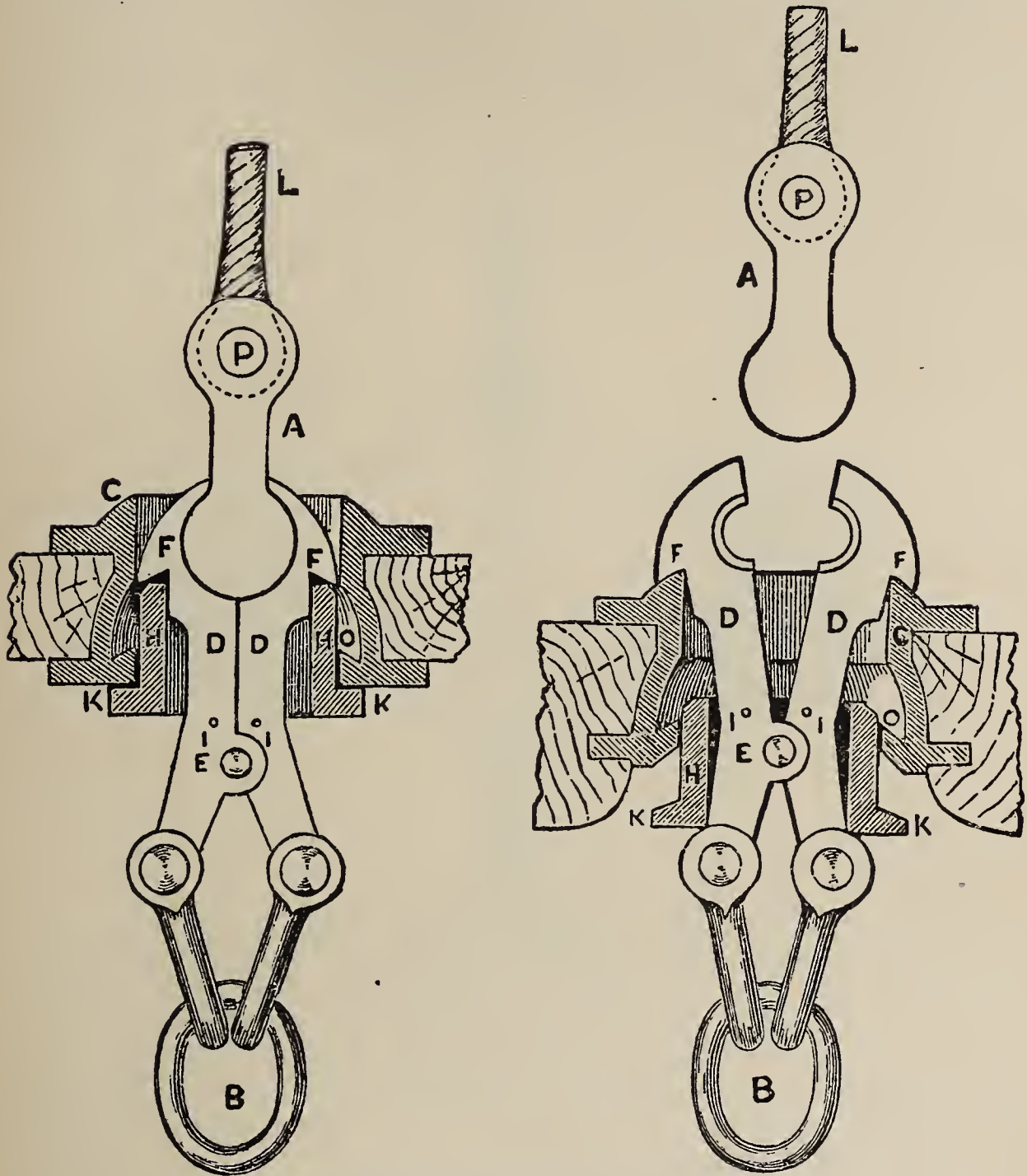
follows:—On the downward stroke of brake lever E the rod P is moved upwards. When the lever and rod E are forced down to a certain distance, pawl N will take up another tooth. On the return or upward stroke of lever E, the rod P will be brought downwards by lever coming in contact with the collar on bottom of rod P and so screwing up nut to that extent. It will be seen that the lever will not be forced down to the position before mentioned by the same load being applied until the brakes have worn that amount. The adjustment therefore is very slight at each operation, so that there is only a very slight difference in the position of brake piston immediately before and after each automatic adjustment.

Safety Hooks.—In no class of work about a colliery is there more liability to accident than in winding, and yet such accidents are happily rare, no doubt owing, in a large measure, to the careful handling of the engines by those in charge. When it is considered, however, at what speed they have to be worked, and the number of times the cages have to be raised and lowered even in the course of one shift, it is obvious that an accident due to overwinding may easily occur even with the most careful engineman, as some portion of the engine may get beyond control, and prevent it from being stopped at the proper position. It was to obviate the effects of overwinding that safety hooks, which have been applied with much success, were invented. Many colliery owners do not, however, use these hooks or other safety appliances, because they are likely to get out of order and not act when required, and more confidence is placed in having good reliable men at the engines than in any mechanical contrivance. There have, doubtless, been cases where safety hooks have not fulfilled expectations, and where they have even broken when an overwind took place. But this may happen with any piece of machinery, particularly if it is not properly looked after and kept in good working order. If, on the other hand, the use of safety hooks renders men's lives safer in the event of an overwind, there is no good reason why they should not be adopted at all collieries. The Coal Mines Act, 1911 (section 40, sub-section 2), makes it compulsory for the winding apparatus in all vertical shafts to be fitted with a detaching hook.

There is a large variety of safety hooks before the public, but the principle upon which they act is practically the same in all. In some the rope is simply released when the cage is overwound, and in others the rope is released and the cage held fast simultaneously.

Walker's Detaching Hook.—This is one of the most efficient safety hooks in use. Its principle will be understood from figs. 371, 372. The hook consists of a pair of jaws D D working on a centre pin. These jaws are held together and made to retain the strong action bolt in the rope shackle A by means of the clamp K, which is kept in position by the copper pins I, and the outward pressure due to the weight of the load. In the event of an overwind, the jaws pass

freely into the ring C, which is a fixture, but the flanges K of the clamp H coming into contact with the ring C; as in fig. 371, is held stationary while the jaws are pulled through, with the result that the copper pins I are sheared off, and the hook jaws F F are forced open by their lower portions being drawn into the clamp, in which position



FIGS. 371 and 372.—Walker's detaching hook.

they are firmly locked, as shown in fig. 371; the rope then passes over the pulley, and the load remains suspended. .

This hook being made without side plates, is not liable to get fast, is simple in construction, and can be quickly and easily re-connected.

West's Hook.—This hook is also simple in arrangement, and is composed of the body A (see fig. 373) and two sliding catches B and B', fitted with a copper releasing pin C and a locking bolt D.

When an overwind takes place, the wedge-shaped portions of the sides $B B'$ come in contact with a fixed plate E , and are forced into the outer steel box A , whilst the opposite ends are forced out as in the figure, allowing the shackle and pin to be liberated and held suspended on the plate E .

King and Humble's Hook.—This hook consists of two outer plates $a a$ and two inner plates, all of which are pivoted upon a strong centre pin b (see figs. 374, 375). The winding rope is attached to the top shackle d , and the cage and chains to the bottom shackle e .

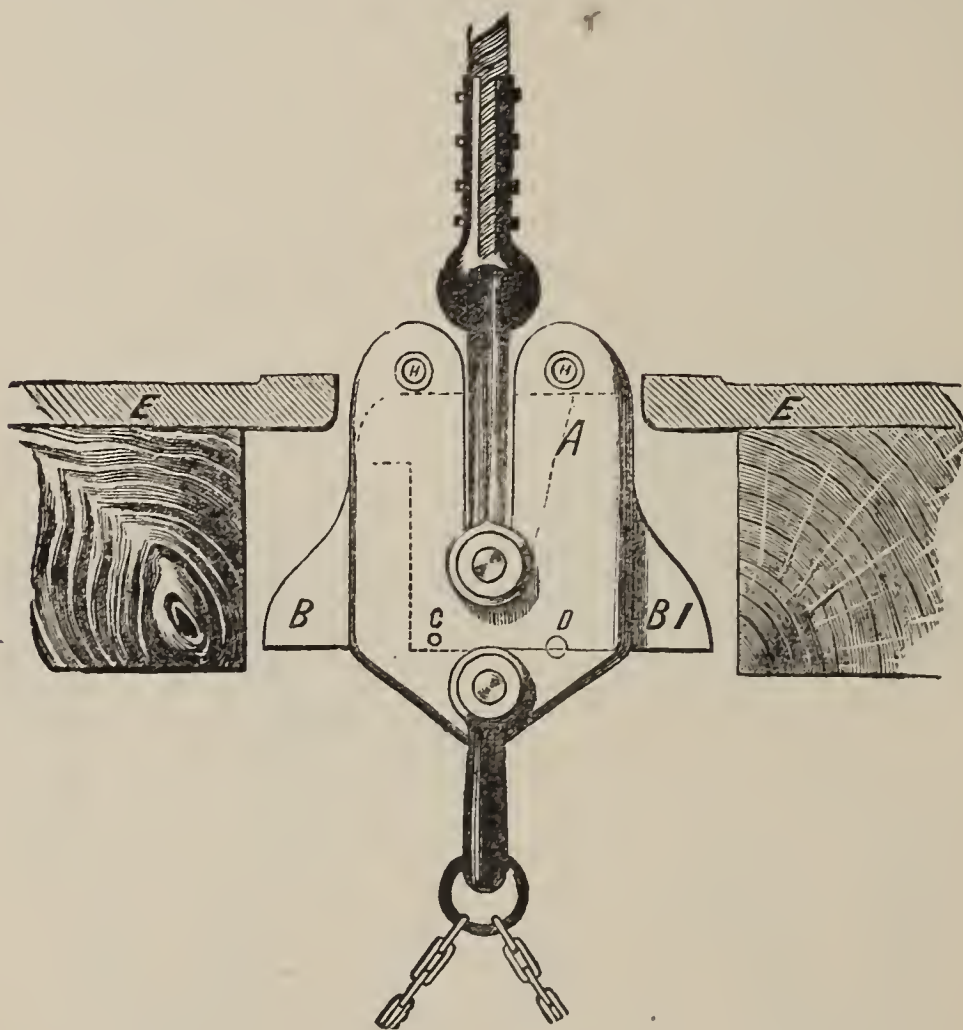


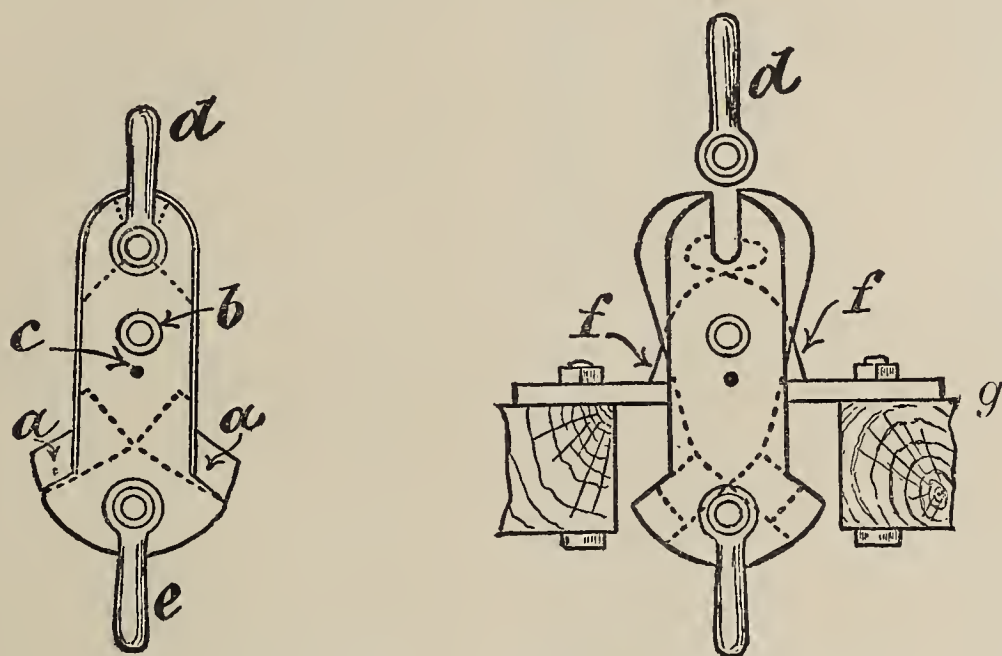
FIG. 373.—West's hook.

The wrought-iron catch-plate g , through the centre of which passes the winding rope, is securely fixed to the head frame immediately under the pulley wheel.

In the case of an overwind, the hook is partially drawn through the centre hole in the catch-plate, until the bottom jaws of the inner plates of the hook come in contact with the underside of the catch-plate, when they are pressed inwards, shearing the copper pin c , causing, by the same action, the upper jaws to extend, thus releasing the rope, and, at the same moment, the hook locks upon the catch-plate. The latter is so constructed that there is just sufficient space between the lower jaws and the locking jaws for the catch-plate to insert itself, hence the hook cannot be sufficiently detached through

the catch-plate to allow the locking jaws to get on the top side of the plate. As soon as the under jaws are forced out, the hook is therefore locked on the upper side of the catch-plate. King and Humble's hook is also furnished, in case of an overwind, with an automatic lowering arrangement which consists of an elongated slot just above the centre pin. When an overwind occurs, the rope is brought back over the pulley to the hook, for which a spare shackle is provided. This is passed through the rope shackle and down over the hook to the lowering slot, whereupon the rope is slightly tightened, which causes the inner plates of the hook to close, and the hook with the cage attached can now be lowered on to the pit keps.

Safety Cages.—Safety hooks, such as those described above, are meant only to prevent accidents in cases of overwinding, and afford



FIGS. 374 and 375.—King and Humble's hook.

no security against accidents resulting from the rope breaking while the cage is running in the shaft. To guard against this, innumerable safety cages and appliances have been invented, although few of them have proved to be of any real value in practical working.

On the Continent, as in Germany, the use of safety cages is enforced by law; in Britain, however, such appliances are not compulsory, and among colliery proprietors, at least in their present form, find but little favour. Most of them depend for their action on a grip or spring which ordinarily is not in contact with the guides, but which, in the event of the rope breaking, is released, and clutches them in order to prevent the cage from falling. While they may be of some use where winding is carried on at low speed, they are practically useless at most modern collieries where the speed is often very high. In such cases they often fail to act on an emergency, or allow the cage to fall back with such velocity that the guides are greatly damaged or even broken, and the cage is precipitated to the bottom

of the shaft. If men are in it, the shock is likely to be so great as to either pitch them out or dash them against the top of the cage.

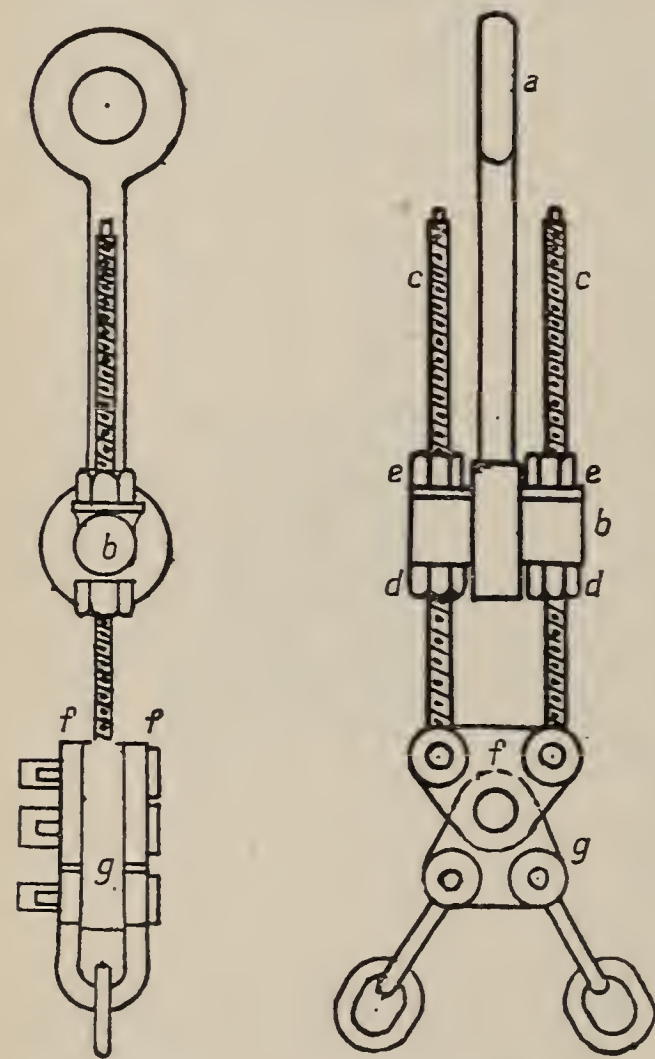
A serious accident occurred in a mining district in Germany, with one of these protected cages, supplied with safety grips and a controlling lever worked from the cage itself.

Notwithstanding these precautions, and the fact that everything was in working order, the appliances proved useless, and gave way, with the result that the nine men in the cage were killed.

The best preventive against such accidents occurring is to use only the best quality of winding ropes, to give them careful treatment, and to inspect them frequently.

Adjusting Screws.—In ordinary practice the length of winding ropes is adjusted by increasing or diminishing the spare coils on the drum and by refixing them or cutting a portion off and recapping. By such methods it is, however, very difficult to secure exact adjustment without much labour and care. To obviate this, adjusting screws have been applied. Figs. 376, 377 show the construction of these screws.

They consist of a strong steel rod *a*, terminating at each end in an eye. The shackle at the lower end of the rope is attached to the upper eye. A round block *b*, with a hole in each end large enough to admit of the easy passage through it of



FIGS. 376 and 377.—Adjusting screws for winding ropes.

the screws *c c*, is placed in the lower eye. Screws *c*, $2\frac{1}{2}$ or 3 ft. long, with strong threads and an eye at their lower end, and provided with nuts *d*, screwed on to them, are passed through each hole in the block, and the nuts *e* are then screwed on to them from above. Each nut *e*, while resting on the block, supports its own screw. Two triangular plates *ff*, with a hole at each angle, are attached, by means of pins passing through two of their holes, to the eyes at the lower ends of the screws, which they then enclose between them. The third eye in each of these plates hangs vertically below the steel rod which supports the block. A third triangular plate *g*, with three holes, one at each angle, is inserted between the two first, and a

pin is passed through one of these holes and through the unoccupied holes in the two plates above it. Two short pieces of chain are attached to the remaining holes in the lower triangular plate by means of shackles, the cage being attached to these chains. The ropes can be adjusted in a very short time, without much labour, by means of these screws.*

Cage Props or Keps.—‘Keps’ or ‘props’ are required at most collieries, as a rest for the cage and to keep it in position during the changing of the tubs. Until the enactment of the Coal Mines Act, 1911 (section 40, sub-section 4), probably 80 per cent. of the collieries in Scotland used no keps, the engineman maintaining the cage in position by applying the brake to the winding drum until the tubs are changed, by which means the ropes are said to last longer. It is also claimed that there is less liability to accident, owing to the absence of ‘keps,’ which require to be opened and shut every time the cage comes to the surface. There can be no doubt but that the output from the cages can be handled with greater facility when no keps are used, at least that has been the experience at Scotch collieries, and, on the grounds of safety also, it is difficult to understand why their use has been made compulsory. The writer, in all his extended experience, never knew of an accident taking place as the result of keps not being used, but he has known of several where accidents were directly caused by the keps. The ordinary form of ‘keps’ usually consists of four legs pivoted at right angles to each other, and attached to a lever for opening or shutting them. They are generally allowed to swing out while the cage is running in the shaft, and are automatically opened by the cage itself, when it arrives at the surface, but they have to be opened by hand when the cage is about to make its descent. The commonest form of keps, made wholly of iron, is shown in fig. 378.

Stauss Keps.—In these keps (figs. 379 to 382), the invention of a German engineer, the cage is held firmly and securely in position when it arrives at the landing stage, and is released again for descent into the shaft, without the necessity of lifting a foot or two, as with the ordinary form, to allow of their being first drawn back, which often causes a sudden jerk or strain to be given to the slack rope.

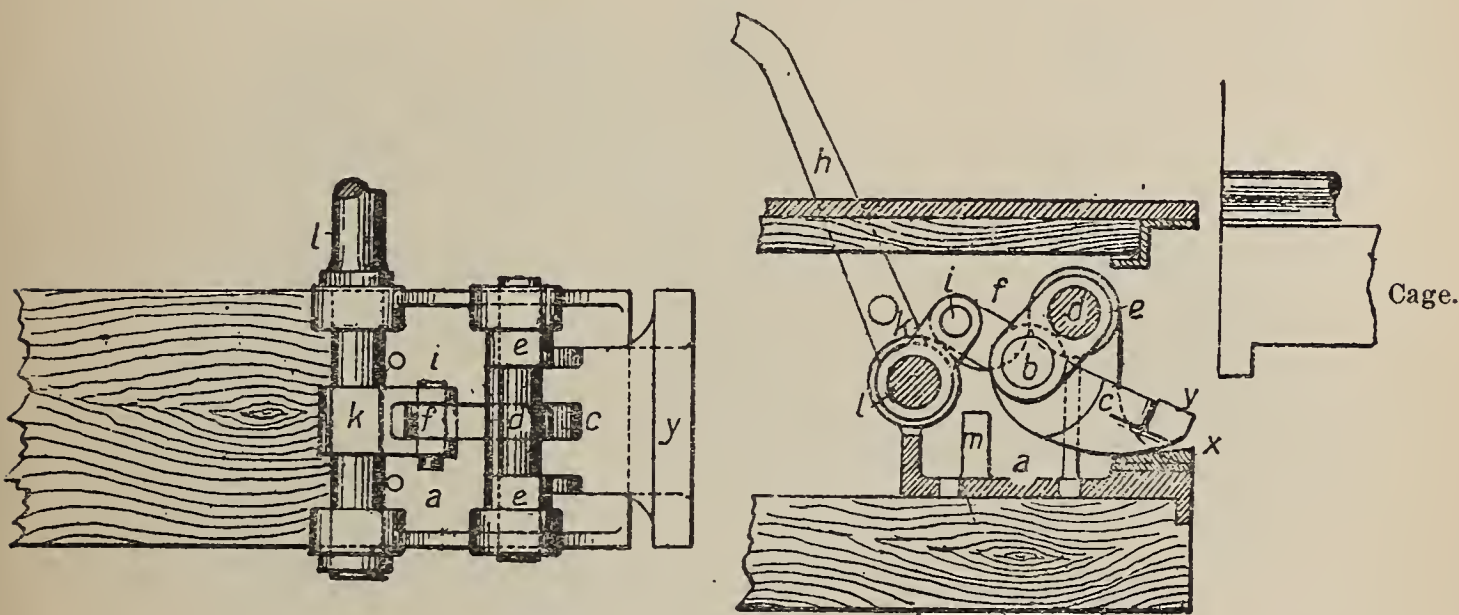
When the cage is to be held fixed, it rests upon the surface *y* of the catches or tappets *c*; these catches resting upon the part *x* and against the pin *b*, and being held fast both in a horizontal and in a vertical direction. The latter function is discharged by the bell-crank *e*, which presses against the shaft *d*; whilst horizontal movement is prevented by the crank *f*, which presses against the shaft *l*, through the bolt *i*, of the lever *k*. In this way the steadiness of the cage is secured, and displacement is prevented. The hand lever *h* presses the lever *k*, when in this position, down against the block *m*, which is fixed to a casting *a*, so that side play is also prevented.

* *Lectures on Mining*, by Prof. Wm. Galloway, p. 7.

When *k* is brought into the position shown in fig. 382, and indicated by dotted lines in fig. 381, *i* is brought to the position *i'*, and *b* to *b'*, by which cage means the catches are withdrawn from under the cage and are lowered at the same time, so that the latter can descend the shaft. When the cage again arrives at the pit-mouth, the hand lever *h* is pushed back into its first position, the catch *e* projects, and the cage is secured.

The advantages claimed for these keps are: simplicity of construction and working, and a saving in ropes and engine-power, the short jerks which injure the former in lifting the cage before it descends being done away with.

Hydraulic keps are used on the Continent, but they are apt to get out of order, and are not reliable, owing to their complicated nature and to the water freezing in the pipes in winter.



FIGS. 381 and 382.—Stauss keps, open.

Fig. 383 shows the construction of these keps. Instead of four rigid arms (as in ordinary keps) there are four short cylinders, *b b*, each provided with a stuffing-box and plunger *c*. Hinged to the top of each plunger are movable pieces *d d*, which take the place of the rigid arms in ordinary keps. The four movable pieces are connected together by means of rods *e e* and levers *f f*, and move inwards and outwards like the arms of ordinary keps. The cylinders are connected to each other by a pipe *g*, common to all four, which also communicates with the cylinder of an accumulator. There is a stop-cock on this pipe which can cut off communication with the accumulator. Suppose the four plungers to be in their highest position, and the stop-cock shut as in the figure. The loaded cage ascends the shaft, and reaching the surface pushes the four movable pieces *d d* aside, passing up between them. The latter immediately fall back, and the edge is lowered on to and arrested by them, and the liquid in the cylinders, having no outlet, prevents

the plungers from descending. When the full tubs have been replaced by empty ones, the stop-cock is opened, allowing the liquid to pass into the accumulator until the movable pieces *d d* are clear of the cage.

Automatic Apparatus to Prevent Overwinding.—In the operation of winding, the engineman may misjudge either the speed of the engines or the distance the ascending cage is from the surface, with the result that an overwind may take place. Or the engineman in a moment of forgetfulness may forget to shut off steam at the proper time, or he may be suddenly overcome with illness and be unable to control his engine. In either case a serious accident may ensue, especially if men are being raised on the ascending cage. To prevent such accidents it is now compulsory to fit all winding engines with automatic apparatus to prevent overwinds, if the

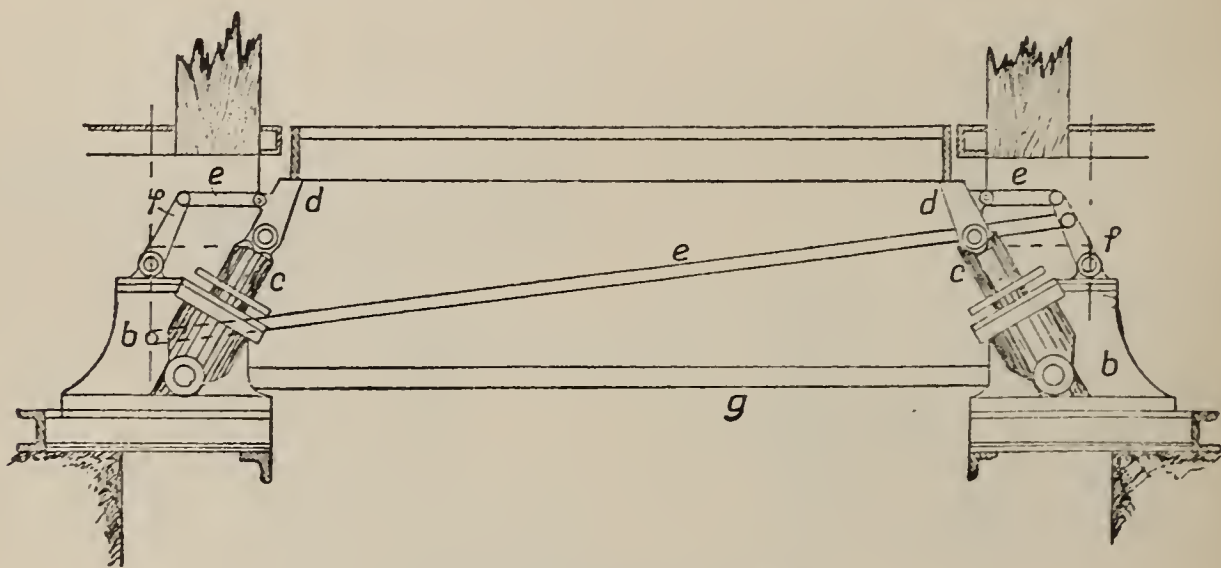


FIG. 383.—Hydraulic keps.

shaft is more than 100 yards in depth (see Coal Mines Act, 1911, section 40, sub-section 2). There are now a great many of such automatic contrivances on the market, and it would be impossible to describe all of them in a text-book. Besides, it has yet to be seen how many of these overwind gears will successfully meet the purpose for which they were designed. In the case of an overwind much damage is often done, which cannot be prevented even by the use of detaching safety hooks, the objects of which are to prevent the cage from falling back in the shaft. Appliances are also used to prevent loss of life if the cage happens to get overwound. In Germany every winding engine requires to be fitted with a steam brake which enables it to be brought to a standstill at once when required, even when going at full speed.

To make the brake self-acting, an hydraulic arrangement is sometimes used which the cage, when lifted too high, actuates itself. About half-way between the winding drums and the shaft, a pump, with reservoir and accumulator, supplies pressure to a length of

piping leading to the shaft; and further, to a return length of piping leading back to the steam brakes on the drum. A valve at the shaft permits the pressure from the accumulator to be carried further only when it is raised by a lever, which, in case of accident, the cage itself will actuate.

The pressure thus communicated to the return length of piping acts on a vertical cylinder working on the accumulator principle, and, by a rod and system of levers, may either act on the piston rod of the steam brake direct, or, by actuating the slide valves, admit the steam to the brake cylinder in the usual manner.

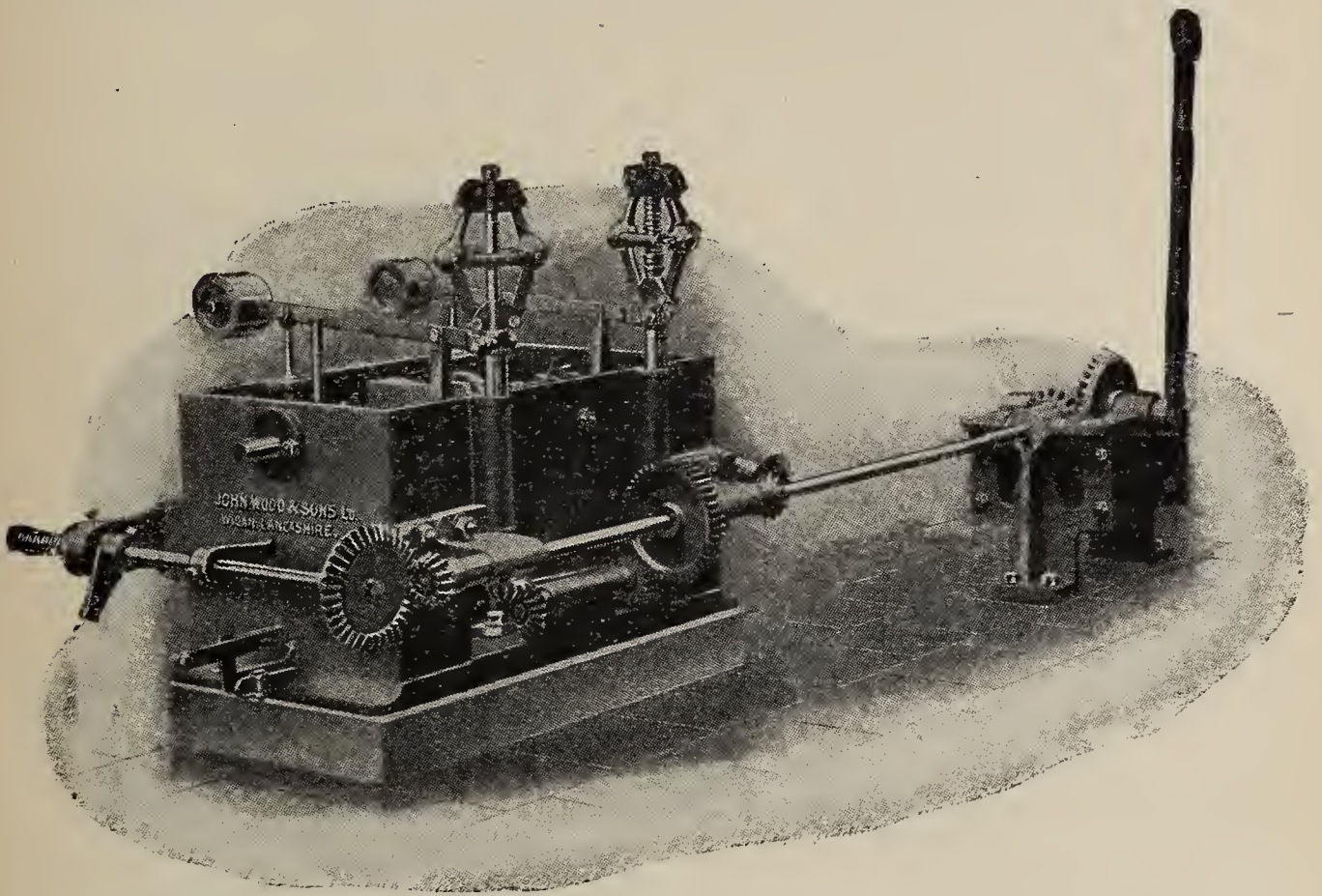


FIG. 384.—The 'Visor' controller for prevention of overwinding, general view showing outside working parts.

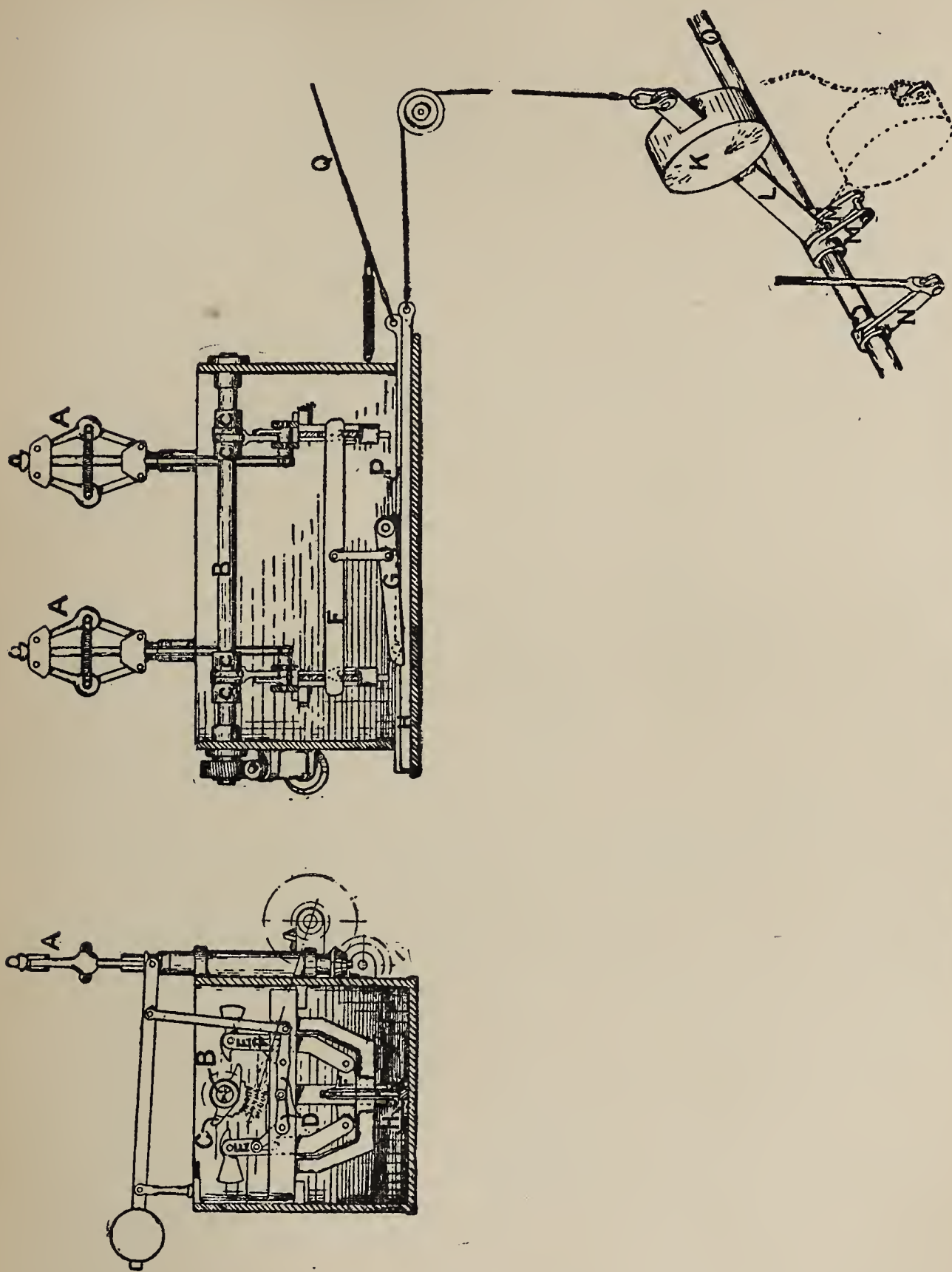
The Visor.—This apparatus is one of the first and most successful designed for the purpose of preventing overwinding, and has been in use at the pits of the Wigan Coal and Iron Co., Ltd., since 1888. Fig. 384 is a general view of the 'Visor.' The two governors are driven by the gear wheels shown, which are driven from the engine by a tail crank or drag link. The two governors run at different speeds, the mode of operation being as follows:—No. 1 governor is set so that if the engineman fails to reduce the speed of the engines at a predetermined point—say two to four revolutions from the banking level—the cams on the revolution shaft come into operation, shutting off the steam and applying the brakes. If speed has been reduced and No. 1 governor has allowed the cams to pass the nebs,

and the engineman does not continue to reduce speed, No. 2 governor will come into operation at a point, say, one to two revolutions from the pit top. By this the speed of the engines is reduced to an extent that renders it almost impossible for an overwind to occur; but should the engines continue to revolve, another arrangement fixed in the headgear in the path of the cages brings the 'Visor' into operation and stops the engines. This latter device has been arranged more particularly to prevent an accident should the engineman by an oversight neglect to reverse his engines, and, therefore, restart them in the wrong direction. This arrangement has been further improved. Instead of the lever being fixed in the headgear, a worm gear is attached to the 'Visor.' During a wind every part of the 'Visor' is working; therefore, there is no fear of it failing to act through sticking. It will be seen that the engineman can neither make a fast nor a slow overwind.

Figs. 385, 386 show the action of the 'Visor.' The governors A A, which act only as speed indicators, are driven by suitable gearing from the crank or drag crank-shaft of the winding engines, as also is the worm shaft B. The latter shaft makes six revolutions per wind, and carries the beaked cams C C, which are adjusted on the shaft to the exact required positions. As the speed of the engines increases, first one and then the other governor flies out quickly and throws inwards (through the medium of levers D) the vertical arms E E, with hooks attached, bringing the said hooks in line of contact with the beaked cams C C. When nearing the end of the wind, if the engines are brought to rest in the usual manner, the governors fall and bring back the hooks out of the line of contact. If, however, through any cause whatever steam is not shut off at the usual point, or the engines not slowed down, the hooks make contact with the beaked cams C C, and thus lift up the sliding frame and bar F. This is connected to the pawl G, which is thus lifted out of the notch in the bar H, releasing the latter, and as this holds weighted levers L in suspension, these immediately fall, and respectively close the starting valve and apply the brakes.

The arrangement of levers for closing the valves on applying the brakes is as follows:—Still referring to figs. 385, 386, O is the actuating shaft for the starting valve or steam brake (as the case may be), and is worked by the engine through the lever N. On this shaft are also keyed the double levers M M; between the same on the loose lever L is the weight K, which is held in suspension by a cord to the draw-bar H of the 'Visor.' When the engineman is working this shaft, the lever L is stationary, and the levers M M work up and down underneath it. When the weight K is released by the 'Visor,' it falls and forces the lever L on the connecting bar of the levers M M, which, being keyed on, turn the shaft and close the starting valve or apply the brake. Of course, two of these arrangements are necessary, *i.e.* one for the

starting valve and one for the brake. It will be noticed that in the 'Visor' there is a second draw-bar P, which is connected by



Figs. 385 and 386.—Sections showing action of the 'Visor.'

the wire cord Q to catches in the pit-head frame. These are fixed over the cages, so that should the engines be started in the wrong direction, this draw-bar P is drawn out, releasing the draw-bar H and stopping the engines.

Fig. 385 shows the method by which increase of speed at the end of the wind is prevented. After the speed of the engines at four or five revolutions from the end of the wind has been decreased sufficiently for the cam to pass the hook in connection with the slow governor, there is arranged in connection with the fast governor A a three-beaked cam C, secured to the worm shaft B. When the engines are, say, from one and a half to two revolutions from the end of the wind, should the speed not be sufficiently decreased, the hook E comes in contact with the first beak on cam C. This, of course, immediately releases the draw-bar H, and the engines are brought to a standstill. If the speed be slow enough to allow the cam C to pass the hook E, and steam be again applied so that the speed of the engines be increased, then the second of the beaks on cam C will be brought into the path of contact of the hook E, and the 'Visor' will be brought into operation. Should the speed of the engines be decreasing sufficiently to allow the first two beaks of this cam C to pass the hook, but not sufficiently to come to rest at the banking level, then the last of the two beaks would put the 'Visor' into operation just before the cage reaches the top. Therefore, it will be seen there are four points of cut-off before the cage reaches the surface.

Fig. 386 shows how the 'Visor' will trip in the event of the cage being started in the wrong direction. The small shaft G at the end of the 'Visor' box is driven by bevel gearing from the drum shaft, and the shaft C is also given a cross traverse. Secured to the shaft G are one or two long cams H, which at the end of the wind (owing to the traversing motion) respectively come to rest just before the point of contact of the draw-bar P. From this it can be seen that should the cage, after reaching banking-out level, still continue to ascend, the cam H will immediately draw the bar P and operate the 'Visor.' The same happens after banking should the engineman start the engine in the wrong direction. To show the quickness of the operation, it is only necessary to draw the bar P one-third of an inch. The radius of the cam H is 4 in., and, supposing the winding drum is 16 ft. diameter, the radius is 8 ft., or 96 in., then $96 : 4 = 24$ to 1 ratio, and one-eighth of 24 in. = 3 in. travel of cage to operate the 'Visor.' This cam, when the cage is at banking level, is set on the shaft so many inches away from P to represent the required distance from the plates at which it is desired to trip. This is an improvement on the old system of levers in the head-gear; owing to the quickness of travel of the cam H no time is lost in bringing the 'Visor' into operation, which is a very important point, seeing that the distance is limited between the bank level and head-gear pulleys in many cases, and the cam H is easily adjusted to compensate for stretching or recapping of ropes.

Fig. 387 shows a pair of winding engines fitted with the 'Visor' and steam brake. The inscription and lettering explain the

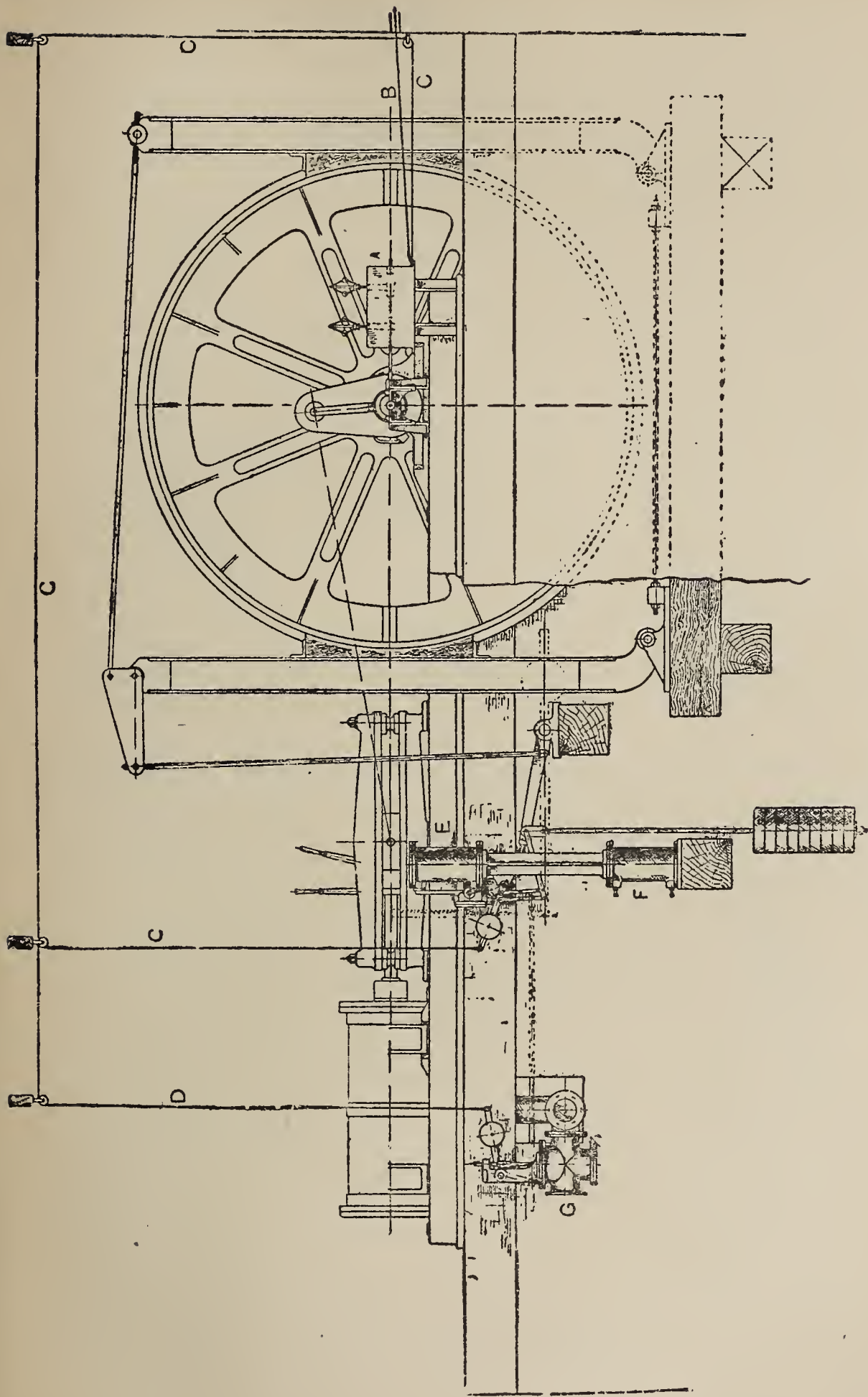


FIG. 387.—Arrangement of 'Visor' and steam brake.

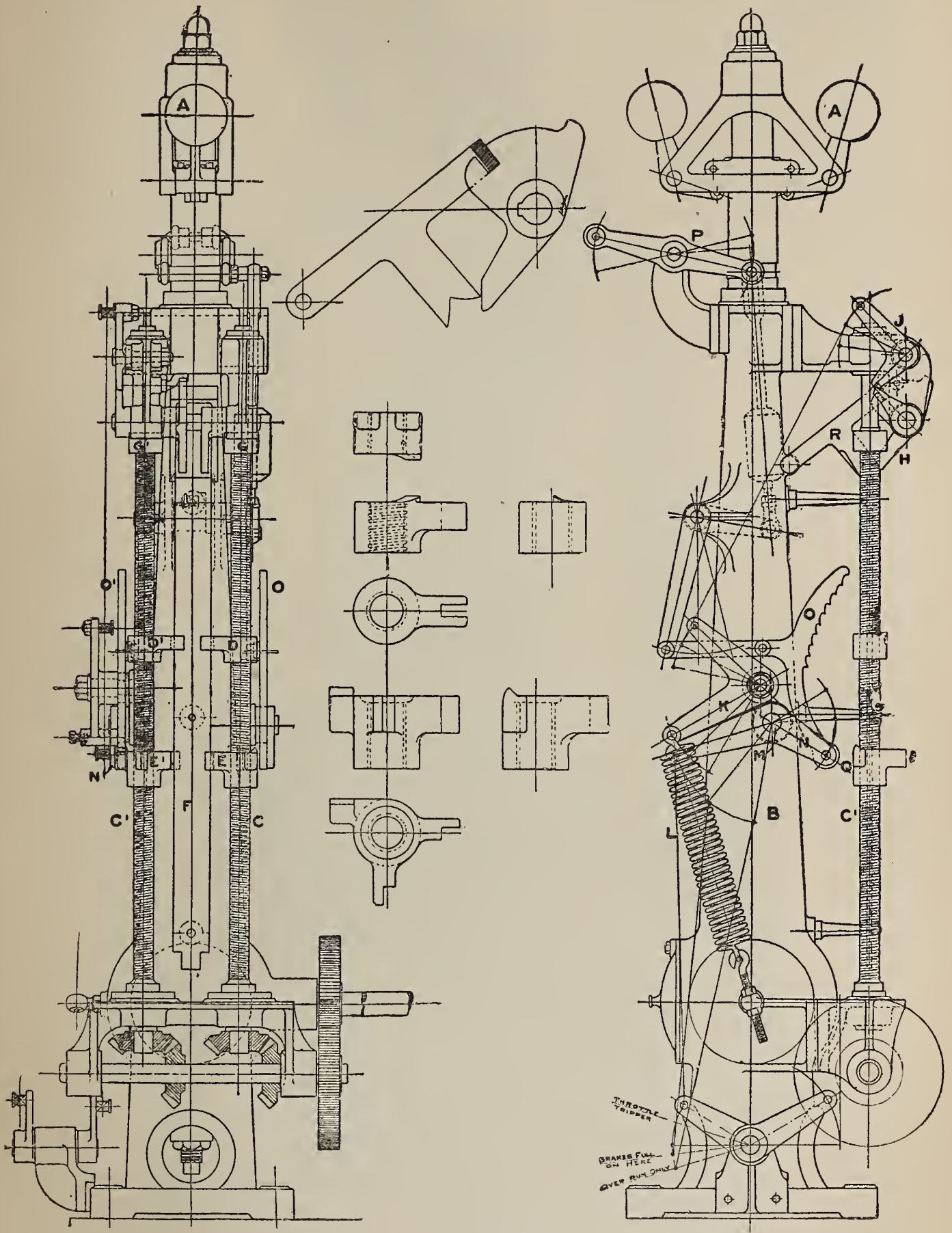
- A, Visor.
B, Cord to headgear.
C, Cord to brake attachment.
D, Cord to stop-valve attachment.
E, Steam cylinder of brake.
F, Dashpot cylinder.
G, Stop valve.

mechanical arrangements. The cylinder of the dashpot F is 2 ft. long and 12 in. in diameter. In this there are two taps, which can be regulated so that the brakes may be applied at will. This is done by adjusting the air pressure to suit the exact resistance necessary for the hanging weights. Opening the bottom tap gives a free passage for the air, which is forced down by the piston. When the 'Visor' is applied, the weight is released and falls on to a catch, thus not only shutting off steam from under the piston in the cylinder E, but also admitting it at the top end of the cylinder.

Whitmore Overspeed and Overwinding Gear.—The Whitmore overwinding gear is so arranged that should a cage rise above a certain predetermined point, the throttles are mechanically closed rapidly, and the brakes are mechanically applied rapidly. If the engine-driver does not close his throttle at a given period in the wind, or by any other cause such as running a light load he exceeds a certain predetermined maximum speed, the overwinding gear closes mechanically the throttle valve, or valves, rapidly, and gradually applies the brakes until the engine is pulled up, which is at a point before it reaches the end of the wind. The overwinder consists of two screws representing the wind of each rope. Where there are two ropes on one drum the screws are both driven from the drum. Where there are double drums, with one rope on each, screws are driven separately one from each drum. The gear consists also of a governor combined and mounted on the same column as the screws.

Figs. 388, 389 show the general arrangement of the standard overwinding gear as usually supplied for winding engines. The flyball governor A is mounted on column B, on which column are also mounted two screws—C and C¹, both screws and governor being driven from crankshaft or from drums. On each screw is fitted two nuts—D and E and D¹ and E¹. The upper nuts, D and D¹, control the engine when the cage or skip is above a predetermined point at pit bank, and it is done in the following manner. When nut D is going upwards, D¹ is going down, and *vice versa*. Now, when the cage is at the end of the wind one of these nuts, D or D¹, comes in contact with the fixed collar G or G¹, and in so doing the nut D rotates with the screw, the wing of which trips certain levers as shown in fig. 389, H, J, and K, due to the tension of the spring L, this lever being connected to a lever at the base of the governor, which in turn is connected to the throttle tripper and to the valve of the brake engine, with the result that once these levers are tripped the throttles are let go from the engine-driver's hand lever and immediately closed, and the steam brake similarly rapidly applied. In connection with this gear there is also a locking lever R, which is also an important feature. With such an arrangement this lever drops into a slot of lever H, and thereby locks

lever H in the adjusted position and prevents it from being tripped by any causes other than by the overwinder, and lever



FIGS. 388 and 389.

H cannot be tripped by the overwinder until the cage or skip is within a few feet of the bank. This prevents an accident

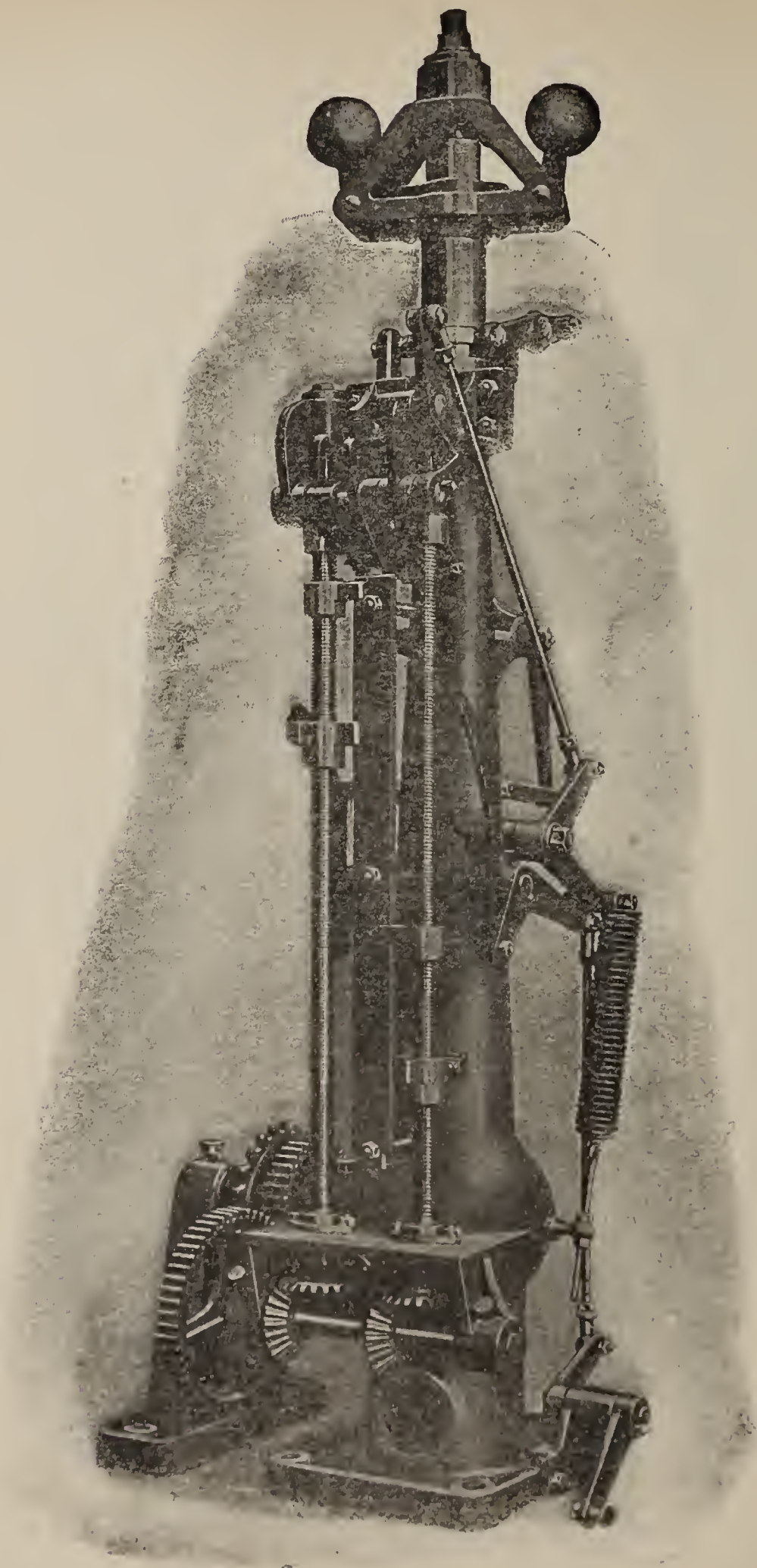


FIG. 390.

occurring should this mechanism be accidentally applied in the middle of a wind when the engine is at a high speed, as the brakes that these overwinders are fitted to are exceedingly powerful when fully applied. To prevent excessive speed when approaching the end of the wind, nuts *E* and *E*¹ are added, and act in the following manner: Pivoted to the column are levers *N*, on the ends of which are pivoted hook levers *O* and *O*¹, one for each screw. It will be seen by figs. 388 and 389 that as soon as the governor rises, hook lever *O* is thrown forward, and if it is sufficiently forward when nut *E* is passing up in front of it, one of the teeth will catch on this nut, and the nut will draw upwards with it lever *O*, and will slowly operate the lower lever and other mechanism in connection with the throttle and the brakes. The throttle is let go very soon after this lower lever has commenced to move, and once they are let go from the hand lever they close automatically, but the brakes are not full on until the lever has travelled practically the whole distance, and by this time the speed of the winding or hauling engine is so much reduced that the whole force of the brake can be applied without danger.

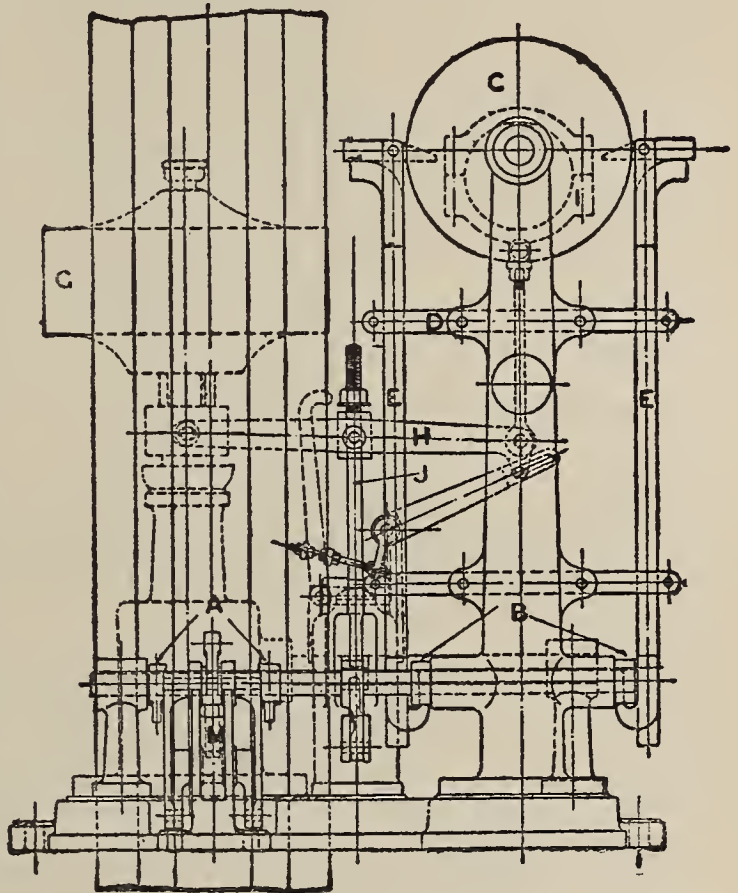


FIG. 391.

'Caledonia' Controller.—

This is another controller which has been recently introduced and fitted at a number of English and Scotch collieries by the makers, Messrs Andrew Barclay, Sons, & Co., Ltd., Kilmarnock.

The controller consists of the following parts mounted on one sole-plate, as shown in figs. 391, 392:—

A totally enclosed governor *G* of special design. A shaft *V* driven by a substantial chain and gearing from the drum shaft at such a rate that it revolves once, or almost once, while the cage moves up or down the pit shaft. An eccentric and rod *I* on the shaft *V* coupled to a floating lever *H*, the opposite end of which is controlled by the governor sleeve. Suitable standards for supporting these parts. A disc *C* carrying a pin, and revolving along with the shaft *V*, but capable of sliding on it. Connections between disc *D* and the throttle valve lever by means of *N* and *O*; a trigger *M* which, when

released, allows the rod P to descend. Suitable catch-rods E, which may be lifted by the pin on the disc D. These rods are furnished with folding hooks at the top, and fixed adjustable hooks at the

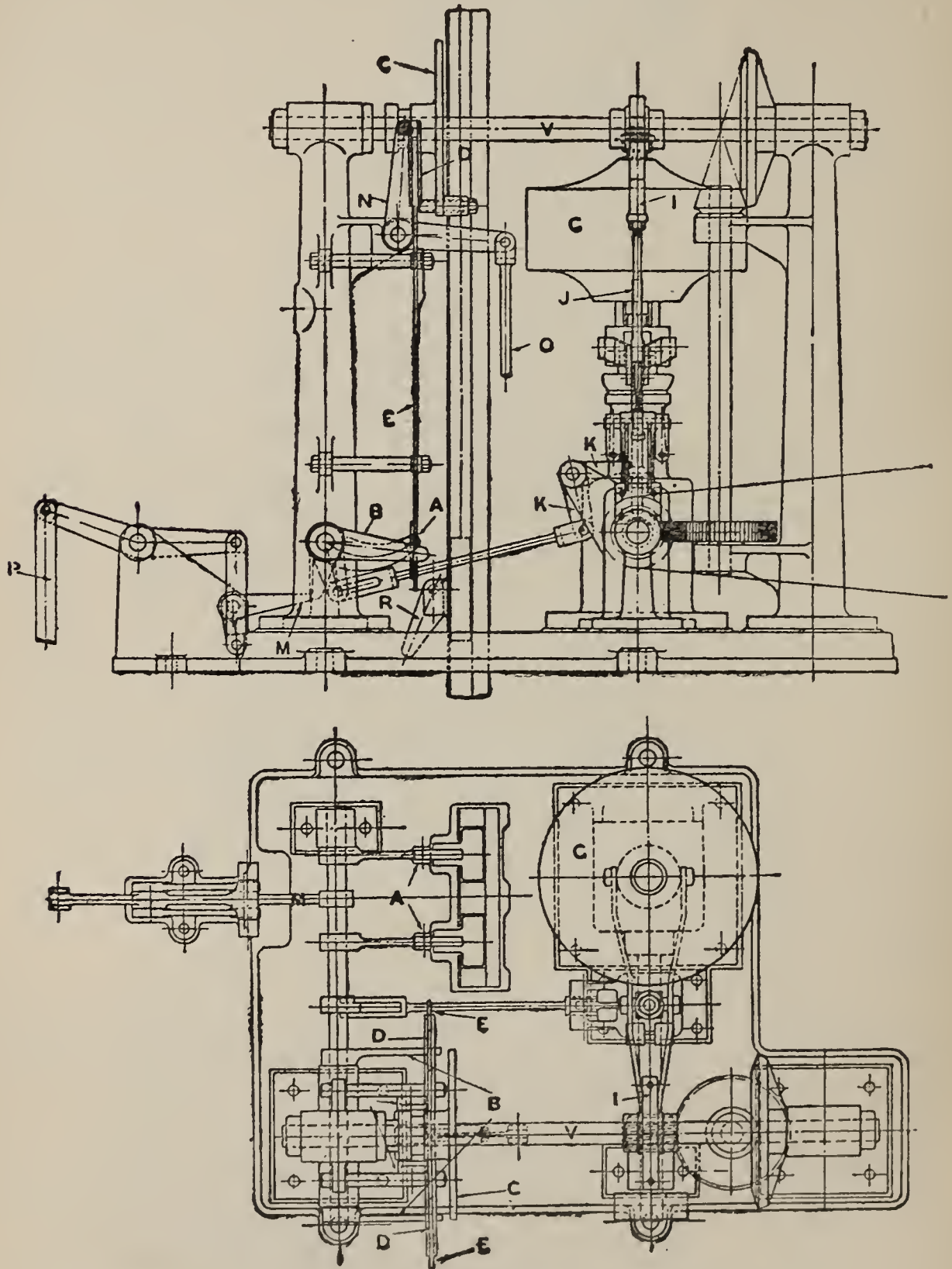


FIG. 392.

bottom. There is also provision for making electrical contact when either of the rods E are lifted. A trigger shaft which relieves the trigger at M when either of the levers on this shaft are turned round. A rod J which, when raised, rocks the bell crank lever K K and

pulls the attached rod. Two vertical guides, each of which carries a sliding hinged piece R, which is moved by a wire rope led to the pit-head, and connected thereto in such a way that during the last two or three feet of the motion of the cage, R is lifted upwards at the same speed as the cage.

This controller accomplishes the following :—

- (1) In the event of the engineman failing to shut off steam at the usual place, a warning bell rings, the steam is shut off immediately after the bell, the brake is put on, and the reversing handle is placed on the centre.
- (2) Should the cage approach the surface at too high a speed, the controller will shut off the steam, apply the brake, and place the reversing handle on the centre.
- (3) Should the cage, even though moving at a very low speed, pass the proper stopping place, the controller will bring the engines to rest by accomplishing the same actions as described above.
- (4) Should the engineman start in the wrong direction, the controller will immediately stop the engines.
- (5) The controller can be arranged to stop the engines at any point in the wind should the speed be excessive.
- (6) The controller entirely obviates the possibility of the cage dashing into the pit bottom.
- (7) Two ranges of speed may be provided, one for winding coal and one for winding men.

Action of the Controller.—At the commencement of a wind, when the engineman opens his throttle, the disc C (referring to the side elevation) is moved to the left, thus bringing the pin which it carries into position for engaging with the folding hooks on E. As the disc C revolves the pin will ultimately engage with one of these hooks, unless the throttle is shut, thus withdrawing the disc to the position it occupies on the side elevation, before the pin reaches the hook. The hooks are adjusted so that the pin will come into contact with the hook if the throttle is not shut at the usual time. If the pin raises one of the hooks, the rod E is lifted along with the hook, the first consequence of which is to make electrical contact and ring the alarm bell—not shown in the diagram. Immediately thereafter the fixed hook at the bottom of the rod E lifts up the lever B and withdraws the trigger M, allowing the rod P to descend. This puts on the brake, shuts off the steam, and centres the reversing handle.

If the cage moves too high even at a low speed, the piece R is lifted against the curved lever A and will rock the shaft and relieve the trigger, when the rod P will descend, putting the brake in action, shutting the throttle, and centring the reversing handle on as before.

When the cage is at the bank, the piece R will be approximately in the position shown, and if the engine is started in the wrong direction the piece R will at once be lifted and act on the curved lever A with the same effect as last described.

Referring to the end elevation, the eccentric I is shown in the position it will occupy at the middle of a wind. As the wind progresses from the middle towards the end, the right-hand end of the

lever H will be raised, and if the speed of the engine does not diminish sufficiently by a fall of the governor, the middle of the lever will be brought against the nut, which will raise the rod J and rock the bell crank K K, and through the attached rod will relieve the trigger at M, thus relieving the rod P and bringing the engine to rest as before.

The governor C is totally enclosed, the parts are working in an oil bath, and it will run for many weeks without lubrication. This governor is absolutely frictionless, consequently the sleeve has the same position for the same speed whether rising or falling.

The 'Caledonia' controller provides a full range of adjustment, so that it may be set for closing the throttle at practically any point in the wind, or a speed may be fixed beyond which it is not permissible for the engineman to work.

Under ordinary circumstances the controller will bring the cage to rest when winding slowly past the bank, or when starting in the wrong direction, in 2 to 3 feet.

When the controller comes into action by shutting off the throttle and applying the brakes when the engine is running fast, two to three revolutions of the drum usually take place before the engine is pulled up, but when the controller comes into action near the surface, the engine is pulled up in this case before the cage comes to the surface, and consequently before the descending cage reaches the bottom.

When the controller has acted it can be reset in a few seconds; no special tools are required, and no readjustment of the parts. The action of this controller is not affected by recapping the ropes or by stretching the ropes, nor by any alteration affecting the length of the ropes. The controller does not in any way interfere with the freedom of the engineman to follow up automatic braking by his hand- or foot- or steam-brake as the case may be, and it may be fitted advantageously to engines with steam-brakes, or with only manual brakes.

Electric Winders.—Within recent years a good deal of attention has been directed to the possibilities of utilising electric power for the winding of coal at mines.

Electric winding has been largely adopted on the Continent, especially in Germany, and also on the goldfields of the Transvaal. In Britain, however, this system has not been very largely used, although quite recently a number of electric winders have been installed at collieries in England and Scotland.

There is much difference of opinion among British engineers as to the advantages to be gained by the adoption of electric winding plant where good steam machinery is already installed, as it would mean the scrapping of what in the majority of cases is highly efficient machinery installed at considerable cost. From an economical point of view it is stated that the running cost of an electric winder is higher than a steam engine; and, further, there is little doubt but that an electric plant requires a much higher initial

capital expenditure than engines using steam direct. With the introduction of the low-pressure turbine for generating power for underground operations, the exhaust steam from the winding engine can now be utilised with great advantage. There are certain conditions, however, where it might be an advantage to install electric winding. For instance, if a new colliery was about to be opened out, a central generating plant could be laid down to generate current for all the operations about the mine, on the surface and underground. Or new shafts may require to be sunk at a considerable distance from the already existing shafts of a colliery, and it might be inconvenient to lay down a steam generating plant at such new shafts. In this case the electric power could be generated at the existing shafts and conveyed to the new shafts with economy. Again, at new shafts it might be possible to obtain electric current from a public supply company at a sufficiently low price to warrant its use and the installation of electric winders with economy. There are also many collieries, or groups of collieries, where coke oven gas could be utilised for raising steam at a central electric power plant with economy.

The whole question between electric and steam winders is very largely one of economy in working expenses and initial capital expenditure. Before one can definitely state whether electric winding will be economical and suitable it will be necessary to carefully consider each individual case upon its own merits.

There is no doubt that many mining engineers prefer steam winding engines owing to their simplicity in construction and working; and while electric winders may be quite suitable for moderate depths and outputs, it is doubtful if their application would be commercially successful to deal with large outputs from deep shafts, which are so common in the British coalfields, at least in the present transition stage of electric winders.

Types of Electric Winders.—The electric winders at present in use may be grouped under three heads, viz. * :—

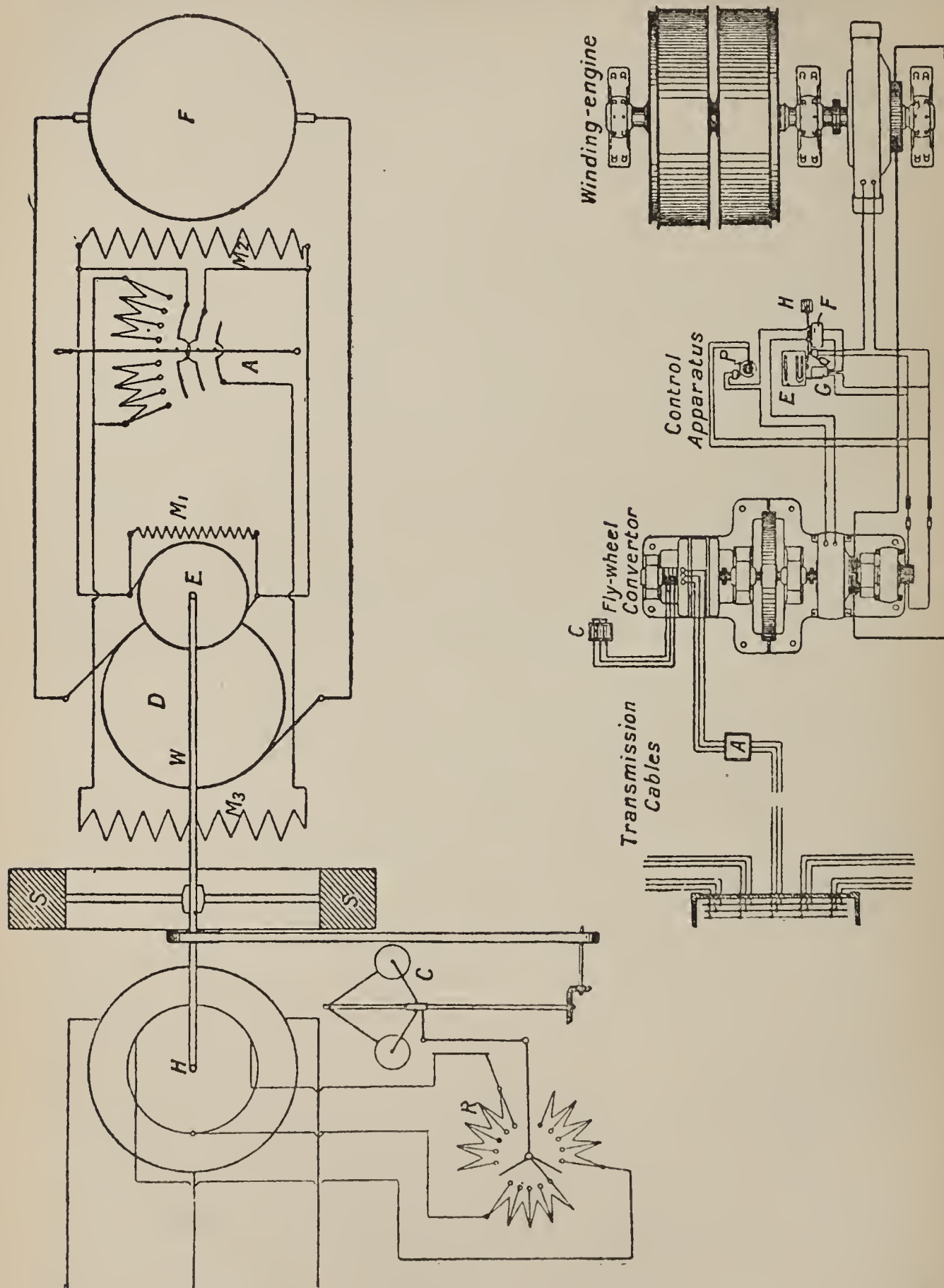
(1) The ordinary induction motor driving direct or through gearing, speed regulation being obtained by adjustment of resistance in the rotor circuit.

(2) The Ilgner system, which consists of a motor-generator with fly-wheel on the same spindle, the generator (dynamo) being coupled direct to the motor driving the winding drums, the motor and generator being both direct-current machines, and usually of the induction type. The fly-wheel acts as a reservoir of energy, and by automatically regulating the speed of the motor-generator set by means of resistance in the rotor circuit of the motor, the current or power demand from the supply mains may be kept within a predetermined limit.

(3) The Ward-Leonard system, which is practically the same as the Ilgner system, except that in the former there is no fly-wheel.

* *Iron and Coal Trades Review*, 2nd September 1910.

There are a number of other systems in use, chiefly modifications



Figs. 393 and 394.—Electric winding gear.

of the above, but the majority of the electric winders laid down have been constructed on the Ilgner and Ward-Leonard systems, the latter being considered the better type, as the control is easier and more definite, and the speed can be varied or maintained at

will for any load condition with greater efficiency. The Ward-Leonard control also lends itself better to automatic operation from the depth indicator.

Figs. 393, 394 * illustrate a winding engine constructed on the Ilgner system, by which the fluctuations of power required during winding are averaged by a converter placed between the winding engine and the generators. The shaft of the converter carries:— (1) The converter-motor H, fig. 393, with a horse-power equivalent to the average of that required for continuous winding, including pauses between the winds. This motor is driven off the main circuit, and may be wound for taking polyphase, alternating, or continuous current; (2) a heavy fly-wheel S; (3) a continuous-current dynamo D; and (4) a small continuous-current dynamo E. The winding-drum may be driven by one or two continuous-current motors, according to its size, but the principle is not affected in the latter, as both motors are supplied with current from the same source. The winding-motor F is driven by the generator D, the magnets of both D and F being excited by separate connections, M_2 and M_3 , from the small dynamo E. Before winding commences, the converter is run up to a high speed of rotation (600 to 700 revolutions a minute), sufficient current being supplied to H from the mains for this purpose, the same being regulated by the governor C and the resistance R. Until winding is commenced the excitation of the magnets of the generator D is switched out so that no current is supplied to the winding-motor F. The wind is started by moving the starting lever A to one side, this operation exciting the magnets of the generator D, and so sending current in one direction into the winding-motor F. The power, however, required at starting is considerably more than the converter-motor H is wound for producing, and in the ordinary course of events the speed of this motor would fall to nothing in attempting to supply it; this is prevented by the heavy fly-wheel S, which, revolving at a high speed, contains sufficient stored-up energy to keep the converter running at a speed little less than normal until the extra demand for power is past. On throwing the lever A over to the opposite side, near the end of a wind, the current in the magnets of the generator D, and accordingly to the commutator of the winding-motor F, is reversed; and the winding-motor F, until it stops revolving, acts as a generator to the generator D, and is braked in an effectual and economical manner. A portion of the power originally given out is thus returned to the converter, and assists the converter-motor H in re-accelerating the fly-wheel up to the full number of revolutions. The engine is very simply controlled by means of the lever A for starting, reversing, braking, and stopping, and by a compressed air brake for holding the engine between the winds.

* *The Colliery Guardian*, 19th May 1905.

The details of the control and switch apparatus are shown in fig. 394. A is a high-tension switch box; C, starting switch for converter-motor; E, lever frame; G, commutator; H, oil dashpot; F, starting gear; P, lifting arrangement for the safety brake, with emergency switch; and a regulating switch. By means of this arrangement the converter-motor (H, fig. 393) is supplied with an almost constant supply of power from the main circuit throughout, the fly-wheel supplying the extra quantity of energy required at the beginning of the wind, and this again being made up to it before the commencement of the next wind, partly by the converter-motor H, and partly by the power returned to the fly-wheel in braking the winding-motor.

Mr Francis Thursfield * states that the advantages claimed for electric winding are :—

- (1) The ability to run odd trips at any hour without the warming of the cylinders, etc.
- (2) More delicate control.
- (3) Less space is required for the winding plant; the motor being fixed on the top of the headgear if necessary.
- (4) Greater facilities for applying automatic protective devices.
- (5) Special shafts, for intermittent use at a distance from the main plant, are more easily provided for.
- (6) Steadier winding.
- (7) Power to control the acceleration.
- (8) Higher rates of de-celeration are economically possible.
- (9) Heavy conical drums are seldom necessary.

Size of Winding Engine.—The calculation of the proper size and strength of the various parts of a winding engine belongs more to the province of the mechanical than to that of the mining engineer. Nevertheless at most of the examinations for colliery managers' certificates, questions on the sizes of winding engines are set, often with very insufficient data to work on, and sometimes with such as no mechanical engineer would accept.

The load which an engine has to overcome is of two kinds—viz., a 'dead load' at the beginning of the wind, and a 'live load' when the cage is in motion. It requires more force to move a dead load than to keep a live load in motion.

The simplest method of calculation is to take the work done in the shaft during one revolution of the drum, at the worst part of the wind. This will be at the moment of the cage coming to the surface, at the moment when the cage with the empty tubs has landed in the pit-bottom, and before the full cage has been brought to rest by the keps, so that the engine has the full weight to support without deriving any advantage from the descending cage.

To leave sufficient margin the load may be taken as equivalent to the combined weights of the coal raised, the rope, and the empty tubs.

* *Trans. Inst. Min. Engs.*, vol. xli. p. 72.

The diameter of the cylinder may then be calculated from the equation,

$$D^2 \times .7854 \times P \times L \times 2 \times E = W \times \text{circumference of drum},$$

$$\text{or } D = \sqrt{\frac{W \times \text{cir. of drum}}{.7854 \times P \times L \times 2 \times E}}.$$

When D = diameter of cylinder in inches.

P = effective steam pressure in lbs. per sq. in.

L = length of stroke in feet.

E = modulus or efficiency of engine (taken at $\frac{4}{5}$ for coupled engines).

C = circumference of drum in feet (diameter $\times 3.1416$).

W = weight of load in lbs. (coal + rope + tubs).

EXAMPLE.

Calculate the size of winding engine required to draw 500 tons per shift of eight hours from a depth of 250 yds. What length of stroke, steam pressure, and size of drum would be required?

$$\text{Tons per minute} = \frac{500}{8 \times 60} = 1.04.$$

Suppose the speed of cage is, on an average, 20 ft. per second.

$$\text{Then duration of wind} = \frac{\text{depth in feet}}{\text{speed of cage}} = \frac{250 \times 3}{20} = 37\frac{1}{2} \text{ seconds.}$$

Allow for time to change tubs at top and bottom (say 15 seconds).

$$\text{Total duration of wind} = 37.5 + 15 = 52.5 \text{ seconds.}$$

Calling this 53 seconds to avoid fractions, the load of coal raised per wind

$$\begin{aligned} &= \frac{1.04 \times 53}{60} \\ &= .91 \text{ ton, or } 18.2 \text{ cwts.} \end{aligned}$$

Suppose that two tubs are raised each wind, each tub holding 10 cwts. of coal, and weighing 4 cwts. when empty; $4 \times 2 = 8$ cwts.

Take the weight of cage at three-fourths the weight of coal raised = 13.8 cwts.,

$$\text{Then coal + tubs + cage} = 18.2 + 8 + 13.8 = 40 \text{ cwts., or } 2 \text{ tons.}$$

Then to find the circumference of rope as a preliminary to finding its weight—

$$C = \sqrt{\frac{W \times M}{3}} = \sqrt{\frac{2 \times 10}{3}} = 2.58 \text{ in. (say } 2.6 \text{ in.)}$$

W , in this case, being the total weight just found (2 tons).

Weight of rope in lbs. per fathom = $C^2 \times .9 = 5.99$ (or 6 lbs. per fathom).

$$\therefore \text{Total weight} = 6 \times \frac{250}{2} = 750 \text{ lbs.}$$

Allow the effective steam pressure to be 50 lbs. per sq. in., and the diameter of drum to equal 150 $\times$ diameter of rope,

$$= \frac{2.6 \times 150}{12 \times 3.1416} = 10.3, \text{ say } 11 \text{ ft. diameter.}$$

Then, applying the formula given above,—

$$D^2 \times .7854 \times P \times L \times 2 \times E = W \times \text{circumference of drum}$$

$$D^2 \times .7854 \times 50 \times L \times 2 \times \frac{4}{5} = \{(18.2 \times 112) + (8 \times 112) + 750\} 11 \times 3.1416$$

$$D^2 \times .7854 \times 50 \times L \times 2 \times .8 = \{2038.4 + 896 + 750\} 11 \times 3.1416$$

$$D^2 \times L \times 50 \times .2 = 1842.2 \times 11$$

$$D^2 \times L = \frac{1842.2 \times 11}{50 \times .2}$$

$$= 2026.42.$$

The length of stroke should be 2 to $2\frac{1}{2}$ times the diameter of cylinder. Assume a stroke of $4\frac{1}{2}$ ft.

$$\text{Then } D^2 = \frac{2026.42}{4.5} = 450.31;$$

$$\text{and } D = \sqrt{450.31} = 21.2 \text{ in. (or 22 in.)}$$

From the above calculation, the size of engine required would be a pair of coupled horizontal engines with

Cylinders 22 in. diameter;

Length of stroke $4\frac{1}{2}$ ft., working with an

Effective steam pressure of 50 lbs. per sq. in. ; and

Diameter of drum 11 ft. ;

the cage being taken at 13.8 cwts., and the two tubs at 4 cwts. each, with a circular steel rope, 2.6 circumference, weighing 6 lbs. per fathom.

This method of calculation is not, of course, given as theoretically correct, but it will give an approximately correct answer to the question.

Professor Merivale gives the following formula for calculating the size of cylinder when the engine is counterbalanced :—

$$A = \frac{C \left(W + \frac{W}{2} \right)}{LP}$$

Where L = twice the length of stroke in feet.

C = circumference of drum in feet.

W = weight of coal per wind in lbs.

A = area in sq. in. of two cylinders ; or half the area of cylinder if there be but one.

P = maximum pressure of steam.

Taking this formula for the above case—

$$A = \frac{11 \times 3.1416 \times \left(2038.4 + \frac{2038.4}{2} \right)}{4.5 \times 50 \times 2} = \frac{34.5 \times 3057.6}{225.0}$$

$$= 234.41 \text{ sq. in.}$$

$$\therefore D = \sqrt{\frac{234.41}{.7854}} = 17.3 \text{ in., diameter of cylinder.}$$

The size of winding engine required can also be worked out in the following way :—

Example.—What diameter of cylinder would be required to raise 1000 tons of coal from a depth of 200 fms. in eight hours ; the tubs 5 cwts. each, and carry 14 cwts. of coal, 4 tubs being raised on each cage, the latter weighing 40 cwts. ? The effective steam pressure is to be 60 lbs. per sq. in., the stroke of engine 5 ft., and the diameter of the winding drum 16 ft.

$$\text{Tons per hour} = \frac{1000}{8} = 125, \text{ winds per hour} = \frac{125 \times 20}{4 \times 14} = 44.6, \text{ or } 45 \text{ for con-}$$

$$\text{venience. Time per wind} = \frac{60 \times 60}{45} = 80 \text{ seconds ; and assume the time taken to}$$

change the tubs is 20 seconds, then the actual time occupied in winding will be $80 - 20 = 60$ seconds, and as the shaft is 1200 ft. deep, the average speed of cage will be $\frac{1200}{60} = 20$ ft. per second. The maximum velocity will be about double

this, or 40 ft. per second. The time in which this velocity is obtained may be taken as $\frac{1}{4}$ th of the total time of winding, or 8.55 seconds.

The circumference of rope required.*

$$= \sqrt{\frac{\frac{L}{4}}{\frac{M}{10} - \frac{1.1 \times 2240}{1.1 \times 2240}}} = \sqrt{\frac{\frac{5.8}{4}}{\frac{10}{10} - \frac{1.1 \times 2240}{1.1 \times 2240}}} = \sqrt{\frac{5.8}{4 - .081}} = 4.25 \text{ in.}$$

Weight of rope per fathom $= C^2 \times .9 = (4.25)^2 \times .9 = 16.3$ lbs. If the head gear is 60 ft. high, then the total weight of rope is $210 \times 2 \times 16.3 = 6846$ lbs.

Let W = weight to be set in motion; two cages, coal, empty tubs on cage, two winding ropes from pit-head pulleys to pit-bottom.

V = greatest velocity obtained, uniformly accelerated from rest = 40 ft. per second.

g = gravity = 32.2.

t = time in seconds during which V was obtained = 8.55 seconds.

L = unbalanced load on engine = coal.

P = effective steam pressure in cylinders = 60 lbs. per sq. in.

N = number of cylinders = 2.

s = space passed through by crank pin in time t .

$c = \frac{2}{3}$ constant to reduce angular space passed through by the crank to distance passed through by the piston during time t .

A = area of cylinder in square inches.

D = diameter of cylinder required.

f = allowance for friction of engines, etc., taken at 20 to 30 per cent.

(a) The greatest work the winding engine has to do is to get the mass W into a certain velocity, uniformly accelerated from rest. This work is difficult to calculate properly, for to do so the energy required to set the winding drum and pulleys in motion would have to be accurately ascertained. To allow for this energy to move these parts, the mass W has been taken to include the weight of the two winding ropes and the two cages.

$$\text{Resistance due to gravity and inertia} = \frac{WV}{gt}.$$

$$\text{Work done in overcoming resistance} = \frac{WV}{gt} \times \frac{Vt}{2} = \frac{WV^2}{2g}.$$

(b) To raise the unbalanced load L , distance passed over in time t .

$$\text{Work done in ft. lbs.} = L \times \frac{Vt}{2} \therefore A = \frac{\frac{WV^2}{2g} + L \frac{Vt}{2}}{P \times S \times N \times C}.$$

If the load is balanced, $L = 4 \times 14 \times 112 = 6272$ lbs., and $W = 24,318$ lbs. In time t the drum will make about 2.5 revolutions, $\therefore S = 2.5 \times 5 \times 3.1416 = 39.25$.

$$A = \frac{\left(\frac{24,318 \times 40^2}{2 \times 32} \right) + \left(6272 \times \frac{40 \times 8.55}{2} \right)}{60 \times 39.25 \times 2 \times \frac{2}{3}} = 535.17.$$

Allowing 20 per cent. for friction, $A = 535.17 + 107.03 = 642.20$, and

$$D = \sqrt{\frac{642.20}{.7854}} = 28.4 \text{ in.} \quad \text{The size of the winding engines would therefore be}$$

28.4 in. diameter with a 5-ft. stroke.

* See formula for calculating size of winding ropes, pages 217, 218.

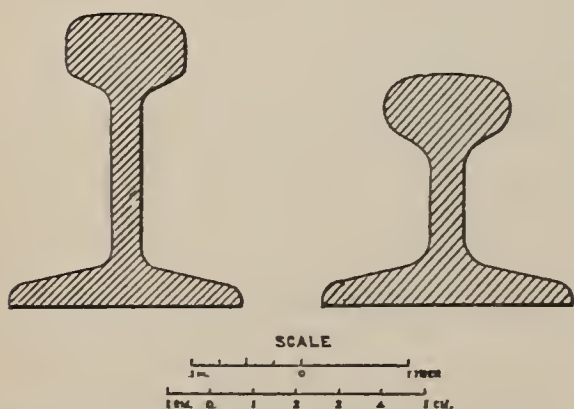
CHAPTER XI.

HAULAGE.

Classification of Methods.—The different modes of haulage employed underground may be classified as follows:—

- | | |
|---|--|
| (1) Manual labour. | |
| (2) Horse haulage. | |
| (3) Self-acting inclines. | { Ordinary balance brae and cut-chain inclines.
Self-acting incline with single tub.
Do. with rakes. |
| (4) Mechanical haulage by stationary engines or motors placed either at the surface or underground. | { Single or dook rope.
Main and tail rope.
Endless rope.
Endless chain. |
| (5) Locomotives. | { Compressed air.
Electrical. |

Haulage Roads.—There is hardly anything of greater importance to an effective system of underground haulage than the laying down of the road. A good road when properly constructed should last as long as the colliery without entailing a great deal of repair or expense. On the other hand, a badly arranged and badly laid road will cause continual trouble and anxiety and entail great expense for repairs, so that it will cost far more in the long run than a good road, to lay down which may, at first, be a little more expensive.



Figs. 395 and 396.—Section of rails.

Rails.—The different rails in use are of three patterns, the flat-bottomed made of cast iron or steel, the 'bridge' rail made either of wrought iron or steel, and circular-head rails usually made of steel.

Flat-bottomed rails made of cast iron with side flange were, at one time, almost universally employed, but the pattern of tub wheel required with this description of rail occasioned so much friction,

that it is being almost entirely abandoned in favour of the newer and more efficient circular-head or T-rail, except in old collieries where there are large stocks of old rails which it is desirable to use up. The newer kind of rail gives the minimum of friction, is easily laid, and, when made of steel, lasts a long time. It generally weighs from 14 to 20 lbs. per yard according to the size of tub used and the weight of load. For main haulage roads, the heavier sections should be selected.

Friction of Rails.—The force required to overcome friction is usually estimated at—

8 to 10 lbs. per ton on a surface railway ($\frac{1}{224}$).	
32 ,,	edge of T-rails underground ($\frac{1}{70}$).
70 ,,	flat-bottomed or tram-plate rails underground ($\frac{1}{32}$).

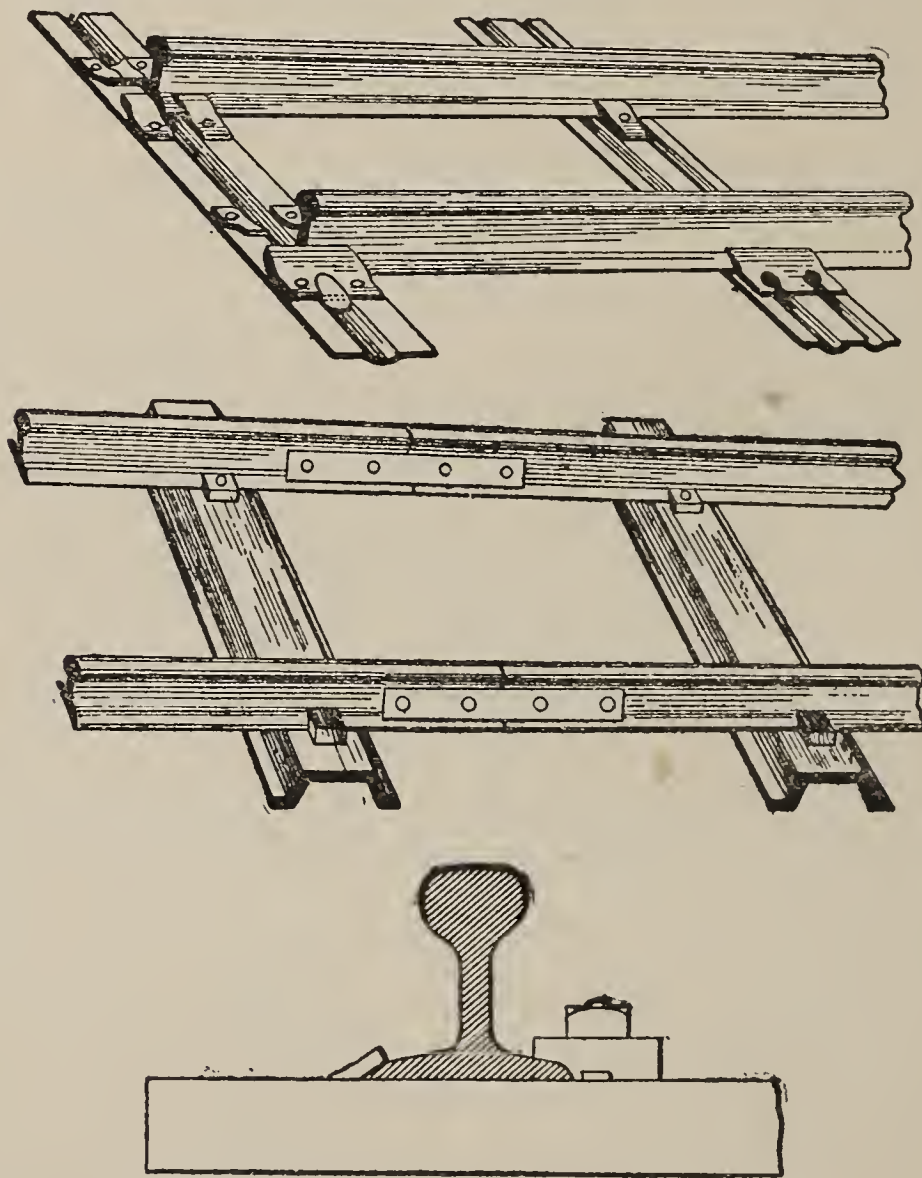
Method of Laying Roads.—Where a large amount of coal has to be hauled over a road, the rails ought to be well laid, on some such system as on a surface railway, with joints 'fish-plated' and rails well keyed and with plenty of 'chairs.'

On the best haulage roads the rails are sometimes laid on longitudinal sleepers, which are held together by cross sleepers, with the joints of the rails fish-plated. The longitudinal sleepers are often ordinary white pine planking 9 in. $\times$ 3 in., and the cross sleepers 5 in. $\times$ 1 $\frac{1}{2}$ in. Wrought-iron and steel sleepers are also used in place of the ordinary wooden sleepers. Laying roads with iron or steel sleepers naturally costs a good deal more than when wood is employed, but the greater durability and stability more than compensate for the increase in first cost. Figs. 397, 398, 399 show the method of laying such roads, and will require no further explanation.

Gauge.—The gauge will depend upon several considerations, such as whether the wheels project beyond or are under the body of the tub, the inclination of the seam, etc. It may vary from between 18 to 36 in., 24 in. being a common gauge. Narrow gauges are most suitable for flat seams, but where the inclination is great the gauge should be increased in proportion. The writer has seen a gauge of 36 in. employed where the inclination was between 30° and 45°.

Tubs.—These are variously called 'trams,' 'corves,' 'hutches,' or 'tubs,' in the different mining districts. The body is usually rectangular in shape, but sometimes they are semicircular at the bottom, which increases their capacity for a given height. The design and size of tubs used will be governed by the varying conditions under which they are required to work, such as the thickness of seam, height and inclination of roads, and whether manual or other kinds of haulage are most largely used. For thin seams where manual labour is used to any considerable extent for haulage the tubs should be made of small capacity and weight. A tub to hold 10 to 12 cwts. should not weigh more than 4 cwts. if constructed

of wood, or 5 cwts. if of iron. The capacity varies greatly, and may be anything between 5 and 40 cwts., but an average size is from 10 to 15 cwts. In the South Wales coal-field tubs holding 30 to 40 cwts. are often used, while in some of the thin-seam collieries of Somersetshire, and in Scotland, tubs holding 7 to 9 cwts. of coal are common. The advantage in having large tubs is that fewer windings at the pit shaft need be made per diem for a given output,

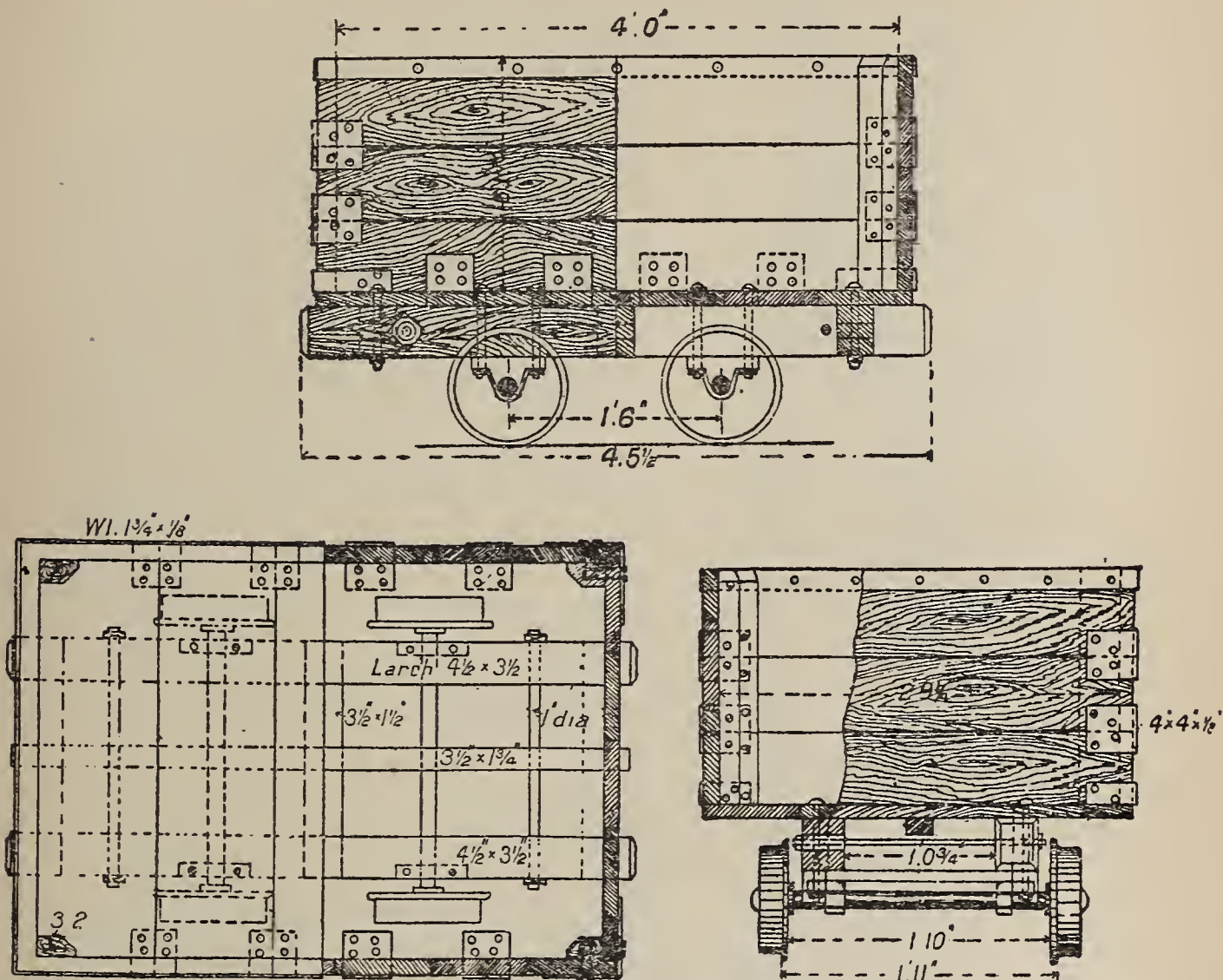


FIGS. 397, 398, and 399.—Method of laying rails.

and that their capacity is large in proportion to their weight. On the other hand, they are clumsy to handle, and, if derailed, are difficult to place on the rails again. On the whole, a tub of medium capacity, *i.e.* 12 to 15 cwts., is to be preferred, as it can be moved about easily and lifted on to the rails by one man when necessary.

Materials of Construction.—Tubs may, as has been stated, be constructed either of wood, wrought iron, or steel. Opinions differ among mining men as to which of these materials is best. For light loads wood is very commonly used, and it has the advantage of being cheap and light, while the cost of maintenance and repair

is less than for steel, and less skilled, *i.e.* cheaper labour, is required for constructing and repairing wooden than for iron or steel tubs. Again, it not infrequently happens that even in the best regulated collieries a train of tubs breaks away on inclined haulage roads, and when such an accident takes place, the tubs will be more or less smashed and broken. If made of wood they can soon be repaired, even although they may be badly damaged, whereas if constructed of wrought iron or steel they would be so twisted and

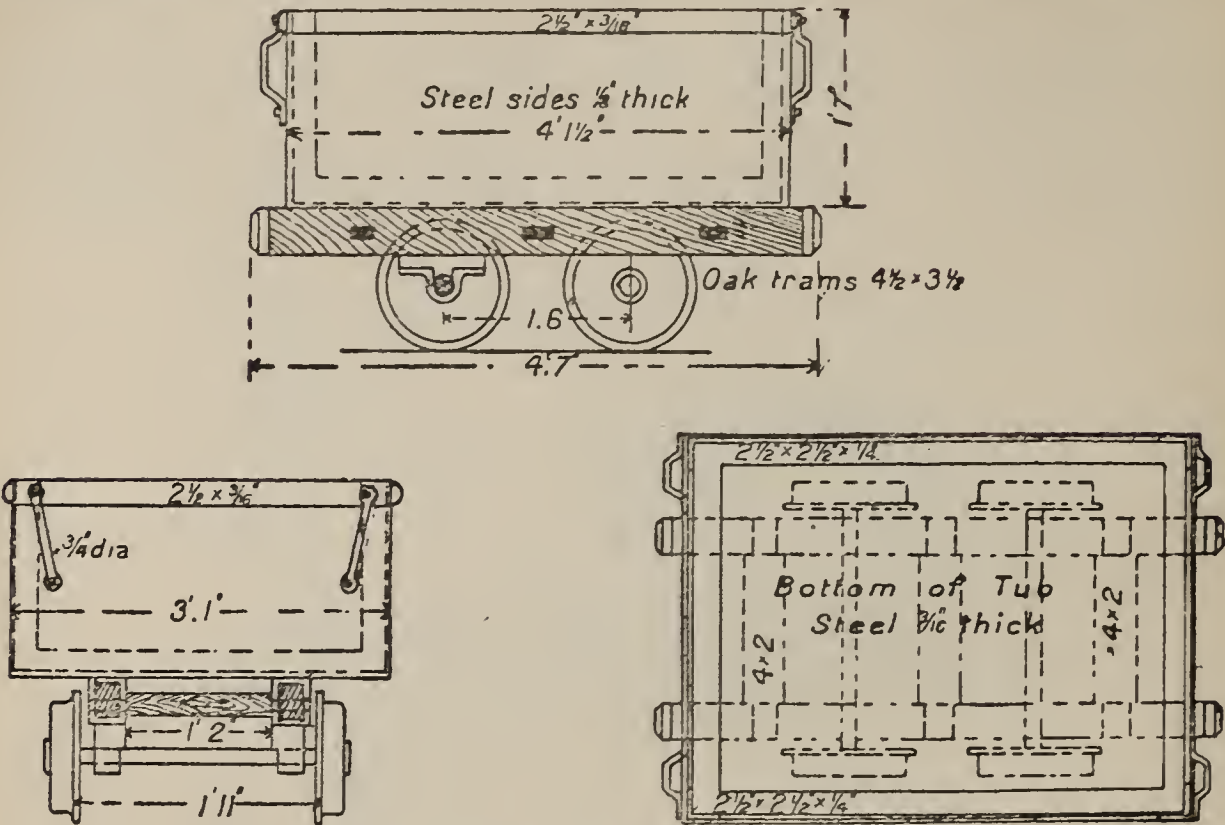


FIGS. 400, 401, and 402.—Details of construction of wooden tub.

bent that the work of a skilled blacksmith would be required for their repair. In thick seams lying at a comparatively small inclination, iron or steel tubs are preferable, especially if the load is large (20 to 40 cwts.).

Figs. 400, 401, 402 show the construction of a tub designed by the author and built of wood. The framework is made of two 'trams' of larch wood $4\frac{1}{2}$ in. $\times$ $3\frac{1}{2}$ in., held together by oak 'starts' $3\frac{1}{2}$ in. $\times$ $1\frac{1}{2}$ in., and further strengthened by two iron rods 1 in. diameter. The body is of larch cleaving $1\frac{1}{8}$ in. thick, bound at the sides and ends with pieces of sheet-iron angle $\frac{1}{8}$ in. thick. It

is also bound round the top with a band of wrought iron. The capacity is $9\frac{1}{2}$ cwts. when filled to the level of the sides, and about



FIGS. 403, 404, and 405.—Construction of steel tub.

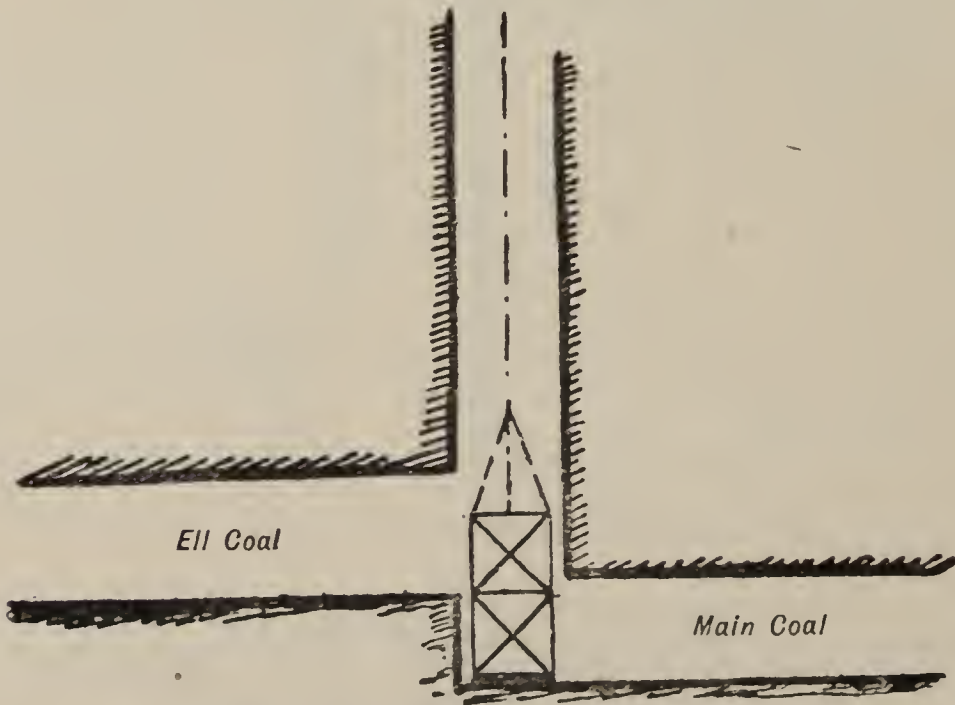


FIG. 406.—Elevation of pit-bottom.

12 cwts. when heaped up above them. The cost of such a tub would be about £2, 15s. or £3 complete.

Figs. 403, 404, 405 show a tub constructed of steel sides and ends resting on a wooden frame. The latter is of larch held together by oak 'starts' 4 in. $\times$ 2 in. The sides are of steel $\frac{1}{8}$ in. thick, and

angle iron $2\frac{1}{2}$ in. $\times$ $2\frac{1}{2}$ in. $\times$ $\frac{1}{4}$ in. running round the bottom, and also at each corner, in order to bind the sides and ends together. The capacity of such a tub is 10 cwts., or 12 to 14 cwts. when heaped. The estimated cost is £4, 10s.

Arrangement of Pit-bottom.—The first consideration in any system of haulage is to have the pit-bottom laid out so as to best accommodate the tubs as they come and go. Many pit-bottoms are too confined, which may be a saving in first cost, but is never satisfactory in working. It should be so arranged that all the empty tubs can be taken off the cages at one side and the full tubs placed on at the other. Where it can be carried out, this is the most satisfactory arrangement.

Figs. 406, 407 show the arrangement of the pit-bottom at Earnock Colliery. Here the coal is raised from two levels, and these are so arranged that both decks of the cage can be charged at the same time, without the cage being moved. Fig. 406 shows a section of the shaft which more clearly explains the working. On the top deck, Ell coal is caged, while the lower deck is reserved for coal from the Main and other seams. To carry this out the lower level had to be formed in the strata above the Main seam, and a stone mine driven at an inclination in order to catch the coal. Fig. 407 shows the plan, which will be readily understood. On both sides of the pit-bottom there are double roads for full tubs and a road for empties. On the side A, which is worked by main and tail rope, the road for the empties dips away from the shaft for a certain distance at a gradient of 1 in 50 and then rises to the level of the other road at an inclination of 1 in 30, so that no labour is entailed in pushing the empty tubs forward to the desired position. The two full roads on each side are constructed to hold about 80 tubs. From the high to the low stage a by-road is formed, so that, if there is not sufficient coal coming in to

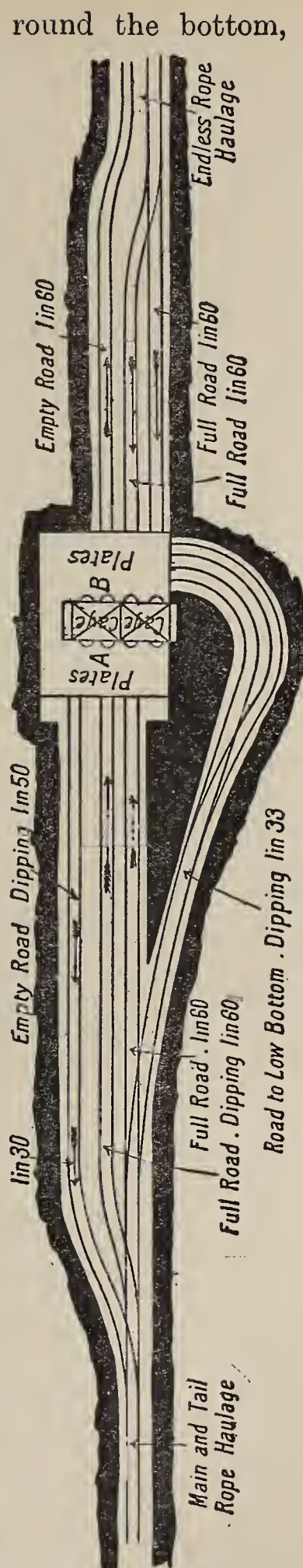


FIG. 407.—Plan of pit-bottom.

keep the low deck going, a train of tubs can be run down from the high level A to the low level B. The whole arrangement works very well, over 1200 tons being often raised in eight hours, with four tubs on each cage. Of course this arrangement would not suit in every case, but it may be taken as an illustration of a well-planned pit-bottom. Fig. 408 shows another arrangement of laying out a pit-bottom for endless rope haulage. The loads come in at one side of the shaft and the empties go off at the other side, the gradients of the roadways on both sides of the shaft being such that the tubs gravitate from the points of detachment and attachment respectively.

Manual Labour.—The haulage of tubs by men and boys can only

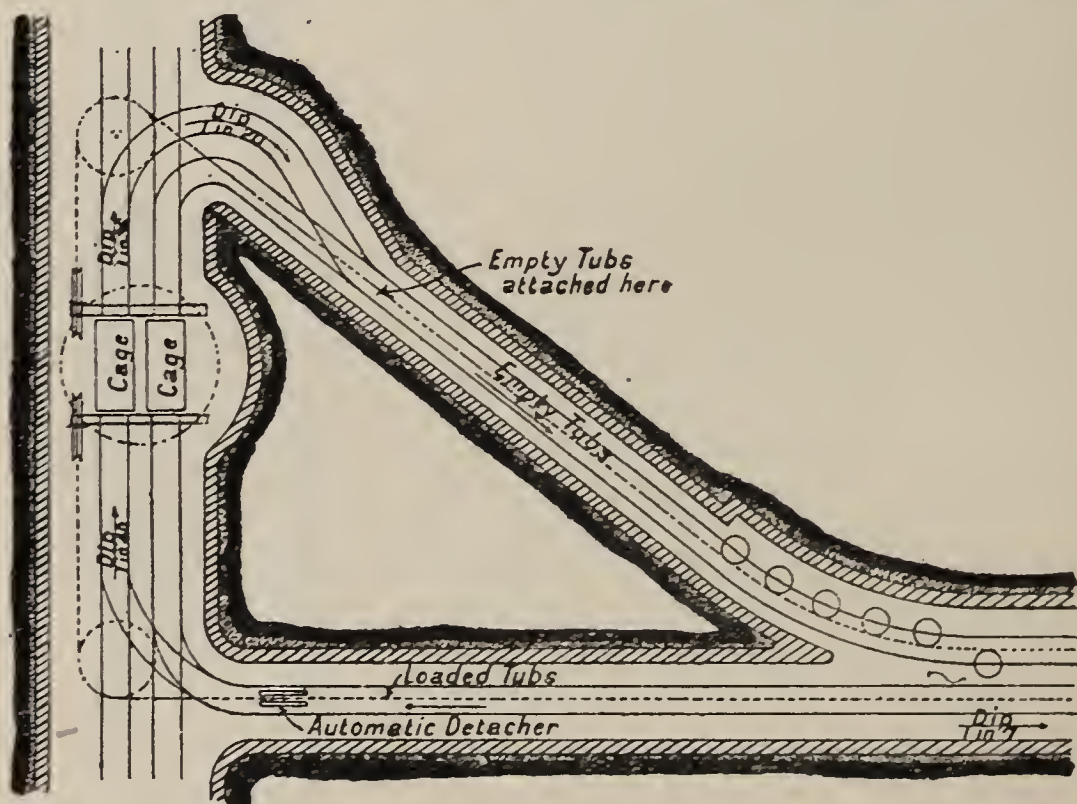


FIG. 408.

be employed with advantage where the drawing roads are of moderate length and comparatively level, and where the tubs are not too large. If the length of road is great, or the inclination high, it is the most expensive system that can be adopted, and in any case it is always better, where the height of seam will allow, to bring the tubs direct from the face by means of ponies or by some other method of traction. Manual haulage should therefore be confined to short distances on level roads next the working face. In some small collieries where the seams are thin and the roads flat, it is often the only method employed for hauling the coal from the faces to the pit-bottom. It has been calculated that a young strong man will only push 3 tons a distance of one mile in twelve hours along a level road laid with edge rails, road and tubs in good order, and the tubs fitted with wheels 10 in. diameter.

Horse Traction.—Haulage by horses is employed more or less in nearly all collieries, there being sometimes 100 to 150 horses underground. If the inclination of the roads is not too great, it is often an economical and convenient form of haulage, especially if there is a slight inclination in favour of the load towards the pit-bottom. Horse haulage is either confined to turning out tubs from the working face to mechanical haulage or between lyes or sidings, and sometimes from lyes direct to the pit-bottom, especially in the initial stages of a new colliery, before the permanent haulage has been laid down. A horse is capable of exerting a tractive force of 120 lbs., when travelling at the rate of two to three miles per hour, and can keep this up for a period of ten hours, which enables it to draw 24,000 lbs. or 10 tons for a distance of twenty miles, or 200 tons for a distance of one mile per diem. Two-thirds of this estimate is a fair average of the work which a horse usually performs. The mileage depends greatly on local circumstances, and on the size and kind of horse, but on moderately level roads horses ought to travel fifteen or sixteen miles per day, and in roads dipping 1 in 20, eight or ten miles per day, if the roads are, in each case, in good order, with long runs and few stoppages.

Horse Roads.—The most economical form of horse road is where the work done, in drawing in an *empty load*, is equal to the work done in taking out a *full load*; which for edge rails should be an inclination of about 1 in 140, or $\frac{1}{4}$ in. per yard, in favour of the full load, when the roads are in good order.

If a tub is placed on a line of rails laid perfectly level, a force will be required to move it. The pull or push necessary to move the tub is called the *tractive force* (T), and the force acting against or holding it back is termed resistance, or more commonly friction (F). Before the tub can be moved, T must be greater than F. The resistance or friction varies with (a) the weight of tub or load; (b) the type of rails used; (c) the velocity of speed at which the tub is moved. With mining tubs friction chiefly arises from (1) friction of wheels and axles; (2) friction of wheels on rails (especially at bad joints and crooked rails); (3) air currents.

The first two causes are the most important in underground work, and, therefore, to reduce friction and make the tubs more easily hauled, it is essential to have well-turned and well-lubricated axles, wheels of as large diameter as possible, and well-laid roads, kept as free as possible from débris. The tractive force (T) required to move a tub on an underground road will depend on the amount of friction it meets with, and the inclination or gradient of the roadway.

In pushing a tub up an inclined road, we are practically lifting a certain proportion of its weight (which is a varying proportion according to the gradient of the road) and overcoming the earth's attraction or gravity (G). On a perfectly level road, therefore, the

tractive force required will be only that sufficient to overcome friction; if the tub is being moved on a road with the gradient opposite to the direction of motion, the tractive force required will be that necessary to overcome friction plus the load due to gravity ($F + G$), while if the gradient be in favour of the tub the tractive force will be friction minus load due to gravity ($F - G$).

The tractive force required to overcome friction, or, as it is called, the coefficient of friction, for mining purposes may be taken as follows:—

Waggons above-ground on surface railways	.	8 to 10 lbs. per ton	$= \frac{1}{2\frac{1}{2}}$.
Tubs underground on edge rails	.	32 „ „	$= \frac{1}{7\frac{1}{6}}$.
„ „ bridge „	.	37½ „ „	$= \frac{1}{8\frac{1}{6}}$.
„ „ tram „	.	60 „ „	$= \frac{1}{3\frac{1}{3}}$.
Ropes, rollers, etc., underground	.	80 to 112 „ „	$= \frac{1}{2\frac{1}{10}}$ to $\frac{1}{2\frac{1}{8}}$.

The power or force required to overcome friction varies as: (1) the diameter of the axle; (2) the weight of the tub or load; (3) coefficient of friction; (4) inversely as the diameter of the wheels.

Resistance to Traction.—

Let F = resistance in lbs. due to friction upon a level line of rails in fair condition,

W = weight of tub in lbs.,

D = diameter of wheel in inches,

d = „ „ axle „ „

m = coefficient of friction $= 0.0882 \frac{d}{D}$;

then $F = mW = 0.0882 \frac{d}{D} \times W$.

This formula is based on friction for edge rails laid in good condition.

Example.—If a tub, whose gross load is 14 cwts., with wheels 10 in. diameter, and axles 1½ in. diameter, is moved on a level line of rails, what would be the friction or resistance?

$$F = 0.0882 \times \frac{1.5}{10} \times 14 \times 112 = 20.74 \text{ lbs.}$$

Resistance due to Gravity or Gradient.—

Let I = fraction representing the gradient,

H = vertical height in feet load (W) is moved through,

L = lineal distance „ „ „

W = load in lbs.,

R = resistance in lbs. due to gradient;

$$\text{then } I = \frac{H}{L} \text{ and } R = \frac{WH}{L}.$$

Again, if W = weight of full or loaded tub in lbs.,

w = „ „ empty „ „

m = coefficient of friction as before;

$$\begin{aligned} \text{then } R &= mW + \frac{WH}{L} \\ (1) \quad &= F + \frac{WH}{L} \end{aligned} \left. \vphantom{\begin{aligned} \text{then } R &= mW + \frac{WH}{L} \\ (1) \quad &= F + \frac{WH}{L} \end{aligned}} \right\} = \begin{array}{l} \text{Resistance due to friction and gravity with} \\ \text{inclination against load or load going uphill.} \end{array}$$

$$(2) \left. \begin{aligned} R &= mW - \frac{WH}{L} \\ &= F - \frac{WH}{L} \end{aligned} \right\} = \begin{array}{l} \text{Do. with inclination in favour of load or going} \\ \text{downhill} \end{array}$$

Further, if we let w = weight of load going uphill,
 W = " " " " downhill,
 F and f = friction of load going uphill and downhill
 respectively,

$$\text{then } F - \frac{WH}{L} = f + \frac{wH}{L},$$

$$\text{or } F - W\frac{H}{L} = f + w\frac{H}{L},$$

$$\text{and by transposing we have } F - f = W\frac{H}{L} + w\frac{H}{L},$$

$$\text{or } F - f = \frac{H}{L} (W + w)$$

$$\therefore \frac{H}{L} = \frac{F - f}{W + w}.$$

But we have seen that $\frac{H}{L} = I$; $\therefore I = \frac{F - f}{W + w}$, which will be the gradient required on a horse-drawing road where the work going downhill (out-bye) is to equal the work going uphill (in-bye); or substituting the values of F and f , we have

$$I = \frac{mW - mw}{W + w}$$

$$= \frac{m(W - w)}{W + w}.$$

I here represents the sine of the angle of inclination; the angle itself can be got from the sine by the use of trigonometrical tables.

Example.—What should be the gradient for a horse haulage road, if the load consists of 12 tubs, each holding 15 cwts., and weighing 5 cwts. when empty, allowing $\frac{1}{10}$ for friction?

$$\begin{aligned} I &= \frac{\left(\frac{12 \times 15}{70}\right) - \left(\frac{12 \times 5}{70}\right)}{(12 \times 15) + (12 \times 5)} \\ &= \frac{\frac{180}{70} - \frac{60}{70}}{180 + 60} = \frac{\frac{180 - 60}{70}}{240} = \frac{1.71}{240} = \frac{1}{140}, \end{aligned}$$

or 1 in 140 (the gradient required).

Feeding of Horses.—This forms one of the heaviest items in the keeping of horses underground, as a fair-sized horse will cost 10s. to 14s. per week for food and bedding alone. The aim is to keep horses in the best possible condition at the minimum of cost, and this can be easily accomplished if the right feeding-stuffs are selected. Formerly the feeding consisted chiefly of boiled food, composed largely of beans, barley and bran, along with dry hay. Now it has

been pointed out by qualified authorities, that bran has little or no feeding value unless as a laxative, and that barley and beans should only be used sparingly. Nearly all collieries now adopt dry feeding, or 'chop,' as it is termed, for the horses, this being found to keep them in harder and better condition than soft or boiled food. The latter, however, has its place, as animals, like men, appreciate a change, and this may be given at least once a week with advantage. Some green food during summer will also be relished, and is beneficial. The following 'chop' is sometimes given :—

Cut hay,	.	.	.	.	.	8	cwts.
Oats,	.	.	.	.	.	5	„
Beans,	.	.	.	.	.	2½	„
Indian corn,	.	.	.	.	.	2½	„
Pease,	.	.	.	.	.	2	„
Total,	.	.	.	.	.	20	„

Mr J. B. Hamilton, in a paper read before the Mining Institute of Scotland, gave the following mixtures for a daily feed :—

	No. 1.	No. 2.	No. 3.
Beans or pease,	2 lbs.	2 lbs.	...
Maize (bruised),	6 „	4 „	4 lbs.
Oats,	4 „	3 „	3 „
Bran,	...	2 „	2 „
Hay (cut),	12 „	12 „	12 „
Straw (cut),	2 „	2 „	2 „
	26 „	25 „	23 „

No. 1 is a daily feed for a pony about fifteen hands high hardened to its work ; No. 2 for same size of pony new to work ; and No. 3 is for a pony doing no work.

Mr Hunting* says that the quantity of food must be regulated by the amount of work required ; but about 100 lbs. of corn, crushed and mixed with about 56 lbs. of chopped hay, will form an average week's provender for each horse. He considers that beans (or pease) and hay with either (1) oats and bran, (2) barley and bran, (3) oats and maize, (4) maize, will form equally good mixtures, and that we must be guided in our selection by current prices.

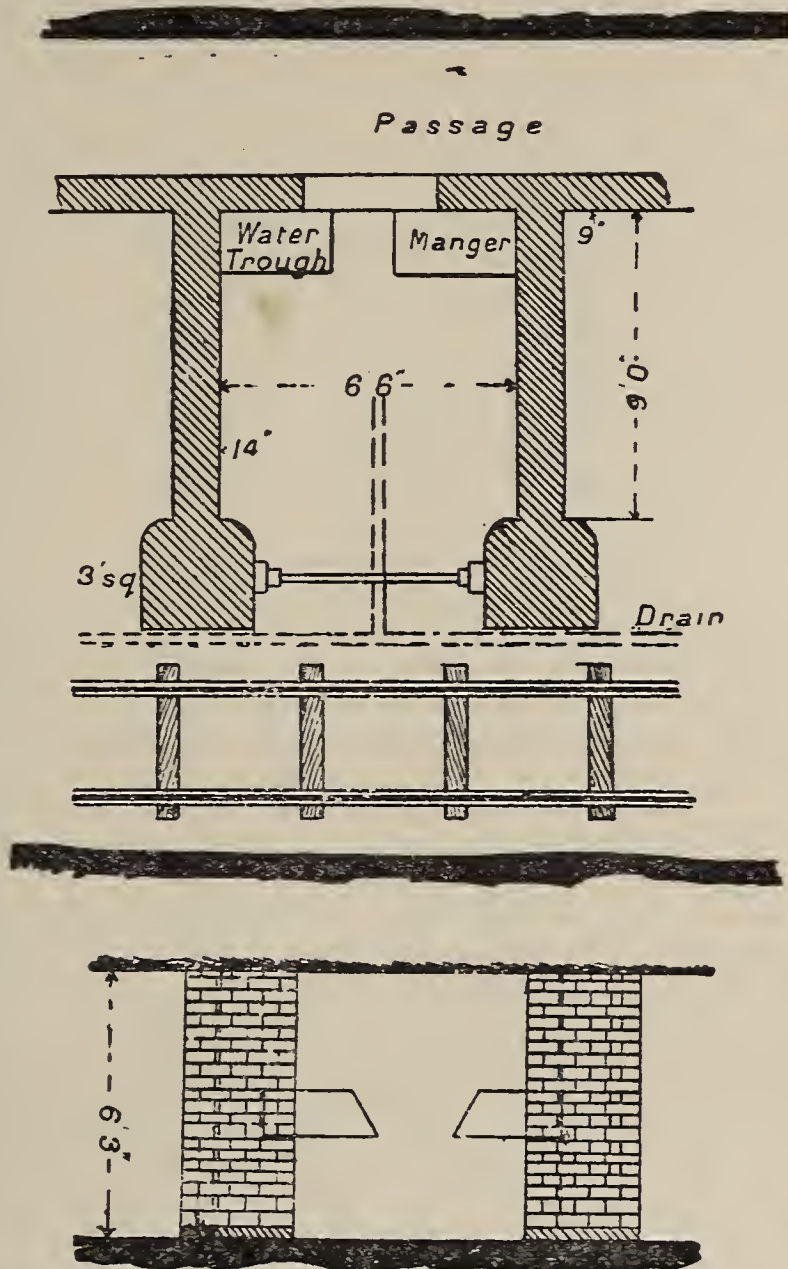
Feeding is not, however, the only thing to which attention should be devoted. Keeping horses in good condition requires a plentiful supply of good clean drinking water, and they must also be properly bedded, and above all, kept well cleaned and groomed. It is too often left to the pony driver to attend to these matters, but no greater mistake can be made, especially if there are sufficient ponies to afford work for an ostler, as many pony drivers are mere boys, who have no experience whatever as to how a horse ought to be cleaned, and the majority of them are too careless to pay much attention to the needs of the horse or pony under their charge.

* *Trans. N. Eng. Min. and Mech. Engs.*, vol. xxxiii. p. 61.

Wherever there are more than a dozen horses underground, it will, as a rule, pay to put them under the charge of a competent ostler or horse-keeper. Horses should also have an opportunity of eating during working hours, as their work is exhausting, and a horse cannot retain sufficient food to maintain it for long intervals. Provision should therefore be made at the lyes or sidings, where the horses stop, for them to obtain food while waiting. A visiting veterinary surgeon should also be appointed, where there are a large number of horses, to examine them periodically, and to report upon their condition. Horses underground ought to be properly shod at stated intervals. The cost of shoeing will on an average amount to about 6d. per week. The total cost of each horse per week may be estimated at 12s. to 15s., including shoeing, attendance, repairs to harness, feeding, and bedding.

Stables.—The arrangement of the stables is a very important matter, and it will pay to expend some thought and care in planning them. Many stables are merely formed in the workings, without any attempt at flooring or drainage, the result being that the floor gets worn into holes, making it very uncomfortable for the animals to rest on.

Figs. 409, 410 show the construction of stables as sometimes adopted. The stalls, which are walled in on each side with a brick wall 14 in. thick, are 9 ft. $\times$ 6½ ft. $\times$ 6 ft. A gutter is formed down the centre of the lower half of the stall, and connected with a main gutter running at right angles to it, and in the direction of the roadway. A tram road is laid along between the rows of stalls, for convenience in cleaning them, and a passage is also constructed



FIGS. 409 and 410.—Plan and section of stables.

along the head of each stable for the distribution of food. The floor should be well laid with bricks, laid sideways, and carefully cemented in, the floor having a rise towards the head of about 2 in. in the yard. A large watering-trough should be placed outside the stables for the horses to drink, when going to and returning from work.

Cost of Horse Haulage.—The cost of horse haulage depends on various conditions, such as the gradient of the road, the condition of the rails, roads, and tubs, especially the wheels, and whether the roads are wet or dry. The cost of horse haulage at the faces, *i.e.* lifting tubs from the faces to mechanical haulage, may be taken as 9d. to 1s. 6d. per ton per mile; the cost of horse haulage when used for conveying coal from lyes to the pit-bottom, in rakes or sets, may vary from 2d. to 6d., or sometimes as high as 1s. per ton per mile. The cost for keep of horses, shoeing, harness, etc., has to be added. This may be taken at $\frac{1}{2}$ d. to $\frac{3}{4}$ d. per ton per mile. Prof. William Galloway gives the following details of the cost of secondary haulage, *i.e.* collecting tubs from working places and conveying them to sidings:—

System.	Agents.	Unit of Weight.	Day's Work.	Cost per Day.	Cost per Ton per Mile.
		lbs.	yard, tons.	s. d.	s. d.
Handputter,	Boy, . . .	550	1610	2 0	2 2·3
Horse, . . .	Man and horse, . . .	2800	6864	6 6	1 8
Pony, . . .	Lad and pony, . . .	896	5577	4 5½	1 4·8
Hand, . . .	Man, . . .	896	5252	3 8	1 2·74

In these examples the gradient of the roads where the different agents are employed will vary considerably.

The Coal Mines Act, 1911, now enforces the following provisions regarding the employment of horses underground. Horses under four years old are prohibited, and they must be certified free from glanders. They have to be housed in properly constructed stables, the stalls to be of adequate size. Stables to be ventilated with fresh intake air, cleaned daily, kept in a sanitary condition, and if made in the natural strata, or lined with timber, to be white-washed every three months. A horse-keeper or ostler is to be appointed for every fifteen horses, and a sufficient supply of good and pure water is to be provided; also a supply of medicines, ointments, and dressings, and a suitable appliance for the destruction of horses when necessary. Horses are not to be worked in an unfit condition, improperly shod, and must have properly fitting harness, including a guard for the eyes. The ostler has to make a daily written report as to the condition of

all the horses under his charge. (For the full regulations see Third Schedule of the Act.)

Self-acting Inclines.—Self-acting inclines may be divided into four different classes:—(1) Balance braes or jig brows, where a balance weight is used. (2) Self-acting incline working with two tubs on the cut-chain principle. (3) Self-acting inclines working in the ordinary way with trains of tubs. (4) Self-acting inclines worked on the endless rope principle with tubs attached singly.

In the great majority of mines, the conditions are such that very often one or all of the above sorts of incline can be employed. Where the inclination of the seam is sufficient to allow them to run freely, they form very easy and economical means of haulage.

Balance Brae or Jig Brow.—This is the simplest form of self-acting incline, and is usually employed at the faces for bringing out a single tub at a time, to a siding or main incline. It is worked by a rope or

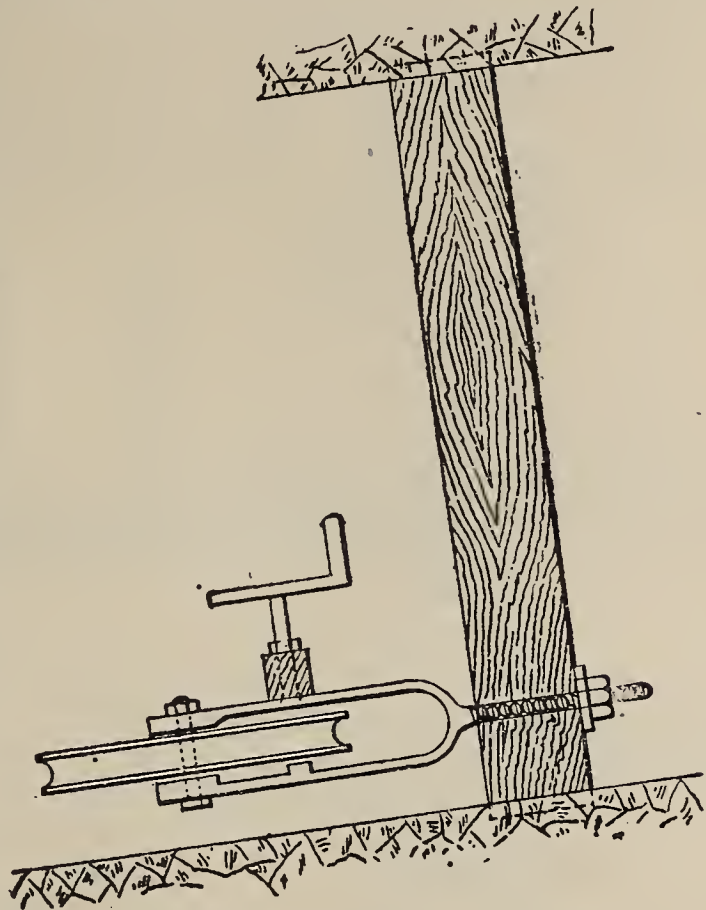


FIG. 411.—Jig brow arrangement.
(It is understood that the floor line inclines about 10° to 15° .)

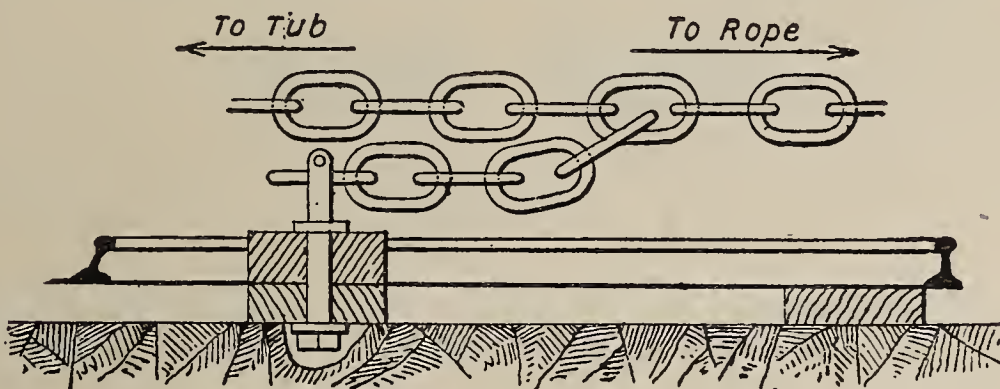


FIG. 412.

chain, more usually the former, passing round a wheel, 18 in. or 24 in. diameter, fixed between two iron jaws which terminate in a screwed bolt passed through a prop at the face, and held tight by a nut and washer (fig. 411). Two sets of rails are provided, one set having a narrow gauge for a loaded carriage or bogie to run on, and

the other set with wide gauge for the tub to run on. The bogie is filled with scrap iron, and is generally so balanced that it can bring up the empty tub without requiring any brake on the wheel. The full tub on descending draws the loaded bogie up to the face, and the bogie on descending brings up the empty tub. To prevent the loaded carriage from running back, when the full tub is disconnected from the rope, at the foot of the incline, the rope is held fast as shown in fig. 412. To the short piece of chain attached to the

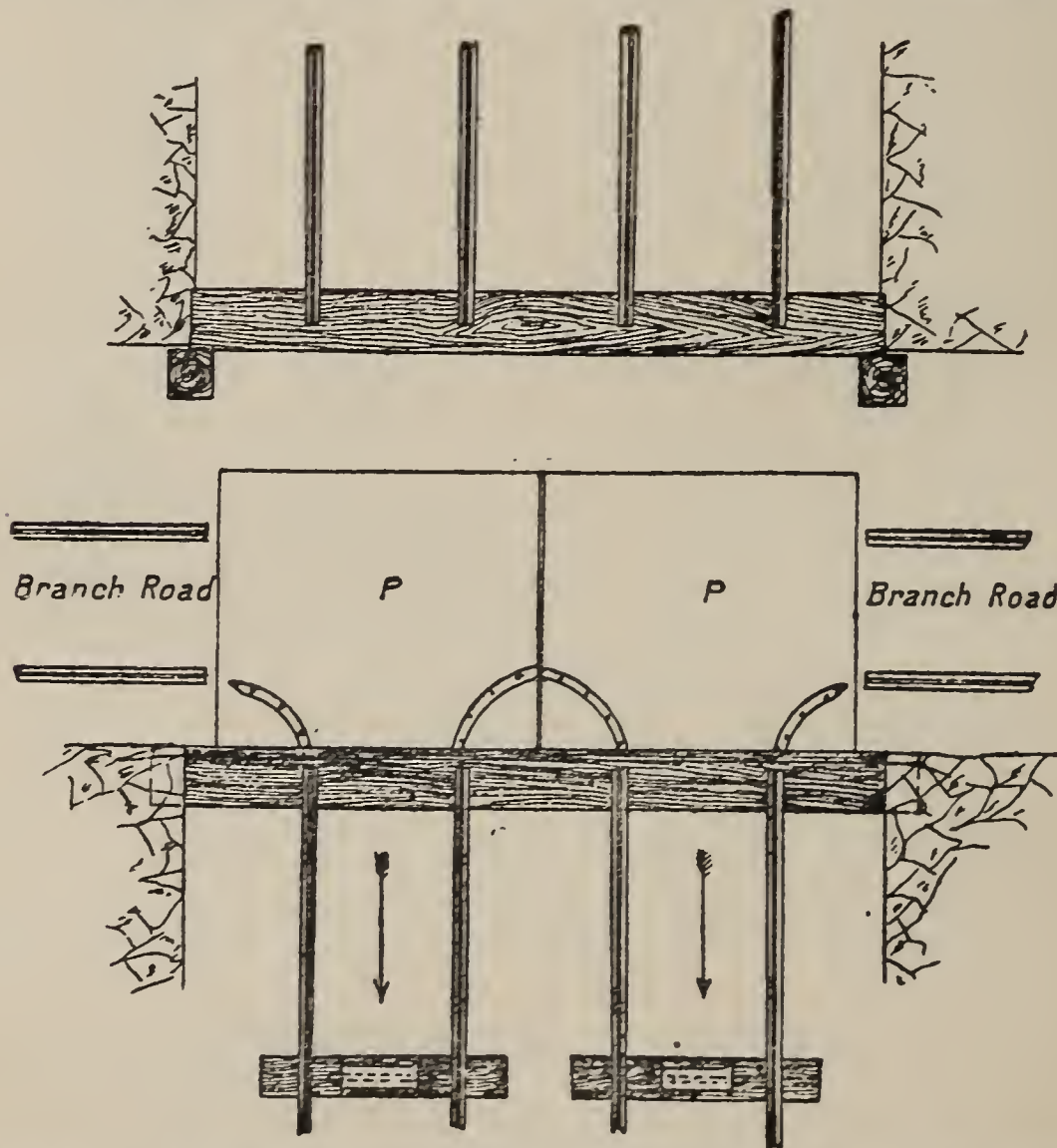
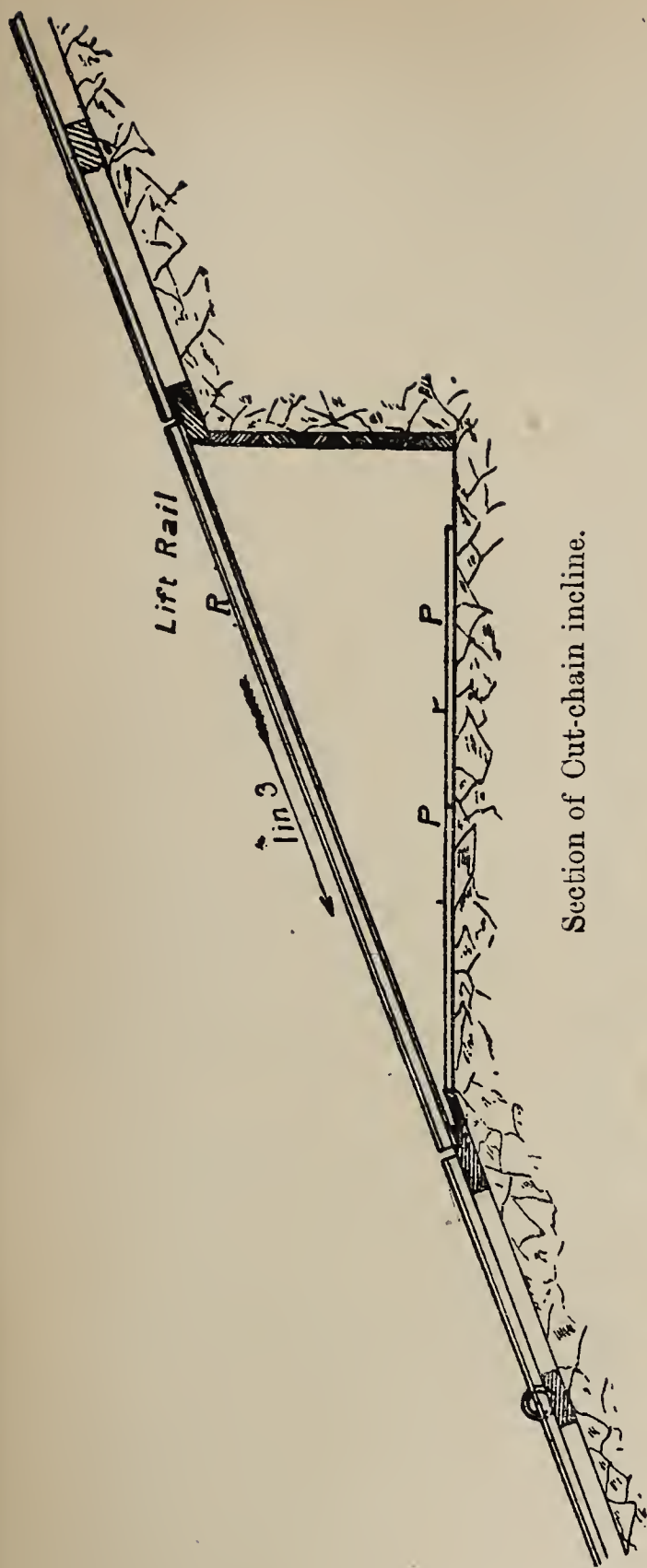


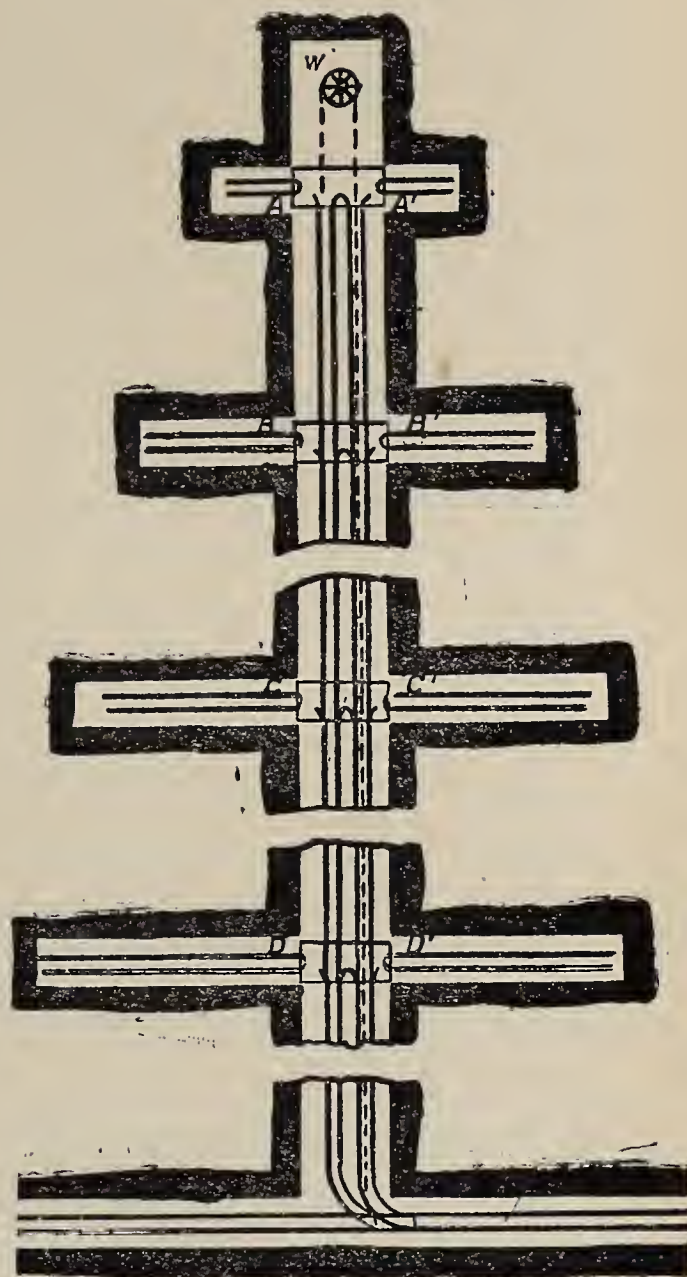
FIG. 413.—Enlarged plan of cut-chain, showing main and branch roads, with lift rails out of position.

capping of the rope, an auxiliary piece is fixed. To a double sleeper is fixed a bolt over which a link of the auxiliary chain is passed, and held in position by a pin. This holds the rope in position until the empty tub is attached.

Jigs are best suited for seams where the gradient is greater than 1 in 6, and the author has seen them working satisfactorily in roads having an inclination of 35° to 40° , but in such cases great caution must be exercised, and the roads should be thoroughly well laid. In highly inclined roads the wheel at the face ought to be provided with a hand brake as shown in the sketch, and the prop well notched into



Section of Cut-chain incline.



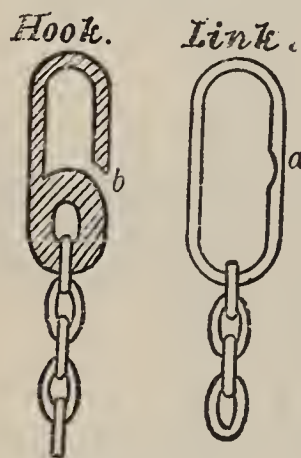
FIGS. 414 and 415.—Cut-chain incline.

The rope passes round a wheel at the top, W, fig. 415, and has connections or 'cuts' at each of the 'hanging on' stages, A B C D, at none of which are any platforms used, but the road is levelled out and two plates P P, figs. 413 and 414, with lift rails R, pass over them to the rails on the opposite side. Fig. 415 shows a plan of how these branches are arranged.

the roof and floor. Unless this is carefully attended to, accidents are almost sure to occur.

Cut-chain Incline.—This is a somewhat different form of self-acting incline to the above, for instead of a loaded carriage being used, a regular double road is required, and the weight of the full tub going down brings the empty tub up to the face. This system is largely used in Fifeshire and other districts of Scotland, and can be employed either in stoop or room or longwall working, if the inclination is sufficiently great. The working of this system will be understood from figs. 413–415.

Method of Working the Cut-chain Incline.—Suppose a tub requires to be run down from A, the portion of the chain not required is detached and left lying on the road between the rails. The full tub is then attached, run down, and an empty one brought to A'; another full tub should be kept ready at A' to be run down and bring up an



FIGS. 416 and 417.

empty to A, so that the chain can be again connected and ready for a 'cut' at any of the other branches B B', C C', D D'. At the parts where the chain is cut, special links are required for disconnecting and reconnecting the chain. These links are shown in figs. 416, 417. The thin part *a* of the link in fig. 417 is made so that it fits the link in fig. 416 through the opening *b*. When the two links are thus connected, it is impossible for them to be separated, as no other part of the link except the part *a* can pass through the opening *b*.

At each branch road, highly inclined (30° to 45°) from the horizontal, the roadway is levelled out and cast-iron plates laid down, with 'lift' R rails to pass over them to connect the rails above and below the branch. When a tub is being run down from a branch road, these 'lift' rails are lifted out of position and laid aside until the full tub is taken down and the empty one brought up, when they are again replaced, which leaves the road joined up ready for tubs to be run from any of the other branches above. If the gradient does not exceed 1 in 4 or 1 in 3, the lift rails are usually dispensed with, the floor of the road at the branches simply being levelled out somewhat, and the plates laid down with a gradient of 1 in 15 to 1 in 20, and the tubs allowed to run over them without the aid of the 'lift' rails. To enable the tubs to pass easily over these plates, they are cast with a groove the gauge of the wheels, so that when the tub leaves the rails the flanges of the wheels enter these grooves, in which they run until the rails are again entered upon below the branch.

When branch roads are worked from one side of the heading or incline, the arrangement for running the tubs is somewhat different.*

* *Trans. Inst. Min. Engs.*, vol. xiv. p. 196.

A wheel is fitted up at the top of the heading in the way already described, and the chain passed round it, one end reaching the foot of the incline and the other lying near the top. At each branch road or 'cut' there is a disconnecting or cut-link in the chain as in the first arrangement, so that the upper portion can be disconnected as required. At the branch roads, plates, with grooves in them, are laid down in the same way as shown for the incline, with branches worked from both sides. At a short distance above these plates, about 6 ft. or so, a small grooved wheel is fixed, preferably between the two lines of rails. Whenever a tub requires to be run down from any of the branch roads the chain is disconnected at the cut-link, and the end of the lower portion passed round this small pulley and connected to the end of the load tub. To prevent the tub from running off before the chain is attached, a block made of a piece of wood 4 or 5 in. square, and of sufficient length to stretch across both rails, is fixed across them. This block turns on a bolt at one end, and when in position is fixed by an iron pin at the other end, which passes through the block into a plank or sleeper on the roadway. When the end of the chain has been drawn round the



FIG. 418.—Self-acting incline.

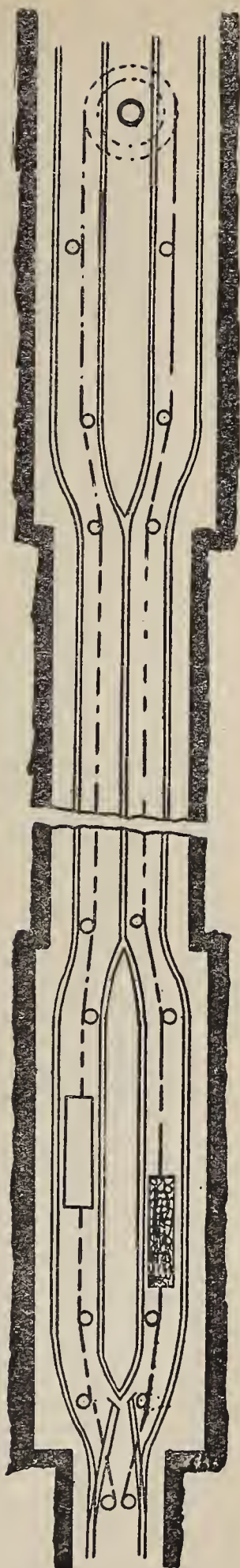


FIG. 419.—Incline with double tram lines.

wheel and fixed to the tub, the latter is run down to the foot of the incline, an empty tub being brought up on the other line of rails to the branch road. Like the arrangement previously described, two tubs should be run from each branch so as to bring the chain back to its original position ready for connecting. If this is not done, and only a single tub is run down, the portion of the chain reaching from the branch road, from where it has been 'cut,' will have to be lifted across to the centre of the line of rails where it was originally, and this can only be done if there are no centre props on the incline. On the wheel at the top, if the gradient requires it, a brake is fitted, operated from each of the branches or benches by means of a wire attached to the brake-handle, and led down the whole length of the incline.

A large amount of coal can be run down these inclines when they are properly constructed. They are best suited for lengths of 50 to 100 fms., beyond which they do not work so well.

The same system can also be used with a balance weight in longwall working where it is often difficult to keep the road wide enough for a double tram road. The cut-chain system



Plan and Elevation of Incline.

FIGS. 420 and 421.—Endless chain incline.

can be worked on any inclination from 10° up to 45° .

Self-acting Incline with Trains of Tubs.—This is the commonest

form of incline, and can be used with great advantage where the gradient is suitable. The most usual arrangement is to have a rope passing two or three times round a drum or wheel, to give sufficient friction and prevent slip. The drum or wheel is fitted with a good brake, and the rope is attached to the full train of loaded tubs, which by their own weight bring a train of empties to the top of the incline. The best plan is to have a double road throughout (fig. 418), as the incline will work more satisfactorily than by using two or three sets of rails. If the roof is bad, it may be difficult to make and maintain a road of sufficient width to admit of a double train line throughout. In such circumstances two or three rails may be used with a pass-by in the centre. Fig. 419 shows an incline with three rails and pass-by.

If the inclination is great, the best results are got by a drum instead of a wheel for lowering the tubs, as it can be better kept under control; on the other hand, the ropes are more difficult to keep in line with the roads, owing to the lapping of the ropes on the drum. If the inclination is not too great, a suitable wheel gives very good results, and has the advantage of taking up little space, and the ropes can be kept in a straight line with each road, if the wheel is fixed horizontally. About 3° is the least inclination an incline will work at easily, but the length of incline is a determining factor, as the longer the incline the greater will be the weight of rope and the friction. On the other hand, the shorter the length of incline and the heavier the load, the better will it work.

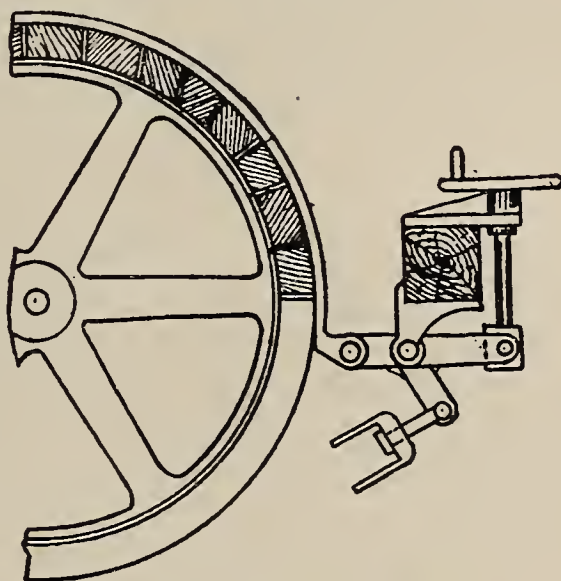


FIG. 422.—Brake wheel.

Self-acting inclines are sometimes worked on the endless rope or chain principle, with the tubs attached singly at stated distances apart. Where the conditions are suitable this plan yields very good and economical results. Figs. 420, 421 show the arrangement of such an incline worked with an endless chain, and fig. 422 illustrates the brake wheel used on this kind of incline, and which is somewhat differently arranged to an ordinary incline wheel, as the attendant has to handle the brake, and also to attach the tubs to the chain. The tubs are attached to the rope at intervals of 20 to 30 yards; where a tub has to be attached the attendant applies the brake and brings the rope to a standstill; the tub is attached, and the rope is allowed to move until the next interval. The advantages claimed for this system are: small cost for upkeep of rolling stock, the slow speed causing fewer breakages; regularity of delivery; and

economical working. The length makes little difference, and there is a little less expenditure in making benches at the top and bottom of the incline. Although roads with regular gradients are best adapted for the application of this system, it can also be successfully employed upon roads the inclination of which varies.

To find the gradient at which a self-acting incline will work, the following method can be employed:—

Let W = weight of full train of tubs.

w = „ empty „

w_1 = „ rope or chain.

F = friction of full train.

f = „ empty train.

f_1 = „ rope, rollers, etc.

I = tangent of angle of inclination.

The weight of the full load must be able to overcome that of the empty load plus the friction of both loads, and of the rope, drums, rollers, and accessories; that is, $(W \times I) - F$ must be greater than $(W \times I) + f$, and $(W - w) I$ greater than $F + f$. From this reasoning we can establish the formula—*

$$(W - w - w_1) \times I = (F + f + f_1) \quad \therefore I = \frac{F + f + f_1}{W - w - w_1}.$$

The angle of inclination thus obtained would be that at which the pull of the full train would exactly balance the resistances on the incline, so that it would require a greater angle of inclination to enable the system to work easily.

Example.—Find the angle of inclination for a self-acting incline to act properly if the trains consist of 10 tubs, the gross load of each full tub being 16 cwt. and each empty tub weighing 5 cwt. The weight of the rope is to be taken at 5 lbs. per fathom, the length of the incline 150 fms., and the weight of drums at 8 cwt. The friction of the tubs may be taken at $\frac{1}{70}$, and that of the rope and drums at $\frac{1}{20}$.

$$\begin{aligned} \text{Then } I &= \frac{\left(\frac{10 \times 16 \times 112}{70}\right) + \left(\frac{10 \times 5 \times 112}{70}\right) + \frac{(8 \times 112) + (150 \times 5)}{20}}{(10 \times 16 \times 112) - (10 \times 5 \times 112) - (8 \times 112) + (150 \times 5)} \\ &= \frac{256 + 80 + 82 \cdot 3}{17,920 - 5600 - 1646} = \frac{418 \cdot 3}{10,674} = \frac{1}{25 \cdot 5}, \text{ or 1 in } 25 \cdot 5. \end{aligned}$$

To make the incline work easily, it should be 20 per cent. steeper, *e.g.*, 1 in 20.

Carriage Inclines.—When the inclination exceeds 45° a carriage is employed to enable the tubs to assume a horizontal position.

Fig. 423 shows the construction of such a carriage, which may be made to hold two, four, or six tubs. These carriages are much used on the Continent, where the measures are greatly inclined. They are also used in England and in the oil-shale and coal-mines of the Lothians in Scotland. On the Continent such inclines are often worked with a balance weight with a single road, so arranged that the balance weight passes under the carriage at the point of meeting.

Blocks.—At the top of all inclines means must be adopted to

* It must be remembered that in the above formula I does not represent the angle of inclination, but the tangent of that angle. The former can be found from the latter by consulting trigonometrical tables.

prevent the loaded tubs from running prematurely down the incline. This is usually accomplished by means of blocks or stops. A common form of stop used is simply two blocks of wood working on pivots, one of which crosses the rail for the tub wheel to rest against, and abuts on the other, which is placed at right angles to it.

'Staple' or 'Blind' Pits.—It often happens that the coal is lowered from one seam to another by means of 'blind' or 'staple' pits instead of by an incline. These pits are not so expensive to make as

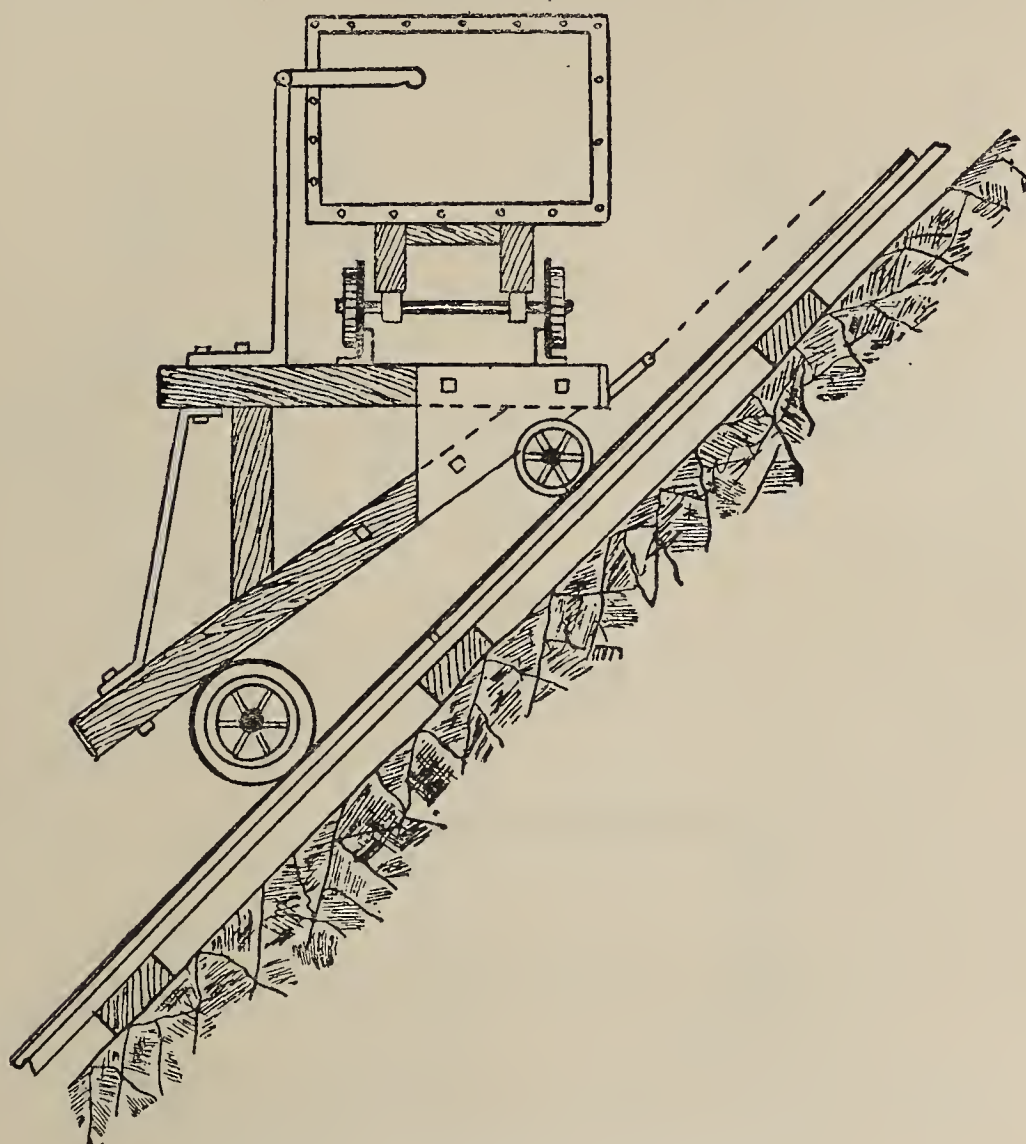


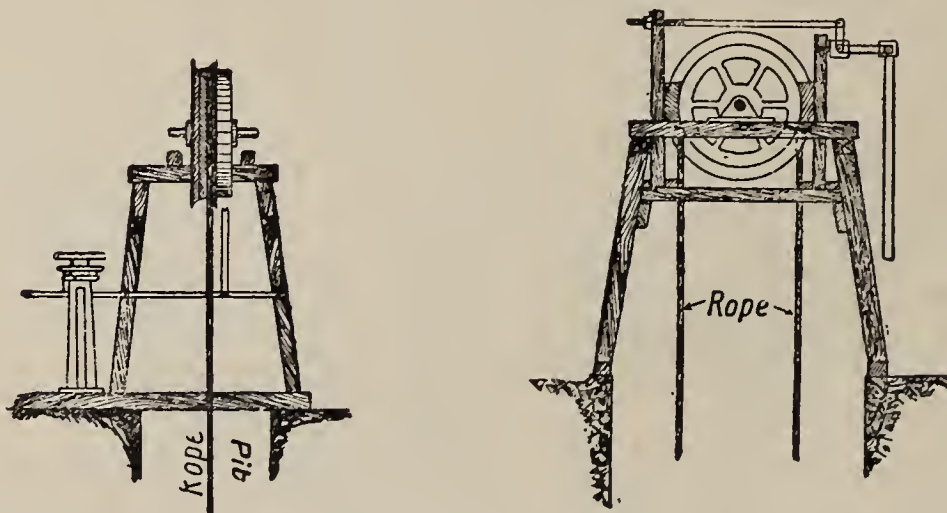
FIG. 423.—Carriage for tub on inclines.

an incline, and if properly fitted and the depth is not great a large quantity of coal can be lowered by their means. The arrangements at the top of these pits are shown in figs. 424, 425. The pits are usually fitted with two cages like an ordinary winding shaft, the cage with the full tub being able to outbalance and raise the cage with the empty tub. Sometimes one cage only is used along with a balance weight, but this is not such a good arrangement. In working, these pits ought to be carefully fenced to prevent tubs from being pushed into them when the cage is not in position. Many accidents have taken place through lack of this precaution.

Haulage by Stationary Engines.—We now come to the different systems of haulage by stationary engines, which play such important parts in the successful working of modern collieries. These systems can be divided into —

- (1) Direct haulage or single rope haulage.
- (2) Main and tail rope haulage.
- (3) Endless chain haulage.
- (4) Endless rope haulage.

The workings of a colliery may be so distributed that two or more



FIGS. 424 and 425.—Arrangement of brake wheel for staple pit.

of the above systems of haulage can be applied in different sections or localities.

Direct Rope Haulage.—When the inclination is sufficiently great

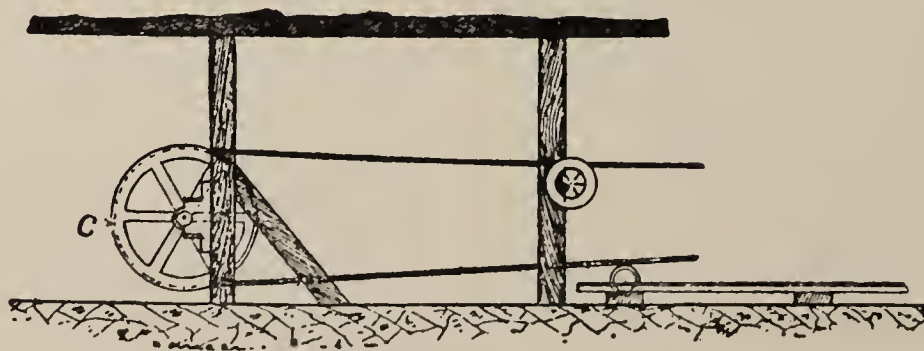


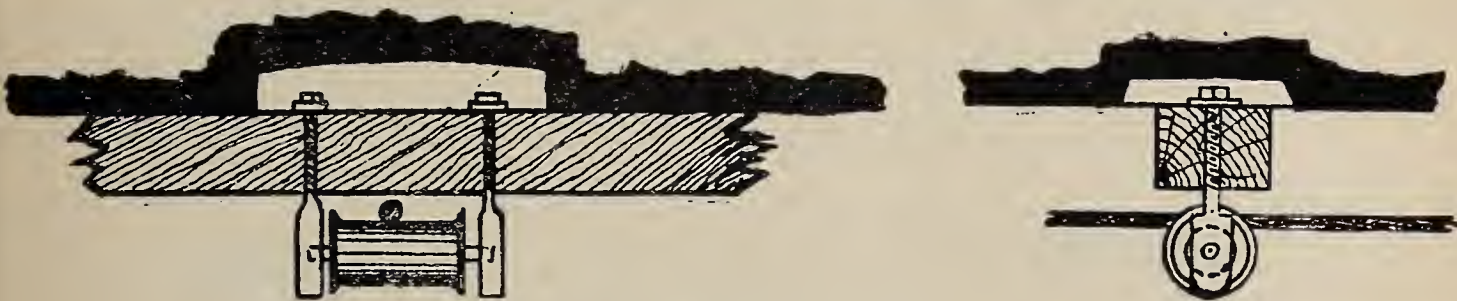
FIG. 426.—Main and tail-rope haulage.

for the train of tubs to run down of their own accord, and to overcome the friction of the rope to which they are attached, and of tubs, drum, etc., a single rope may be used with advantage. An engine with a single drum is employed, the latter being so arranged that it can be thrown out of gear and run loose on the shaft. The engine draws the full train of tubs against the inclination to the pit-bottom, while the empty train runs in-by and drags the rope after it, the drum being out of gear and running loose on the engine shaft.

To work this system successfully the inclination should be not less than 1 in 26. For single rope haulage the engine is best placed at

the pit-bottom, as this gives the engineman the advantage of having everything in view, and mistakes are not so apt to be made by taking the tubs too far when the train is approaching the bottom. The main advantage of this system is that it is cheap, so far as first cost is concerned, a single road and one rope only being required, while any number of branches can be easily worked by the same engine.

Main and Tail Rope Haulage.—When the inclination is insufficient or irregular, and the empty train unable to run in-bye by its own weight, it is necessary to use another rope called a 'tail' rope to draw in the empty tubs. The tail rope is wound on a drum on the same shaft as the main rope drum, and passes along the side or roof of the roadway, till it reaches the far end of the road, where it passes round a wheel *c* placed either vertically or horizontally (fig. 426), and comes on to the main roadway, where it is attached to the end of the full train of tubs, the main rope being attached to the front of



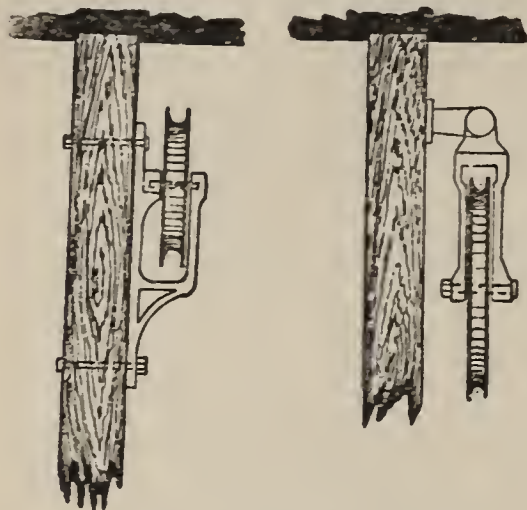
FIGS. 427 and 428.—Guides.

the latter. The tail rope, therefore, requires to be twice the length of the road.

Working of System.—Suppose an empty train of tubs to be standing at the pit-bottom ready to be hauled in to the workings, the drum with main rope will be thrown out of gear, and allowed to run loose on the driving shaft, while that on which the tail rope is wound will be in gear. The engine will haul the empty train in-bye with the tail rope attached to the front of it and the main rope attached to the back. When the loaded train has to be hauled out-bye these conditions will be reversed, the tail rope drum will be thrown out of gear and the main rope thrown in gear, the tail rope being now attached to the back of the train and the main rope to the front. The tail rope is usually a good deal lighter than the main rope, as it is only required to haul in the empty tubs. It will be seen from the above that this system of haulage requires a length of rope three times the length of road, the tail rope requiring to be twice the length and the main rope equal to it, so that when both ropes are connected to the train of tubs it practically becomes an endless rope working on a single road.

Guiding the Ropes.—As only a single road is used, the tail rope is guided either along the side of the road or along the roof; whichever method is adopted the pulleys ought to be set so as to run freely,

otherwise there is great wear and tear on the ropes. When the tail rope is guided overhead, the pulleys are often fixed as shown in figs.



FIGS. 429 and 430.—Guides.

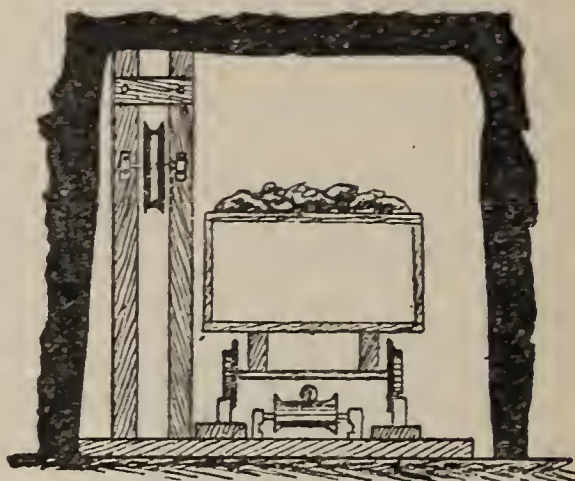


FIG. 431.—General arrangement of roads for main and tail ropes.

427, 428. A pulley or roller is hung by means of two hangers, about 6 in. $\times$ 3 in. $\times$ $\frac{5}{8}$ in. tapering to $\frac{5}{8}$ in. diameter at the top, where it is fixed to a bearer by means of a nut and washer. For guiding

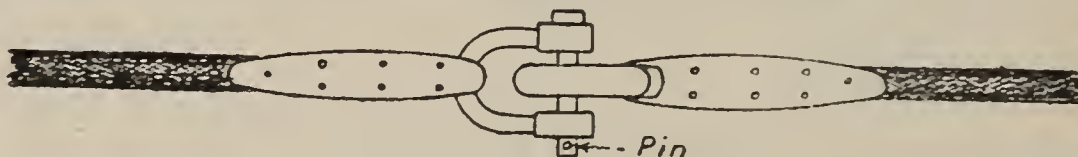


FIG. 432.—Shackle.

the rope along the side of the road the arrangement of pulley shown in fig. 429 may be adopted. If there are curves on the road, swinging pulleys (fig. 430) should be used, which will enable the rope to

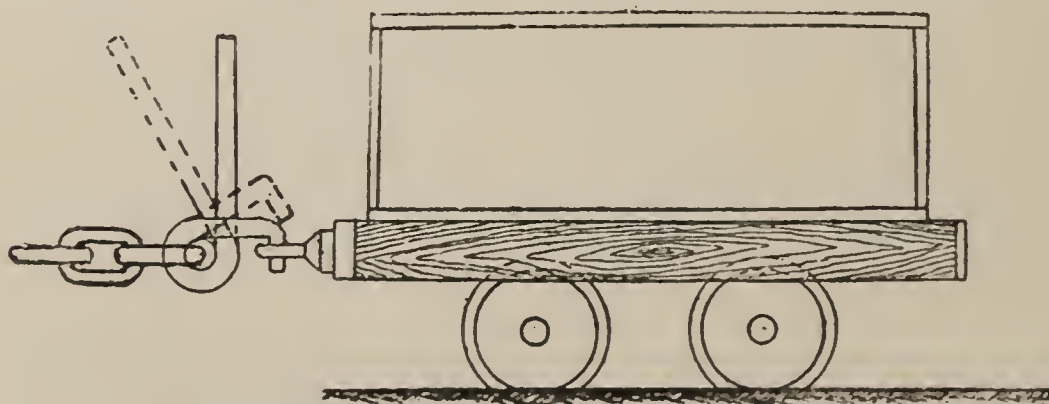


FIG. 433.—Automatic knock-off arrangement.

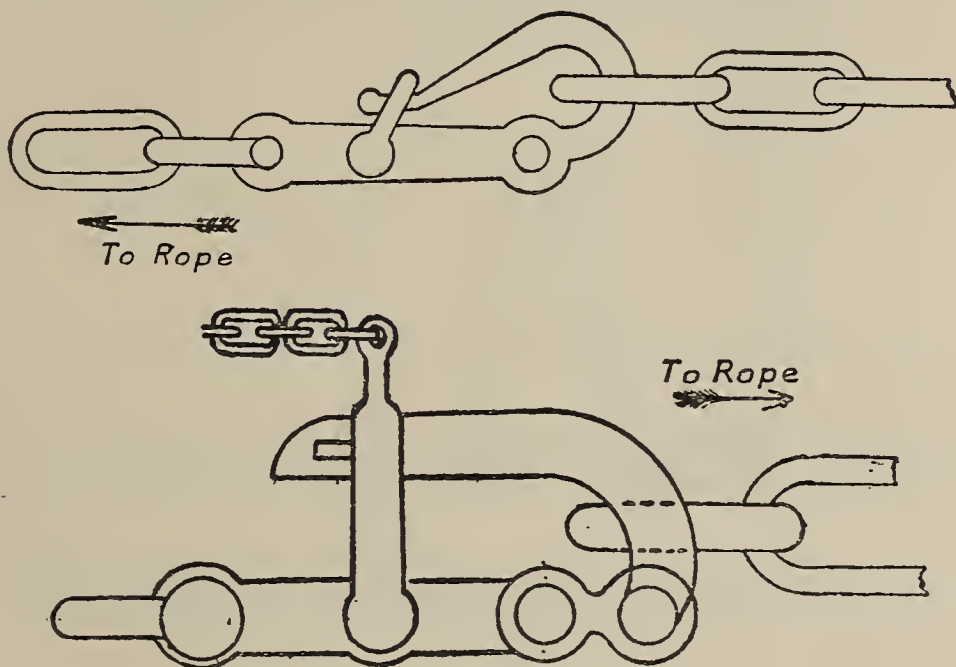
take the line best suited to it. Fig. 431 shows the general arrangement of road with pulleys for main and tail ropes.

Detaching the Rope.—When the train of tubs is near the shaft the rope is detached, and the hauling engine brought to a standstill. The rope may be disconnected either by hand or automatically. When this is done by hand a common shackle may be used (fig. 432),

the connecting bolt having an eye for a small pin to pass through to prevent it from working out.

More frequently an automatic 'knock-off' is used, as shown in fig. 433. Other 'knock-off' arrangements are illustrated in figs. 434, 435.

A somewhat different arrangement for automatically detaching the rope is shown in figs. 436, 437. This apparatus, which is often termed a 'monkey,' is placed on the front of the train of tubs, and consists of a crank *a*, working on a standard *b*, which is fixed to the tub by a fork arrangement, or special clamp fitted to the tub. To the end of the crank is a chain *c*, with a pin at the end for fixing the rope. At the place where it is desired to detach the rope, a beam is fixed across the road sufficiently low to strike the upright

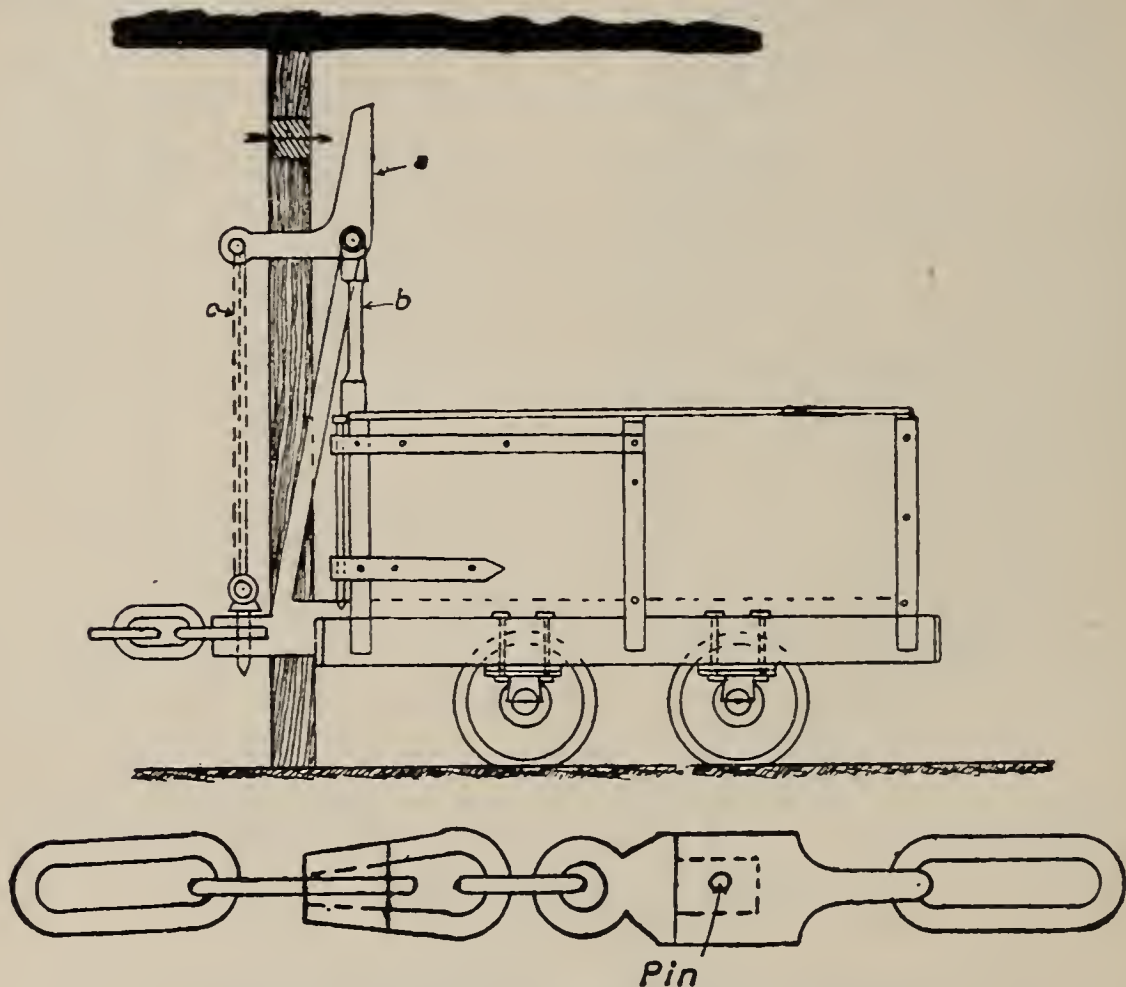


FIGS. 434 and 435.—Other knock-off contrivances.

lever of the crank, thus raising the chain with the pin and releasing the rope.

Working Branch Roads.—Branches from the main road are worked by means of separate ropes, which pass round a wheel at the extreme end of each branch, and extend to the junction with the main road. The working and connecting of these branch ropes may be done in three different ways. In two of these the connection is made when the empty train is brought to the junction of the branch road, while in the third method the connection is made while the empty train is standing at the shaft. In the first method, if an empty train requires to be taken into branch *a* (fig. 438), the end *b* of the main line tail rope is disconnected from the train, and the end *b'* of the branch rope is attached in its place. The main line tail rope is also cut at *a*, and the other end of the branch rope, *a'*, connected to it; the tubs are then drawn into the branch road. In the second method (fig. 439) the main line tail rope is disconnected at *b*, and the end

a of the branch rope connected to the tubs; the engine now draws the rope forward a short distance, and the other end of the rope is connected to *c*, the train being then ready for hauling in. In the



FIGS. 436 and 437.—The 'Monkey' knock-off.

third system the ropes are so arranged that when the full train on the main road reaches the shaft, the two disconnecting points *yy* (fig. 440) are just opposite the branch *c*. If a supply of tubs is wanted in this branch, the rope is 'cut' at *yy*, and the ends *xx* of



FIGS. 438 and 439.—Working branches.

the branch rope are connected instead, before the train of tubs leaves the pit-bottom; a run can then be made into the branch without stopping at the junction, which makes the work both simpler and more expeditious.

The main and tail rope system of haulage is extensively used in some districts, notably in the North of England, where at one time

it was employed to the exclusion of every other method. The speed at which the rope travels is high, varying from six to twelve or fifteen miles per hour. To ensure success with this system, the roads should be as straight as possible, and well laid with heavy rails. Where there are curves on the road, the rope should be carefully guided with bevel pulleys, and 'check' rails used to keep the tubs from becoming derailed. A modification of this system is sometimes adopted, in which an endless rope is worked in opposite directions as required, but this method is not at all to be recommended, and should only be used under exceptional circumstances.

The advantages of the main and tail rope method of haulage are that only a single road is required, the first cost being consequently less than with endless rope haulage with double road; roads with irregular inclinations and curves can be worked easily, although the system is best adapted for straight roads with regular inclinations,

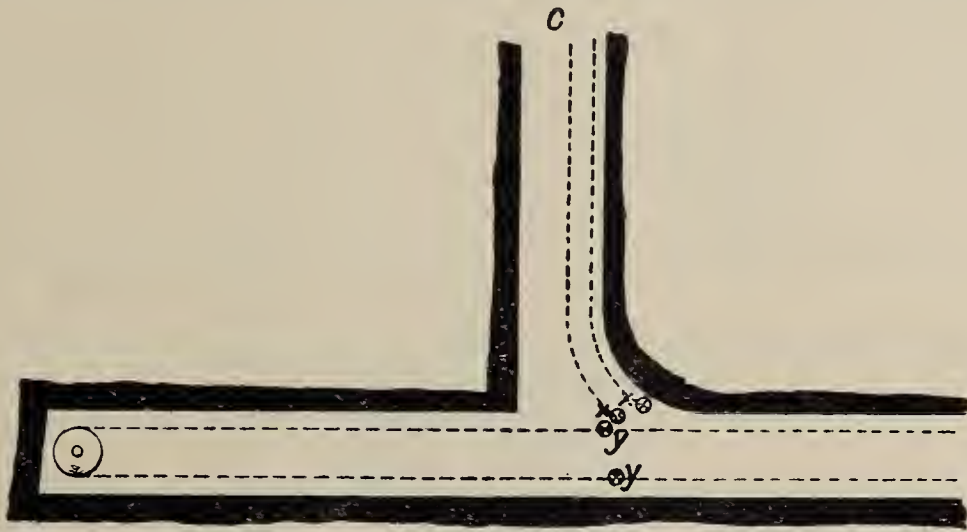


FIG. 440.—Working branch.

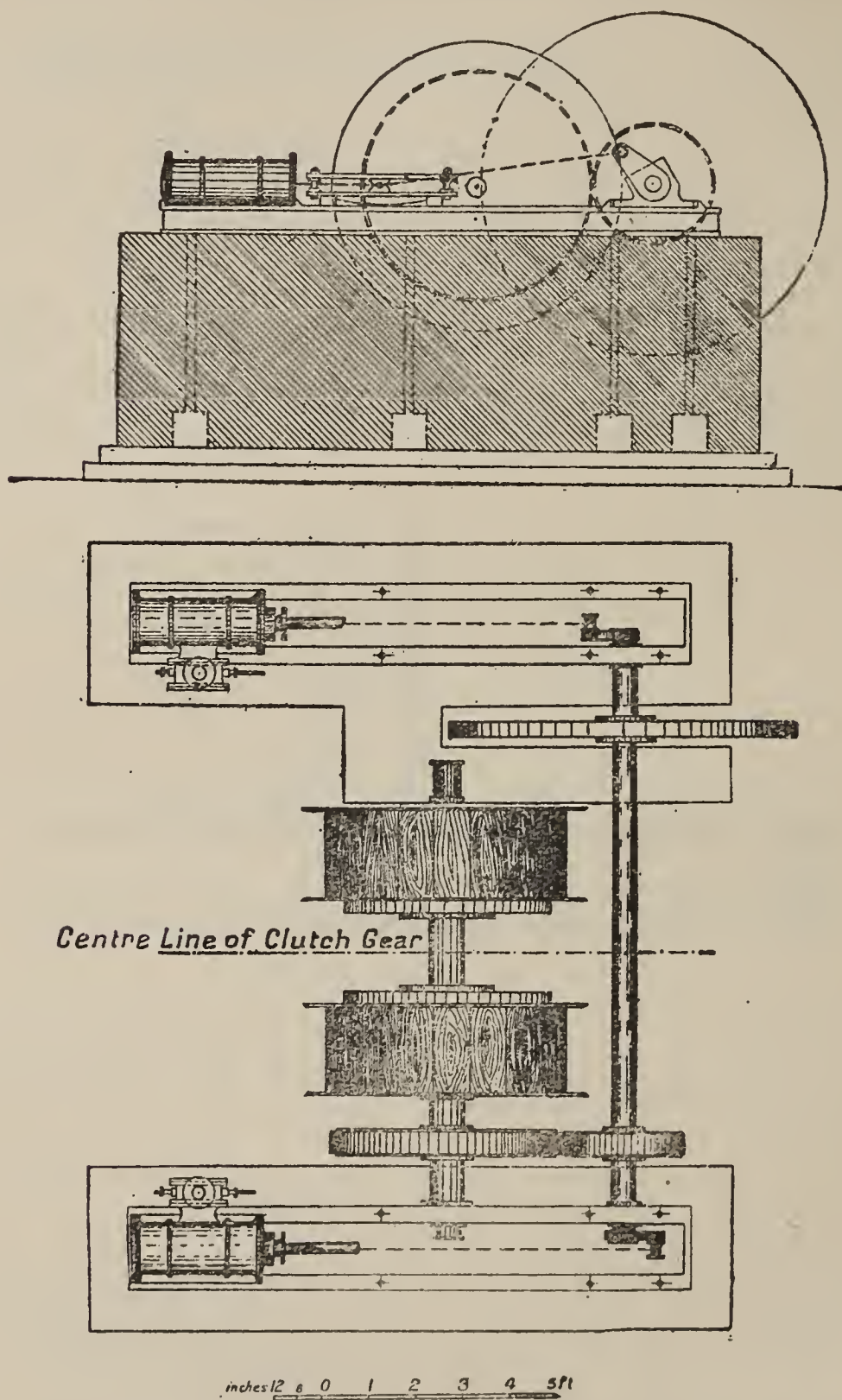
and branch roads can be worked with facility. But against these advantages must be placed the following disadvantages in the working of the system:—

- (1) The high rate of speed at which the rope travels causes fast wear of the rope, and much damage if the tubs get derailed.
- (2) The irregular delivery of tubs at the pit-bottom.
- (3) Larger engines are required to perform a given amount of work, and a larger amount of steam power required than with endless rope haulage.
- (4) The outlay for engine and ropes is greater, as the rope requires to be three times the length of road.
- (5) Greater danger for men and horses travelling on the roads, unless a separate travelling way be provided.

The engines for working the main and tail ropes should have two cylinders, and be geared to the required proportions.

Figs. 441, 442 show the plan and elevation of a good arrangement of engines for this system, both drums being placed on one driving shaft. The engines, as in the direct rope system (and for the same reasons), are better placed underground.

Endless Chain.—This system of haulage is in all respects similar to the endless rope system to be described shortly, although the



FIGS. 441 and 442.—Arrangement of haulage engines for main and tail ropes.

speed is somewhat less and the tubs are attached at closer intervals. It differs from the main and tail rope system, as a double road is required, and the tubs are attached singly instead of being attached in trains; the speed is also much lower, the endless chain being driven at a rate of one and a half to three miles per hour.

The endless chain may be worked either by an engine and wheel above ground, which is the most common arrangement, or by an engine and gearing placed underground.

Chain Wheel.—For driving the chain, a wheel with a broad rim to allow the chain to make two or three laps on it is often used. Sometimes a wheel, such as that shown in fig. 443, is used. It has a broad rim, and a number of 'feet' or 'studs' are screwed into the face which grip the chain and prevent it from slipping. The feet are so arranged that the distance between them is exactly equal to the length of a link. In Briart's wheel for endless chain a series of V-shaped grips or feet of steel are screwed into the rim, as shown in fig. 444. As the links of the chain stretch by wear, the feet can be unscrewed and fixed in a new position to suit the altered lengths of the links.

Sometimes a series of steel blocks are fitted into the circumference of the wheel, and the chain coiled round two or three times to give the required grip. The blocks are secured to the wheel by means of counter-sunk bolts, the part on which the chain works being octagonal in shape, which serves to prevent the chain from slipping.

Lengthening the Chain.—It is often difficult to keep the chain tight, owing to the continuous lengthening due to wear, and a common practice is to allow it to stretch until it becomes necessary to remove a portion to tighten it. It is, however, better to use a 'tightening' carriage similar to that used with the endless rope system.

Working Branch Roads.—Branch roads off the main line can be easily worked with endless chains, and this is one of the chief recommendations of the system. The working of the branches will be understood from fig. 445. In this method three separate chains are used, one for the branch and two on the main road. These chains are worked by a series of bevel-gear pulleys working at right angles to each other. On the shaft to which the horizontal pulleys are attached, a driving wheel is fixed round which the chain passes. The whole of these wheels and gearing are placed below the level of the roadway and boarded over. At a short distance on either side of the branch the tubs are detached from, and again attached to, the chain by an automatic arrangement which is shown in fig. 445. Travelling in the out-bye direction the chain is gradually raised to near the roof by a large pulley working on a shaft (fig. 445); a

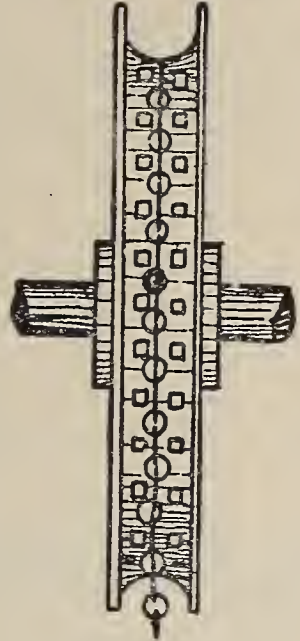


FIG. 443.—Chain wheel.

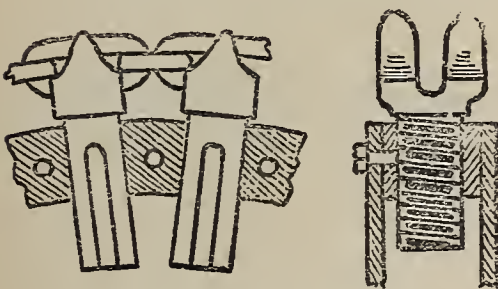


FIG. 444.—Briart's wheel.

small roller pulley guides the chains to this larger pulley, placed horizontally. The road for the loaded tubs is given an up gradient towards the shaft, and the empty road has a dip in the opposite direction. These gradients commence a short distance on each side of the branch. By the rope rising towards the roof and the road dipping the chain is raised out of the gripping fork; the tub being released runs over the part where gearing is placed, and re-connects itself to the chain again automatically. Exactly the same arrangement is required on the empty or in-going chain.

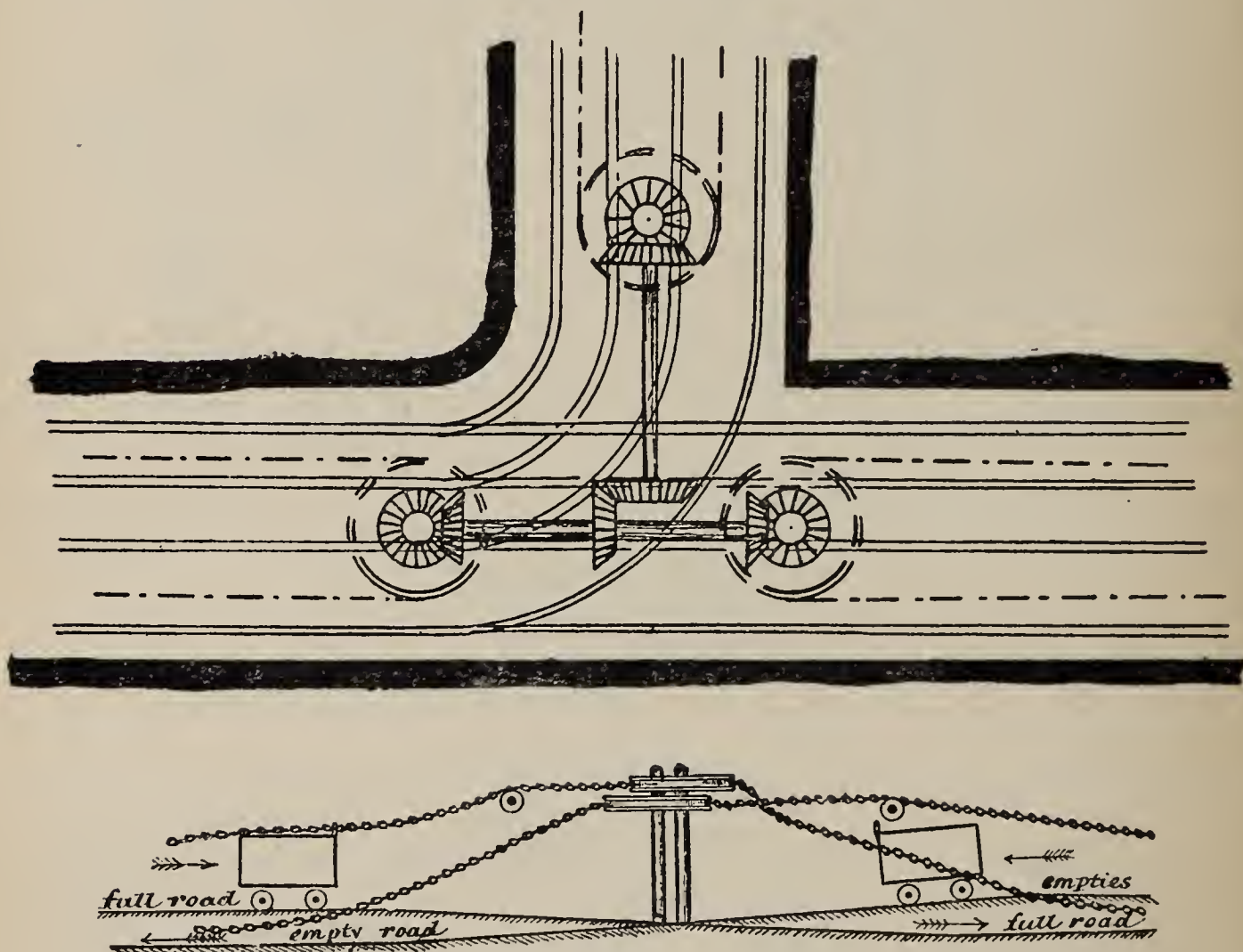


FIG. 445.—Working branches.

Branch roads may also be worked by a series of wheels fitted on to a driving shaft placed vertically, the branch wheels being arranged so as to work with clutches to enable them to be thrown in and out of gear as required. The main driving wheel is keyed tightly to the shaft. This arrangement allows of a branch chain being arrested if anything goes wrong, or if there are not sufficient tubs to keep it constantly moving.

Working Curves.—If a curve is one of short radius, a pair of large wheels are placed at the bend round which the chain passes (fig. 446). The tubs are detached automatically before coming to the curve in the manner already described for working branch roads. When they

pass round the bend they again attach themselves automatically to the chain.

Method of attaching Tubs to Chains.—This is usually done by fixing to one end of the tub an iron fork, into which the train drops. When wood tubs are employed, a fork is used, being fixed by means of two nuts and bolts. With iron tubs the grip is a part of the tub itself, and requires no fixing.

Sometimes movable grips are used to adjust the height of the chains according to the quantity of coal loaded above the level of the sides of the tubs.

The chief advantages of the endless chain system are the slow speed at which it travels, and the small amount of wear and tear in rolling stock entailed, besides the small cost of upkeep of roadway. The



FIG. 446.—System of working curves.

principal disadvantage is the heavy weight of chain to be driven where the haulage is long.

This system is better suited for surface haulage than for underground working, many such arrangements being at work for conveying coal or other material long distances, as when a colliery is so situated that it would be impossible or inconvenient to connect it with the railway by means of a branch line.

Endless Rope.—This system, as its name signifies, consists of an endless rope travelling in a double roadway, and to which the tubs may be attached either singly at intervals along the rope or in trains. The engine for driving the rope is almost invariably placed on the surface, and the power either conveyed direct or by a 'band' rope reaching to the pit-bottom, where it drives a main shaft from which the power is derived for other endless ropes. The general arrangement in endless rope haulage (fig. 447) is as follows:—At the mouth of the shaft

are two pulleys for carrying the rope from the driving wheel into the shaft. At the pit-bottom are two other pulleys, placed vertically to receive the ropes, and immediately below these are two other pulleys, placed horizontally to enable the rope to make a right angle with the shaft, and pass into the workings to the inbye end, where it passes round a loaded wheel *d* or an ordinary pulley placed horizontally.

In another arrangement the loaded wheel or tightening carriage is placed on the empty rope side near the pit-bottom. This is undoubtedly the best position.

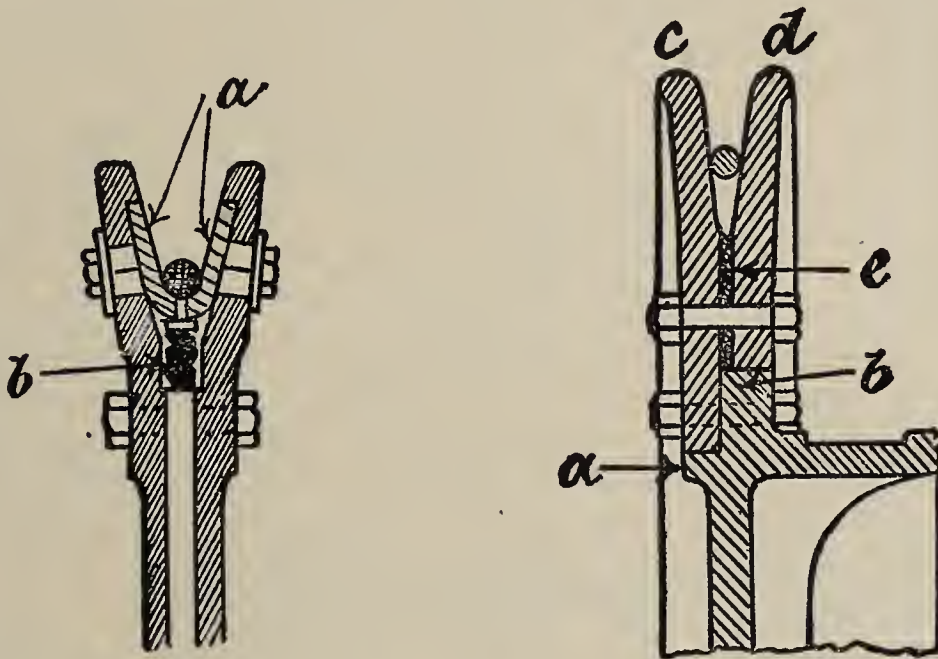
FIG. 447.—Endless rope haulage arrangement.



Arrangement of Engine.—For endless rope haulage, where a large quantity of coal has to be drawn, it is best to employ a double cylinder engine well geared down. With a modern haulage plant for endless rope, the two cylinders are placed horizontally, and the piston rods connected to disc cranks fitted on to the main driving shaft. On this shaft are two small geared wheels, working into two larger toothed wheels, which are keyed on to a separate shaft, on which is also fixed the driving pulley or pulleys, the number of which varies with the number of ropes to be driven. These pulleys are now usually arranged with clutch gear so that they may be rapidly arrested without bringing the engine to a standstill. As to whether the power should be conveyed through a band rope or not, there are differences of opinion. Where more than one endless rope is required to work different sections, the band rope method is to be preferred. If four different sections are worked by endless rope, and each separate rope is worked direct from the surface, this would necessitate four

double lengths of rope being conveyed down the shaft, which, owing to the complications that might arise in the event of any of the ropes breaking or getting entangled with each other, would be

altogether undesirable. On the other hand, when a band rope is used only a double length of rope requires to be led down the shaft. It must not be forgotten, however, that as the whole of the power must be conveyed through the band rope, this would require to be a good deal heavier, and therefore more expensive than the haulage ropes worked from it, while there would also be the increased expenditure for the clutch arrangement at the pit-bottom. With the band rope system, any number of ropes can be worked, according to the power available. They are under the direct control of the attendants at the shaft bottom, who can stop any of them without signalling to the surface, which is a great advantage, as one section may not have sufficient coal to keep the endless rope constantly at



FIGS. 448 and 449.—Barraclough's pulley.

work, and such rope can therefore be thrown out of gear, while the other sections go on as usual.

For a very deep shaft the cost of the band rope becomes, on the other hand, a serious item, although even then it would be less expensive than taking all the ropes up to the surface. Everything considered, the band rope system of conveying the necessary power is much to be preferred.

Driving Pulleys.—The rope is usually actuated by a clip pulley. There is a considerable number of different types of pulley in use. The Barraclough pulley is a well-known device for endless rope haulage. Round the periphery of the wheel and opposite to each other are fixed a number of short taper clips fitted to receive two sliding jaws *a a* (fig. 448), upon which the rope works. These jaws rest on springs *b*, so that when the rope comes on to them, the weight forces them down on the springs, and narrows the opening between them, thus giving the necessary grip to the rope. At the point where the rope leaves the pulley the springs give assistance in releasing it.

Another pulley, much used in Scotland, is shown in fig. 449. The rim consists of two segments *c* and *d*, bolted together. Between these segments is placed a layer of wood *e* on which the rope works. The opening on the rim of the pulley being V shaped, the rope obtains the necessary grip by wedging itself at the bottom of the opening.

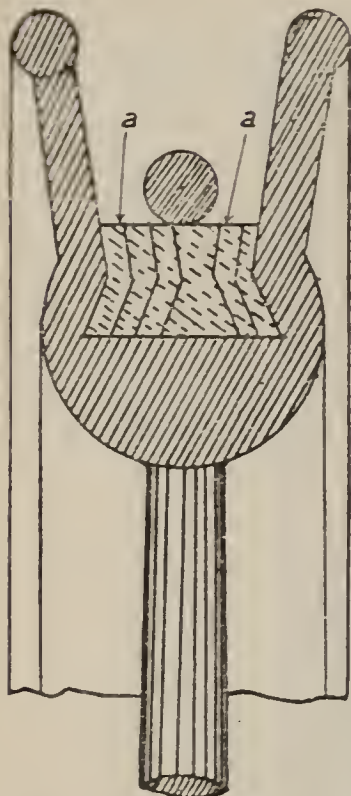


FIG. 450. —Method of gripping rope.

Owing to the low coefficient of friction between iron and iron, and also to the wear of the rope in unlined pulleys, it has become customary to pad them with softer material than iron, to increase the gripping power and so increase the life of the rope. Segments of hard wood are used, but require frequent renewal, and soon lose their gripping power. Segments of india-rubber have been tried with good effects, and last a considerable time. Possibly the best substance for this purpose is well-seasoned leather driven into the rim of the wheel in segments *a a* (fig. 450), and turned true. This is said to give a good gripping surface that will last from two to three years. A pulley now very largely used for endless rope haulage is that known as the Clifton, frequently called the C-pulley. This pulley has a wide rim or throat into which loose cast-steel segments are bolted (fig. 451). The segments form a small inclined plane, having a gradient of 1 in 12 to 1 in 16. There are generally two and a half turns of rope allowed on the rim, the rope entering at the top of the incline and moving to the bottom as each turn of the rope goes off. By this arrangement the friction and the gripping power of the rope are greatly increased. When a large amount of power requires to be transmitted, grooved pulleys and counter-pulleys or lacing wheels are sometimes used to impart the necessary friction to the rope. Both pulleys have from four to six grooves in the rim, the driving wheel *a* having one more than the guide or counter-pulley *b*, which is set immediately in line with the other (figs. 452, 453).

Taking up Slack Rope.—No matter how perfect the driving pulley on the surface may be, some arrangement for taking up slackness will require to be adopted. The usual plan is to have the rope passing round a tightening pulley.

There are various ways of arranging this pulley. Sometimes it is fixed as shown in fig. 454, and fitted with a long screw and nut,

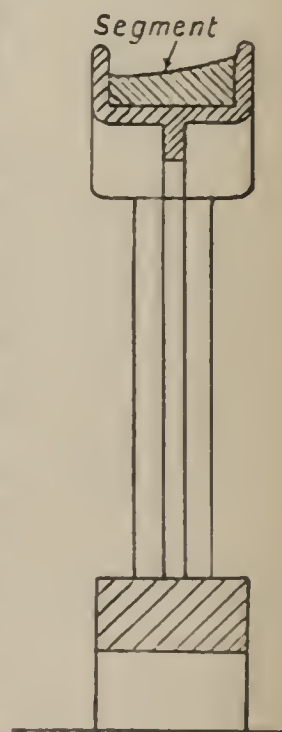
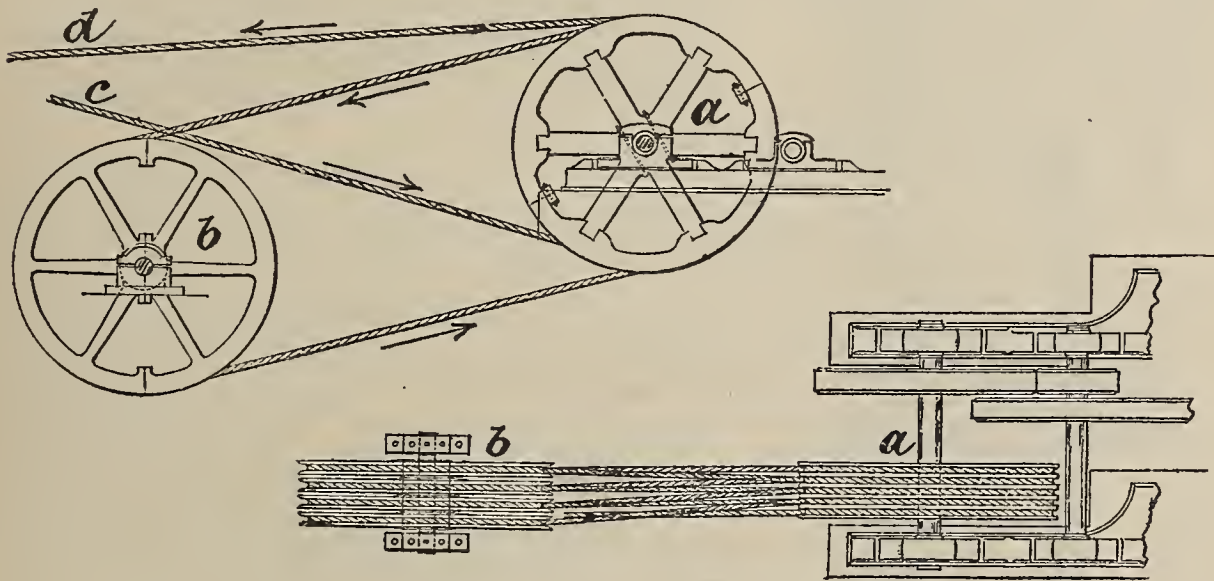


FIG. 451.

which can be adjusted as required. But this system of tightening is not at all to be recommended. What is required is some arrangement that will automatically adapt itself to the varying load and wear of the rope. To accomplish this, a wheel is usually mounted on a tension carriage to which is attached, by means of a chain passing over a pulley, a weight (see fig. 455) working in a small pit below the level of the road. Sometimes the tension-carriage is attached to a loaded tub, working on an inclined plane. The weight required can easily be found when the rope is set to work. The tension pulley is usually placed as near the pit-bottom as possible, although it may also be placed at the further end of the road. It is, however, often very inconvenient to have it in the latter position.

Speed of Rope.—With endless rope haulage, the speed at which



FIGS. 452 and 453.—Grooved pulleys.

the rope travels may vary from one and a half to four or five miles per hour, but the best results are obtained when the rope is travelling at a speed of from two to three miles per hour.

Working of System.—The system can be applied either: (1) with the rope travelling over the tubs; or (2) with the rope moving beneath them. The tubs are also attached to the rope in two different ways: (a) singly at specified distances apart, by means of 'clips' or 'jiggers'; (b) in sets or trains by means of a 'clip-bogie' and gripper.

The first method is undoubtedly the better of the two, as by this system the supply of coal at the pit-bottom is regular, and the load on the engine more evenly distributed, while no attendants are required except at each end of the road or at branches.

Some, however, prefer the 'clip-bogie system,' as being safer, and better adapted for working a number of branch roads. It is somewhat more expensive than the first-named method. With either system a double road is generally employed, although sometimes a

single road is used with pass-byes at intervals. When such is the case, the tubs are run in trains. It is often urged as a defect of the endless rope system, that it requires a double road, the construction of which is often difficult and expensive, especially where there is a bad roof. But a double road is not at all essential for the successful working of an endless rope system of haulage.

At the Palace Colliery and Bent Colliery, Hamilton, systems of endless rope haulage have been successfully at work for years, in both cases arranged on the single road principle. In the first the rope passes under the tubs, while in the second it is overhead, the latter being probably the more successful method.

At a short distance from the shaft are set two parallel single roads, from different sides of the pit-bottom. These two roads are carried for the required distance, and are connected by a cross drift into which the rope passes by means of two wheels. These two parallel roads may be as wide apart as required. The empty rope and empty tubs leave the pit-bottom at one side, while the full rope and loaded tubs arrive on the opposite side. This system has a great advantage over that where the empty and loaded tubs alike require to be handled at the same side of the shaft. The rope can be easily extended into the workings.

At the Bent Colliery much the same system was adopted, but the two roads, instead of being driven parallel, were taken along a more circuitous route, practically to follow the faces, which enabled the tubs to be brought on to the rope with very little secondary haulage. The empty tubs can be taken off, and full tubs attached at any desired

part of the road, and if the empty tub is not required, or is not taken off the rope in the workings, it will return again to the pit-bottom on the side opposite that from which it left.

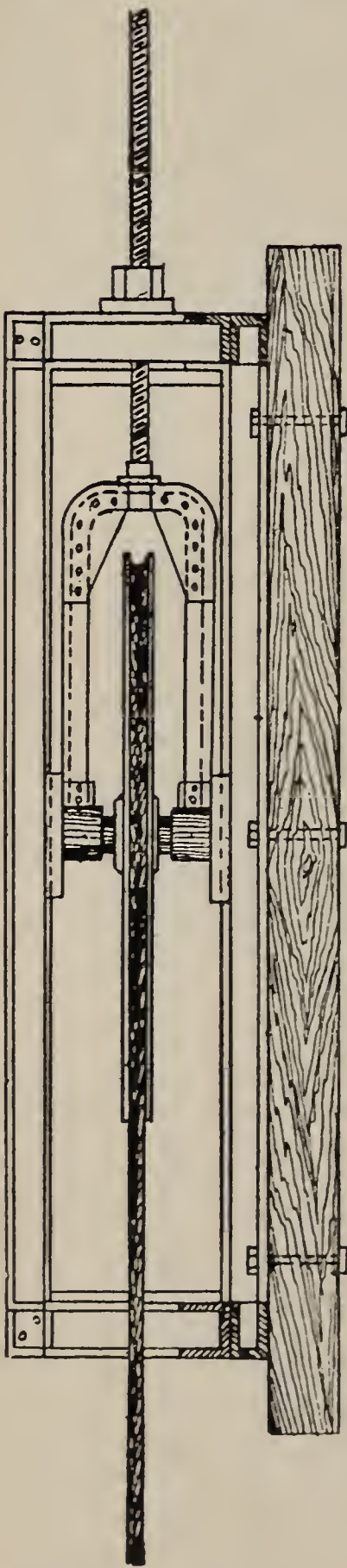


FIG. 454. — Fixed tightening pulley.

Clips or Jiggers.—When the tubs are attached singly to the rope,

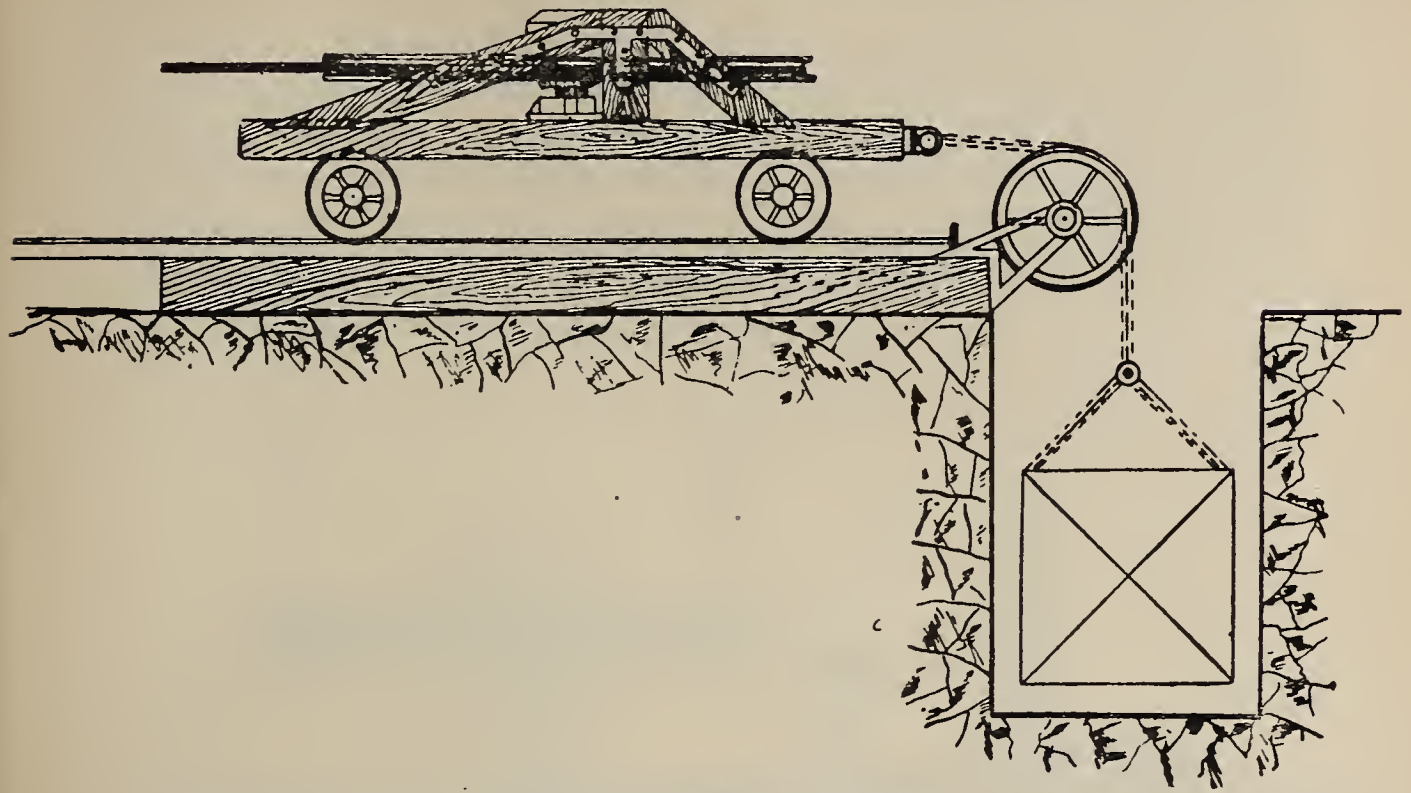
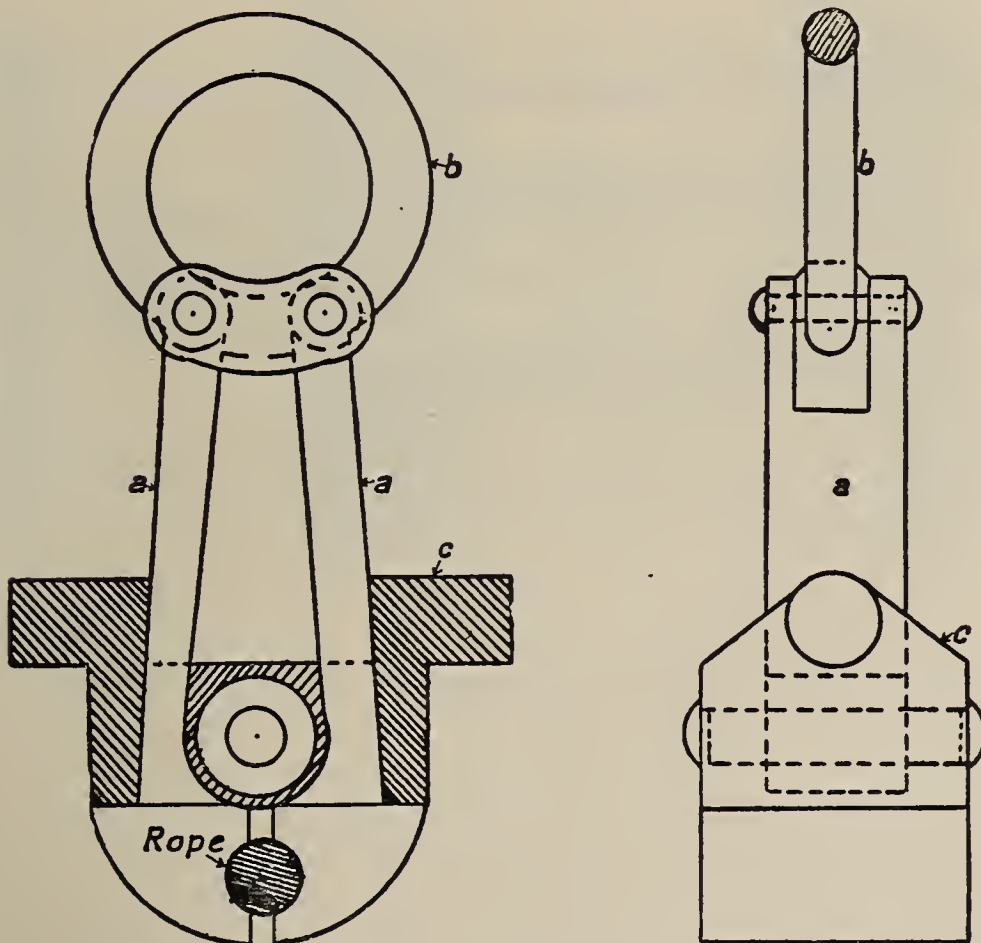


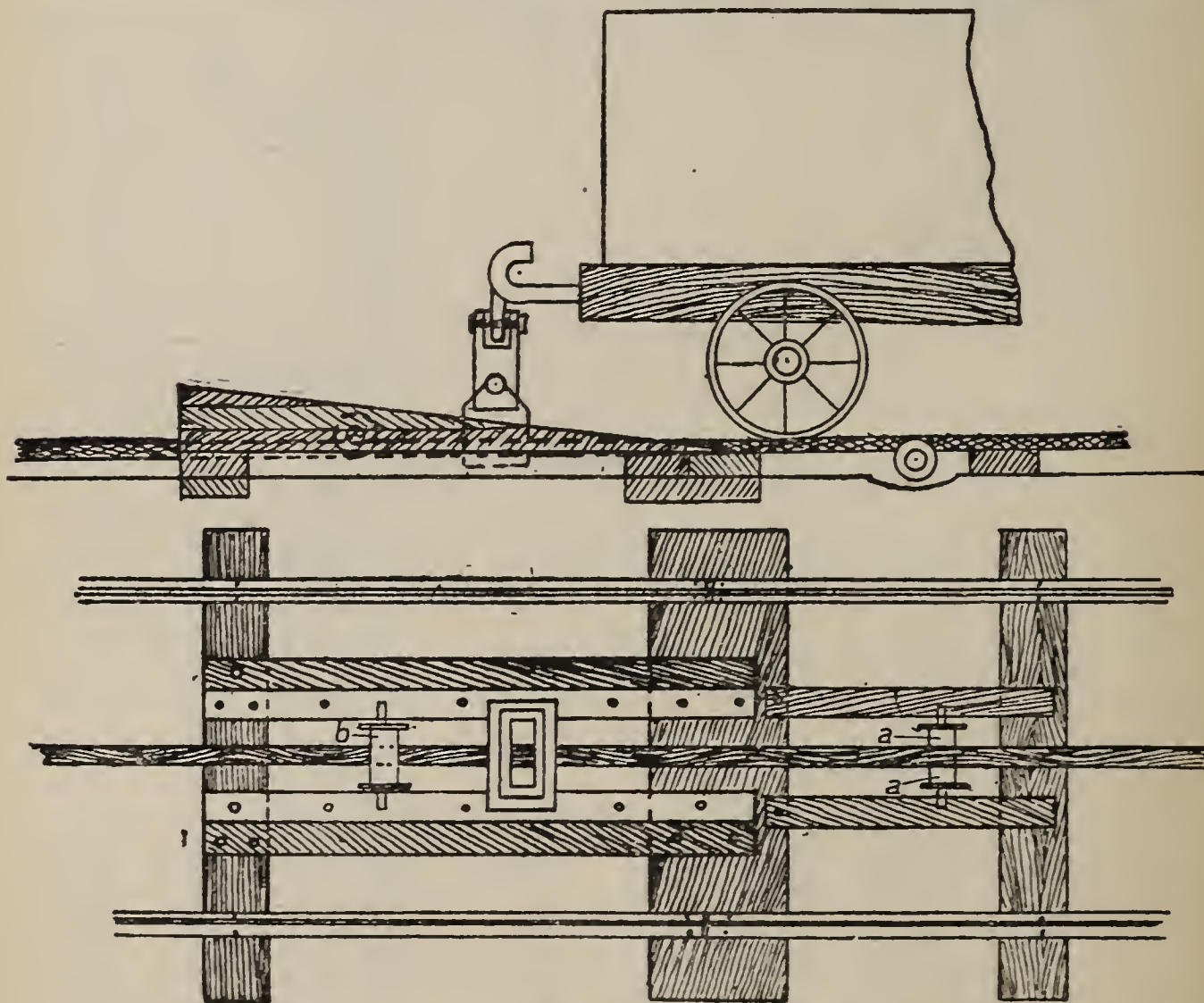
FIG. 455.—Tension carriage.



FIGS. 456 and 457.—Clip or jigger for endless rope.

a clip or jigger is used. A great many different sorts of clips are in use in different districts, each having its own special merits.

A clip used in Scotland a good deal for under-rope haulage is shown in figs. 456, 457. It consists of a pair of jaws *a a*, which are attached to a large link *b*, the latter being in turn attached to the tubs. Over the two jaws *a a* passes a cone or thimble *c*, which, on being driven hard down, presses the two jaws together and grips the rope, which is caught in a groove at the bottom. This clip is easily attached to and detached from the rope, and is very simple in construction, but it is rather bad for the rope, and is best suited for



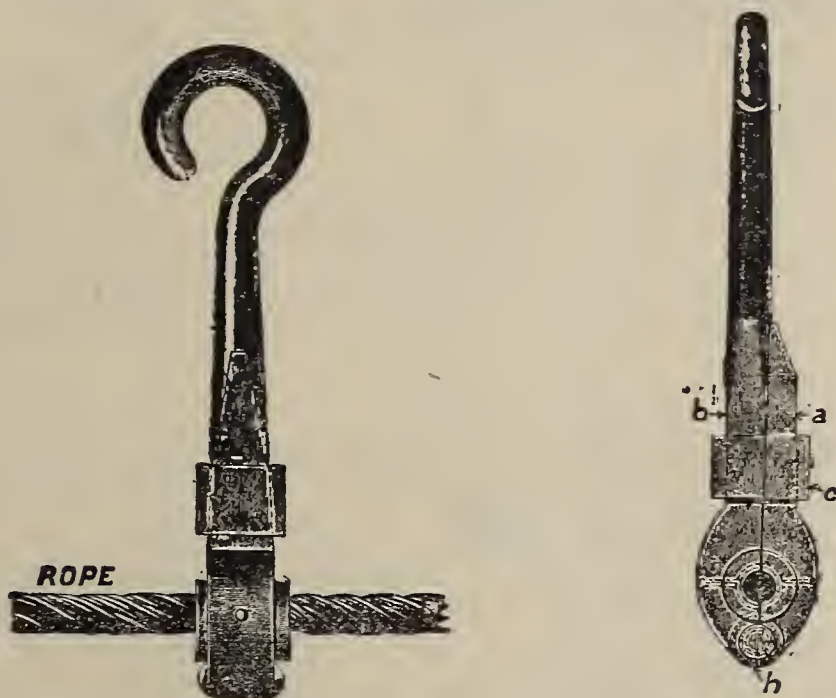
FIGS. 458 and 459. —Automatic detaching arrangement.

comparatively level roads. To detach the tubs automatically from the rope, when this clip is used, the arrangement shown in figs. 458, 459 is adopted.

At the in-bye and out-bye ends of the road a small inclined plane, with an opening in the centre for the rope to travel in, is made in the centre of the roadway. This opening is just sufficient to admit the rope and the bottom of the clip, the cone or thimble edges being caught on the inclined plane, which rises until it is sufficiently high to raise the cone, and allow the jaws of the clip to open and so release the rope. To prevent the latter from rising, two pulleys are fixed at each end of the inclined plane, the rope passing over the

pulley *a* and under the pulley *b*. When the clip has been in use for some time it gets worn and fails to grip the rope properly, so that if the thimble becomes at all slack the tub may come to a standstill, and the rope continue to run through freely. This causes much undue wear, but can be avoided by using steel lining pieces which can be fitted into the jaws of the clip and renewed when required. This clip has been known to work successfully on roads with a gradient of 1 in 7.

Another clip employed is that known as Fisher's patent, which is illustrated in figs. 460, 461. As will be seen, it also consists of two jaws *a* and *b*, the part *a* working on a pin *h* at the bottom of the clip, which enables it to fold over the rope. The two jaws are held



FIGS. 460 and 461.—Fisher's clips.

in position by a sliding thimble *c*, as described above. The clip can be automatically detached in a similar manner to that described for the first clip. A different arrangement of jigger and automatic detachment is shown in fig. 462.* The detaching apparatus is rather more complicated than that already described. It consists of a horizontal axle *a*, carrying two arms *b b'*, of equal length. One of the arms, *b'*, has a circular disc which passes up through the centre of a split rail, and projects about $5\frac{1}{2}$ in. above it. The tub wheel on passing depresses this arm $5\frac{1}{2}$ in. and at the same time raises the other arm an equal distance. This other arm *b* carries a steep cone roller *c*, which is raised with it, and thus raises the rope and slips the cone down clear of the jigger altogether.

Other clips, such as Smallman's, Hanson's, and Humble's, are also used for attaching the tubs for under-rope haulage. The Smallman clip is a very good one for under-rope haulage, and can be used on

* *Trans. Inst. Min. Engs.*, vol. viii. p. 377.

roads of gradients 1 in 4 or 1 in 3. With overhead haulage a different sort of clip is required. The simplest kind for this purpose

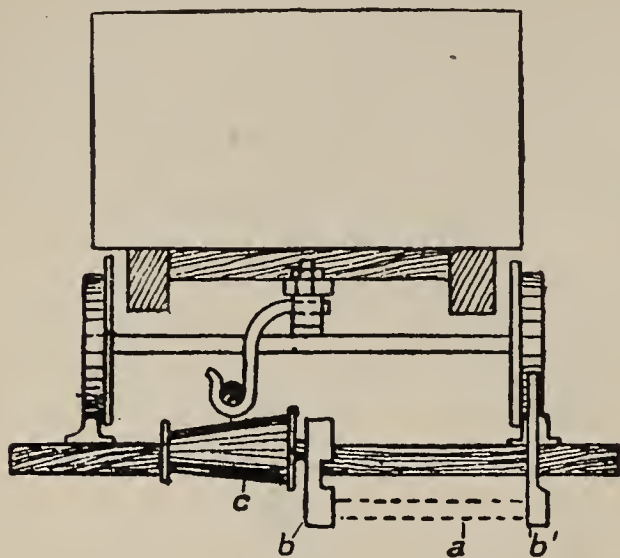


FIG. 462.—Automatic detacher.

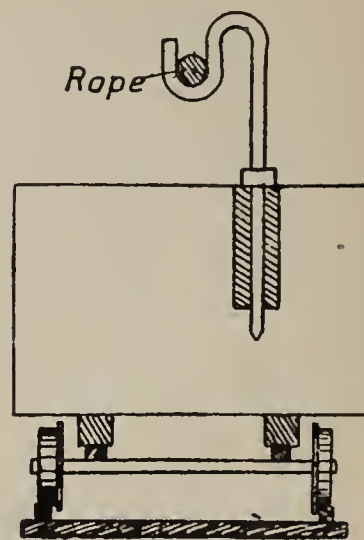
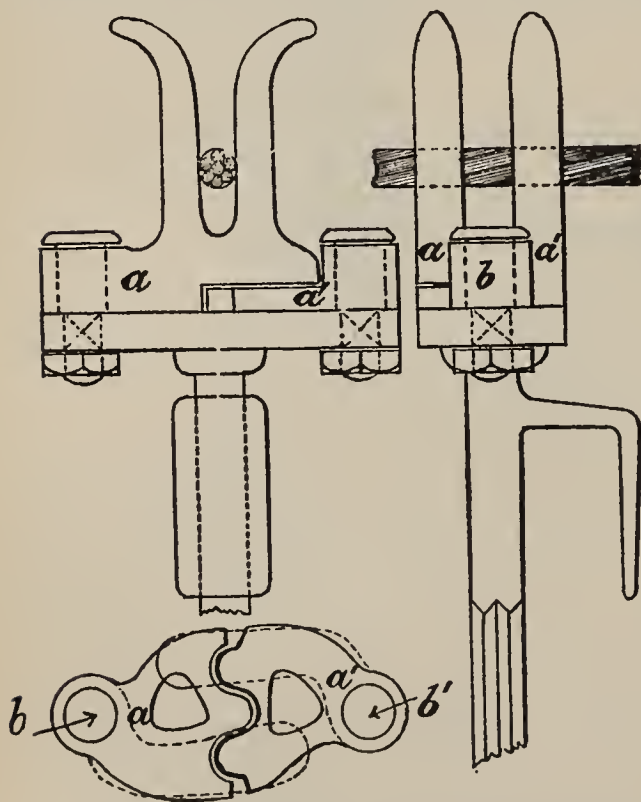
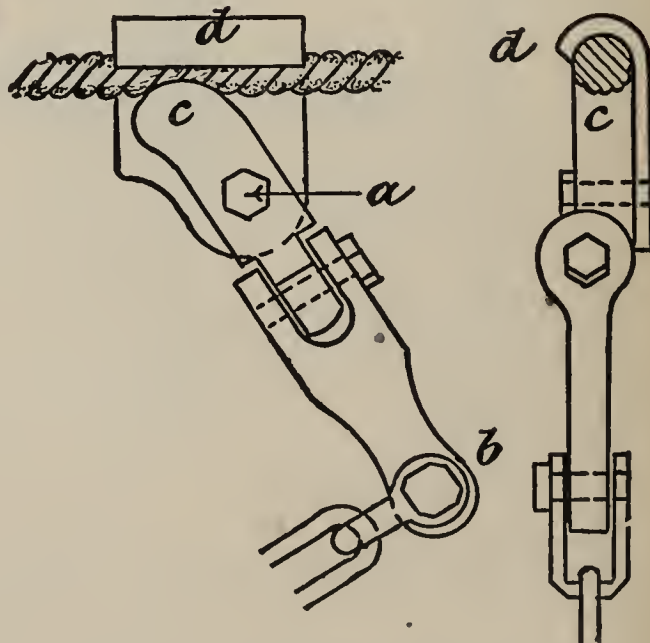


FIG. 463.—Rope clip.

is that shown in fig. 463. It is placed either in the centre or side of the tub, and on gripping the rope it gives it a slight twist, which gives it the necessary hold to carry the tub along. Sometimes two



FIGS. 464 and 465.—Rutherford and Thomson's clip.



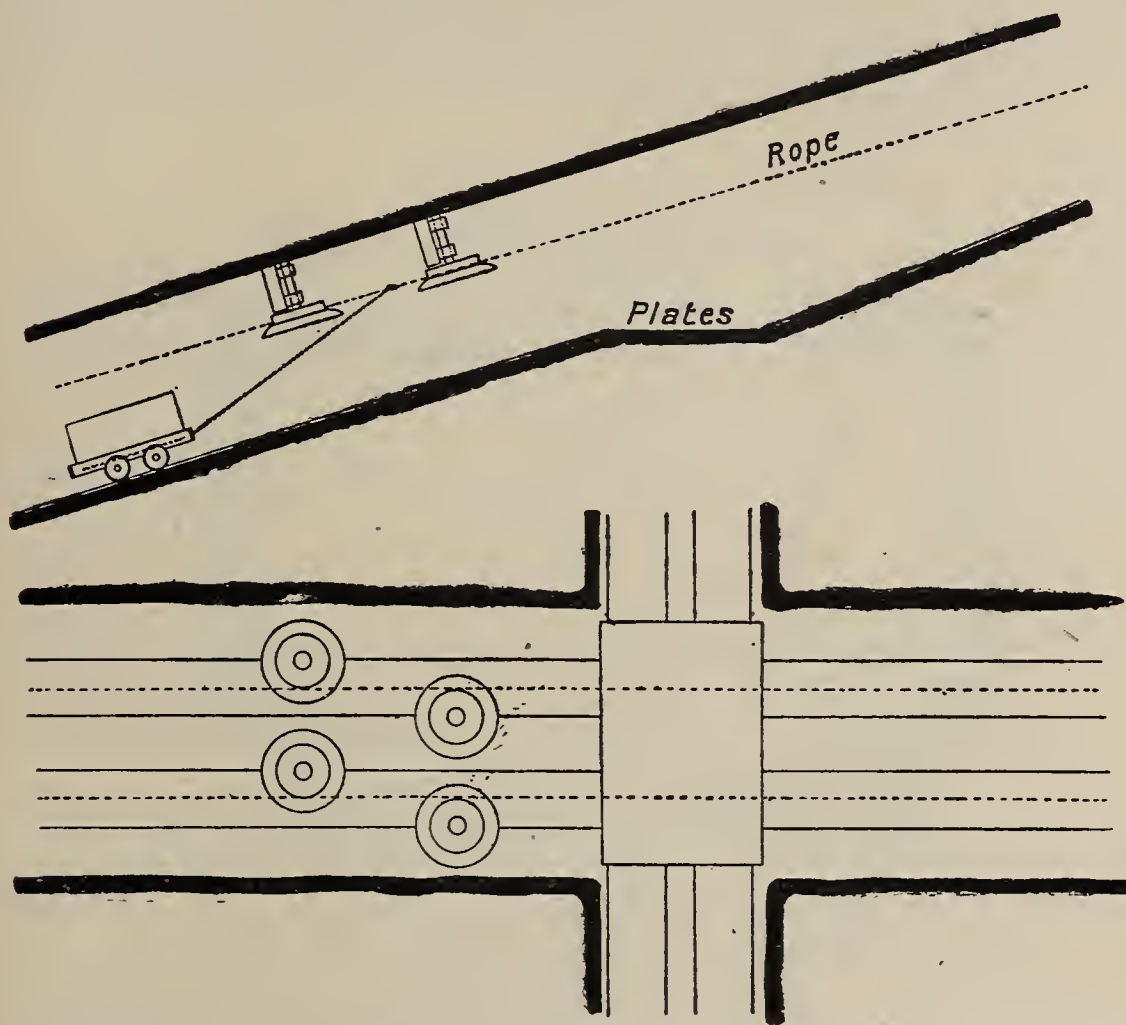
FIGS. 466 and 467.—Ward and Lloyd's clip.

of these jiggers are used, one at the back and one in front of the tub.

Rutherford and Thomson's clip, as shown in figs. 464, 465, is also used for overhead rope haulage. This clip is an application of the principle of the toggle joint, and its grip on the rope is equally firm in whatever direction the load may act. It is equally well adapted

for steep inclinations as for roadways of varying gradients. With this clip, curves and junctions can be worked automatically on the gravity principle in the same way as in endless chain haulage.

Ward and Lloyd's clip,* as described by Mr H. W. Hughes, seems to be very simple and efficient for overhead rope attachment. It consists of a hinged lever, to the bottom of which is attached the chain fastened to the tubs (figs. 466, 467). The lever works about a pivot *a*, and when the weight comes on the end *b*, the rope is gripped between the top end *c* and the curved plate *d*. The lever



FIGS. 468 and 469.—Plan and elevation of hanging-on bench showing position of mushroom pulleys.

is hinged, which allows the clip to fall into the guide pulleys when passing round curves. This grip is best suited for roads having a regular inclination, otherwise two clips will have to be used for each tub.

One of the simplest attachments for overhead haulage is a length of chain, one end of which is attached to the tub and the other to the rope, as shown in fig. 468. The length of chain required depends upon the gradient of the road; for 1 in 3, a common length is 8 or 9 ft., and for 1 in 2, 11 or 12 ft. The end of the chain to be attached to the rope is fitted with a hook, similar to an ordinary coupling

* *Text-book of Coal Mining*, Fifth Edition, p. 271.

hook, and in attaching the tub the end of the chain is given $2\frac{1}{2}$ turns over the rope, and then gripped by the hook.

At hanging-on intermediate benches the rope is kept at the requisite height above the floor by a series of four mushroom pulleys, over which the chain attachment can glide without displacing the rope from the pulleys. This method of attaching the tubs is best suited for roads having a gradient in one direction, *i.e.* against the load tubs, and it can be successfully worked on roads with a high gradient, the author having seen roadways with a gradient of 1 in 2 worked on this system. If the attachment to the rope is properly made, runaways are very rare. The attachment for the empty tubs going downhill is made from behind

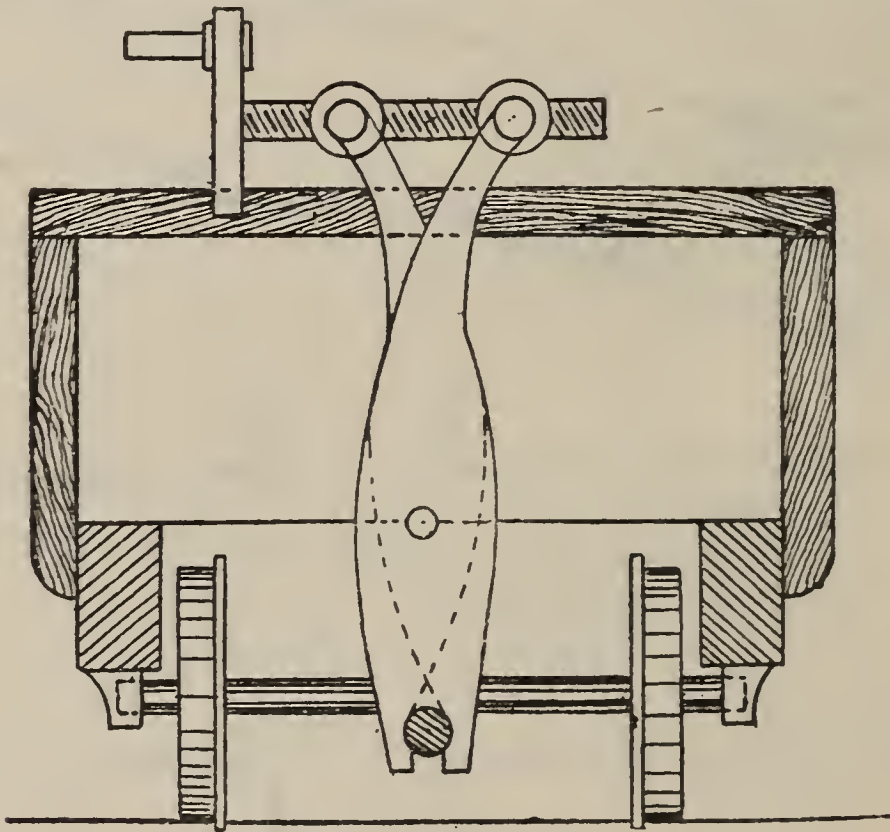


FIG. 470.—Bogie clip.

instead of in front, so that the chain may be kept taut. Figs. 468, 469 show the arrangement of the mushroom pulleys at an intermediate hanging-on level or bench.

Running the Tubs in Sets.—When the tubs are attached to the rope in trains or sets, a bogie with a gripper or shears is used.

Fig. 470 illustrates a form of bogie clip which is much used for this purpose. The gripper swings on a pivot which enables it to pass round level pulleys at curves; it is actuated by a screw and handle in much the same way as the brake screw of a railway van. On inclined roads these clip bogies are usually loaded with scrap metal in front to keep them from 'rearing' and getting derailed.

Catch Blocks.—On inclined roads it is often the custom to have catch blocks or runaway catch points to arrest the tubs should they happen to become detached from the rope. A simple form of catch

block is shown in fig. 471. It consists of a piece of wood fixed on a spindle, the latter not being in the centre. When the tub passes over it the wheel axle depresses it to a horizontal position, but as soon as the tub is passed it assumes its inclined position again, and will be ready

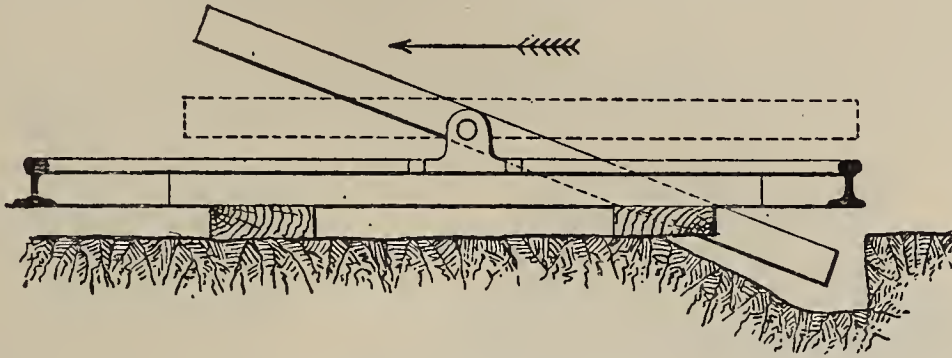


FIG. 471.—Catch block

to grip the tub should it run back. The objection to this type of catch block is that the rope may ride on it and depress it, so that if a tub breaks away it may pass over the catch without being stopped. To obviate this difficulty, catch blocks may be fitted over the rails, as shown in fig. 472. In this arrangement the blocks move on a pin at the lower end, while the upper end is attached to a strong side spring.

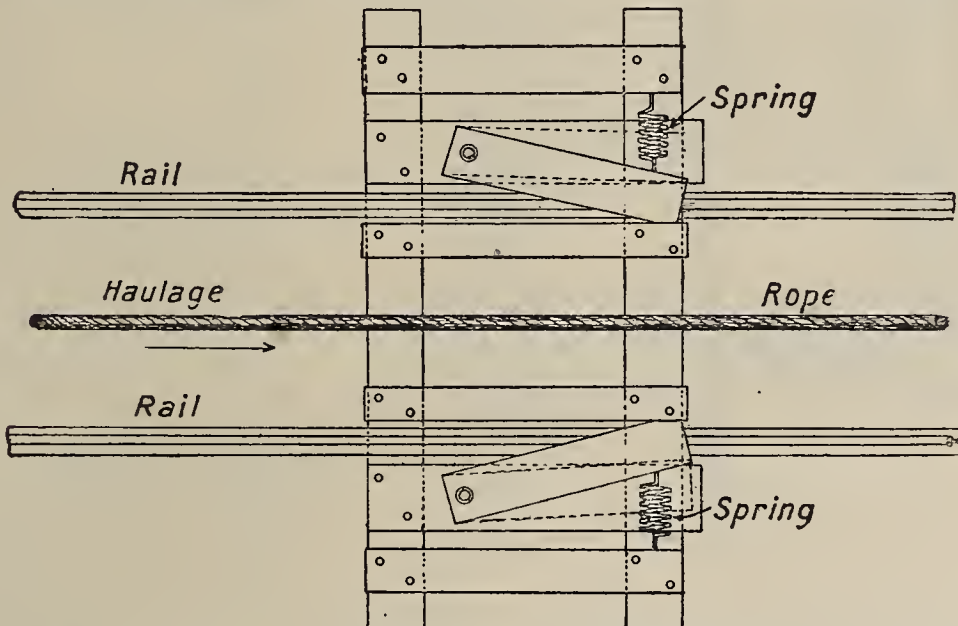
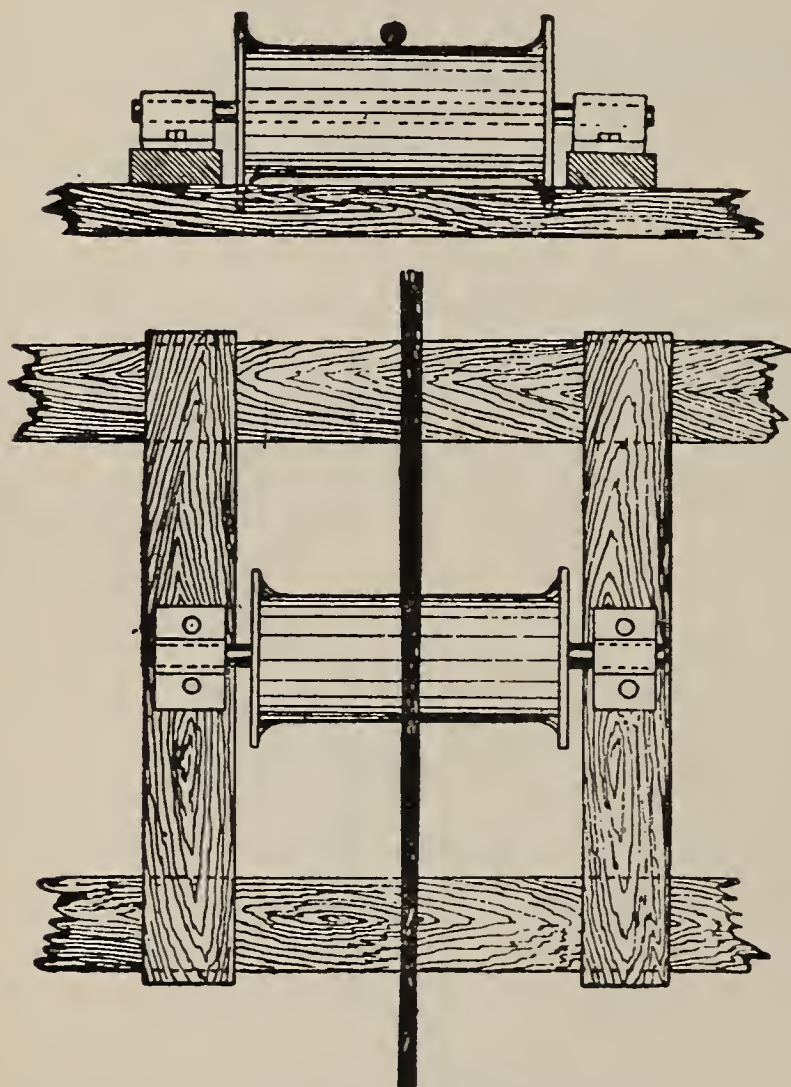


FIG. 472.—Automatic runaway blocks for endless rope haulage road.

The tub in passing opens the block, depressing the spring, and immediately the tub has passed through, the catch block is pressed back into its original position over the rail, so that if a tub breaks away it is caught. Runaway points fixed to the rail, and having a strong spring, are used for the same purpose. The tub on passing opens the points, which close of themselves when it has passed through.

To get the best results from endless rope haulage, the rope should be carefully guided by means of rollers to prevent it from rubbing on the floor and to keep the sleepers from getting cut through. These

rollers may be made either of wood or iron, and properly fixed so as to run freely, a point which is very often neglected, giving rise to much unnecessary friction and wear of the rope. Figs. 473, 474 show the method of fixing these pulleys on haulage roads. When the rope has to be taken round curves it must be guided by means of pulleys. The radius of the curve should be made as large as possible, as in quick bends the friction between the rope and the guiding pulleys becomes excessive, wearing both rope and pulleys rapidly. Where



FIGS. 473 and 474.—Roller pulleys.

curves have to pass round a right angle (90°) it is preferable to guide the rope round such a curve by means of one large pulley instead of a number of small ones. The objection to this system with under-rope haulage is that the tubs require to be detached and the roadways arranged that the tubs will gravitate round the curve clear of the guide pulley and then re-attach again on the straight part. With curves of 90° and larger radius, a common system is to use a number of small bevel pulleys, figs. 475, 476, 477, 8 to 12 in. diameter. These pulleys should be set at such distance apart as will prevent vibration

in the rope between the points of contact. The best arrangement would be to place the sheaves so close together that the periphery of each sheave would strike the line of curve. It may be objected that this entails an unnecessary number of sheaves. The pressure on the pulleys should be distributed so that it will not exceed half the breaking strain of a single wire of the rope on each pulley. The number of pulleys required to meet these conditions may be found from the formula $N = \frac{P}{B}$,

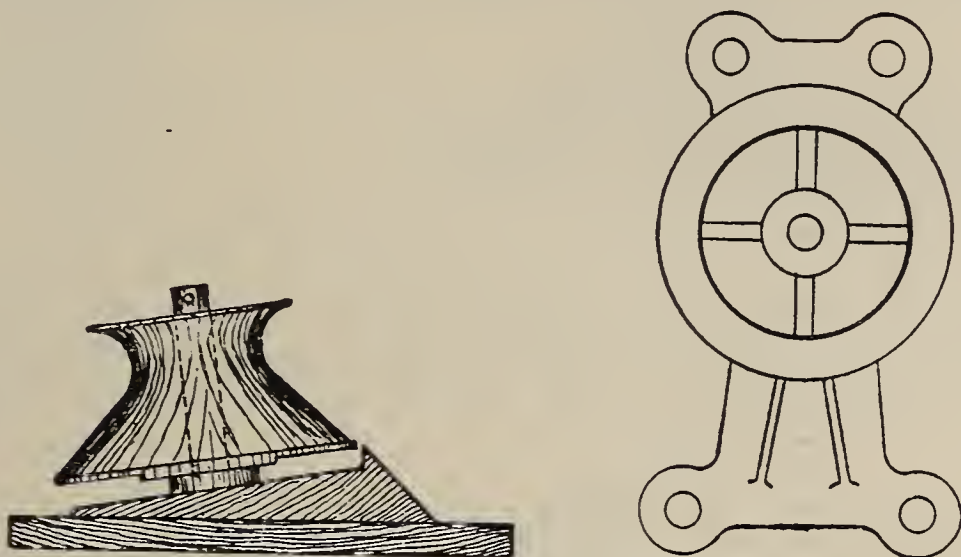
where N = number of pulleys required,
 P = total pressure on curve,
 B = half the breaking strain of a single wire.

Example.—Assume a curve of 90° with a radius of 20 yds., and that total pressure on curve is 5 tons; plough steel rope say $\frac{3}{4}$ in. diameter. Each wire would = .084 in. diameter, and breaking strain = 1240 lbs. Total pressure on curve = $5 \times 2240 = 11,200$ lbs.

$$\text{Then } N = \frac{11,200}{\frac{1240}{2}} = 18 \text{ approximately,}$$

and the distance between the pulleys would be = $\frac{20 \times 3}{18} = 3.3$ ft.

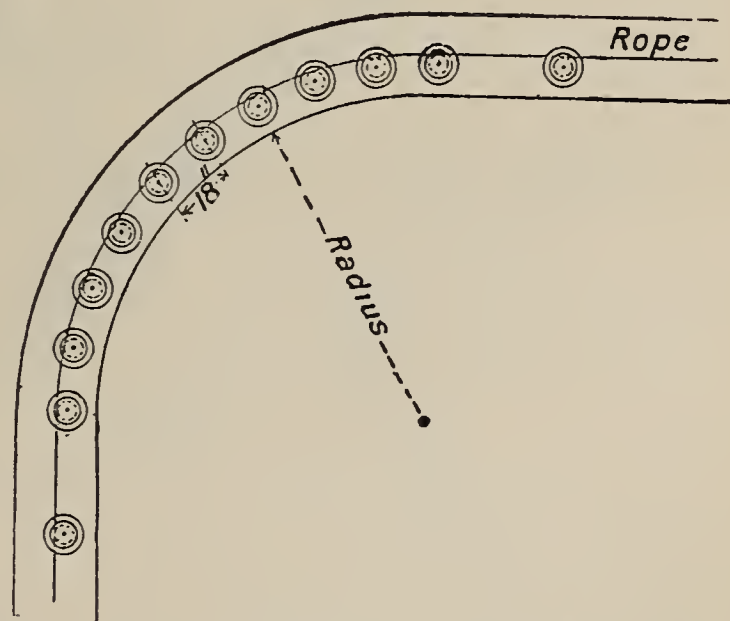
Endless Rope on Inclined Roads.—It has often been urged that



FIGS. 475 and 476.—Bevel pulleys.

endless rope haulage is only suitable for flat or comparatively flat workings; and while no doubt the best results are obtained under such conditions, still the system can be successfully worked at inclinations of 30° or more.

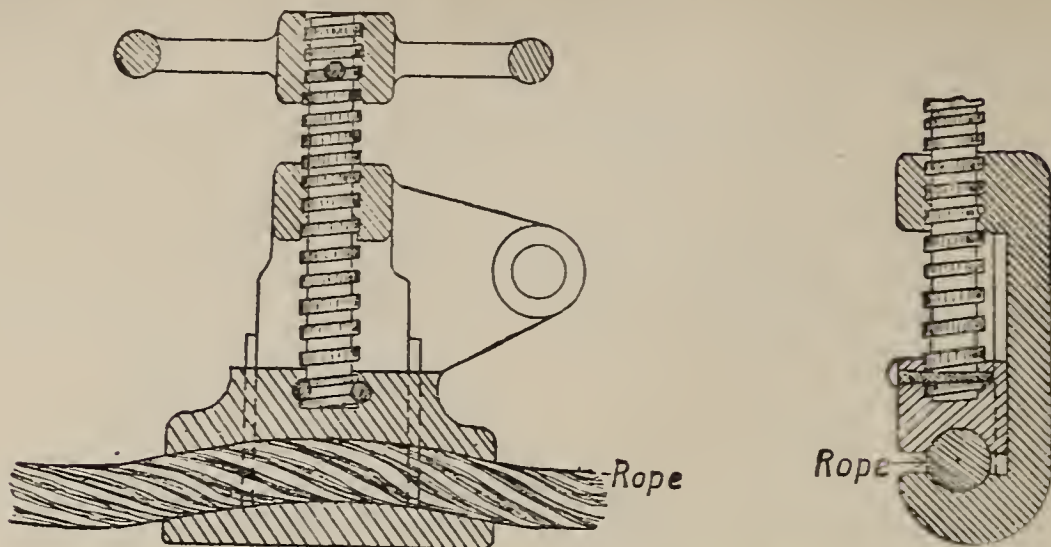
At Moston Colliery, Manchester,* there is a good example of this system of haulage on steep gradients. The road in question is 2900 ft. long, the inclination varying from 26° to 39° . The rope travels at a speed of $1\frac{1}{2}$ miles per hour, and runs beneath the tubs, which are attached to



it by means of a stout parallel jaw-clip. The jaws of this clip are actuated by a screw and hand-wheel, and a hinged iron bar serves to connect it with the draw-bar of the tub, and prevents

* *Trans. Min. Inst. Scot.*, vol. ix. p. 120.

the clip from turning round and fastening itself under the rollers



FIGS. 478 and 479.—Endless rope clip used at Moston Colliery.

between the rails. It also assists the tub to keep the rails while being attached to the rope, at the time of leaving the level landings.

The clip used is shown in figs. 478, 479. The jaws are made to a

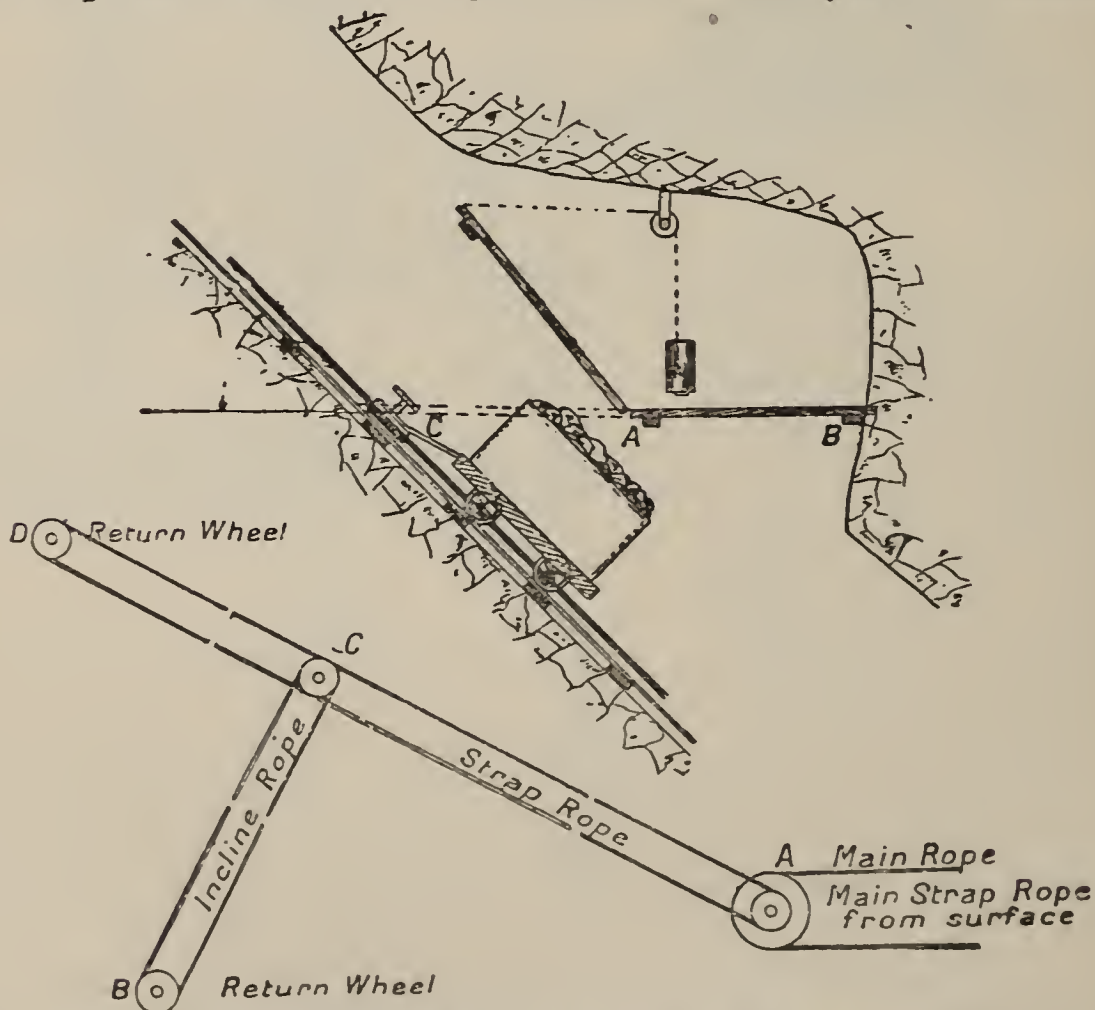


FIG. 480.—Method of working landings at Moston Colliery.

radius of $3\frac{1}{2}$ ft., so that when the screw is tightened the clip has a firm grip of the rope in at least three places, at each end and in the centre. Ordinary clips were found to be of no use in this case,

and hence the adoption of this special type. The full tubs are attached singly, and the empty ones sent down in pairs; both the attaching and detaching being accomplished at all points without stopping or interfering with the speed of the rope.

There are several intermediate landings at which the full tubs are attached and the empty ones detached. Fig. 480 shows the method of working these landings. Sufficient roof is taken down to allow one full or empty tub to stand on the fixed landing A B. There is a movable hinged landing A C, controlled by a balance weight in such a position that when it is lowered the empty tubs will run on to this platform instead of going further down the incline. When the tubs have arrived the clip is unfastened, and they are run on to the fixed landing A B, the movable landing A C being immediately pulled up by the balance weight into the position shown in the figure until it is required again. When a full tub is about to be clipped on to the rope, the movable landing A C is lowered, and the tub brought on to it, and without stopping the rope it is clipped and the tub started up the incline. Two boys are stationed at each of these points to attend

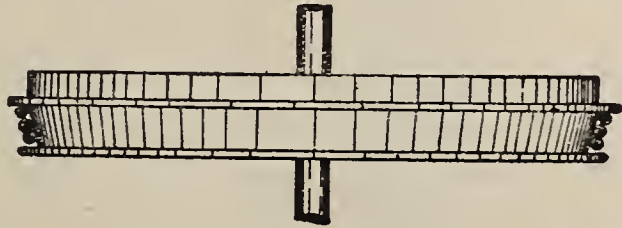


FIG. 481.—Pulley.

to the tubs. The distance between the landings varies from 100 to 140 yds. The power to work this and other ropes is derived from an engine at the surface, being transmitted by means of a band-rope working to a central station underground, the driving engine being compound, with cylinders 15 and 25 in. respectively, and having a stroke of 4 ft. All the pulleys underground are inclined (fig. 481) on the rope surface to the extent of $1\frac{3}{16}$ in. in 6 in., so as to cause the $2\frac{1}{2}$ turns of the rope (which are necessary to give the required grip) to be constantly slipping in the direction of the lesser diameter.

Attached to the pulley working the incline is an ingenious brake arrangement. The brake-rim of the pulley is cast and faced with steel segments. Upon this surface four steel slipper-blocks *aa* (fig. 482) rest, attached to the framework by means of toggle joints *bb*, each pivoted on to a plummer block *cc*.

These slipper-blocks are not set at right angles to the brake-rim, but at such an angle that, upon the wheel continuing to turn in the proper direction (shown by the arrow), the blocks are pushed off the rim; but upon the incline rope being thrown out of gear, the screws *bb* tend to take a position at right angles to the brake-rim, and in so doing wedge the blocks firmly against the rim, and prevent the wheel from turning the opposite way. The rope is thus prevented from a backward movement. To further increase the braking power of the slipper-blocks, they are also attached to the frame by light spiral springs. When the pulley is thrown out

of gear, its first tendency is to revolve in a contrary direction, owing to the weight of tubs and rope on the incline, but this movement is immediately arrested by the automatic blocks coming into action.

The incline for this haulage was laid with a double road throughout, with steel rails weighing 24 lbs. per yard, and fish-plate joints. The amount of coal drawn per shift of nine hours was 300 tons, which is excellent work on a road of this description.

Cost of Haulage.—The cost of haulage is usually stated in pence per ton per mile hauled, and will vary a great deal under different conditions, such as the inclination of the road, the number of branches to be worked, and whether the rope can be kept continually at work during the shift or not.

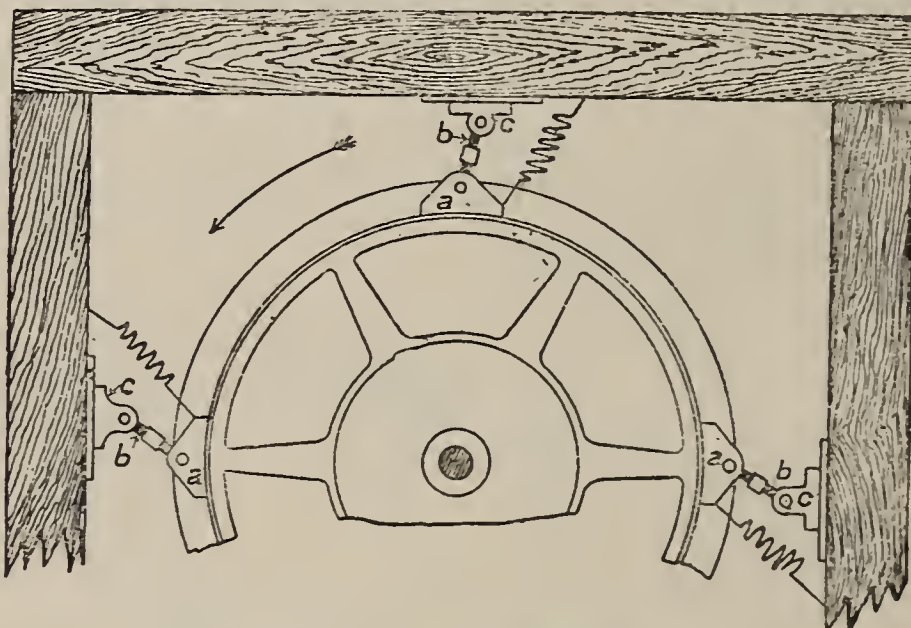


FIG. 482.—Brake arrangement.

The following may be taken as average costs :—

Manual haulage	1s. 6d.	to 3s.	per ton per mile.	
Horse	3d.	to 6d.	„	„
Self-acting incline haulage	2d.	to 3d.	„	„
Single rope haulage	2d.	to 6d.	„	„
Main and tail rope haulage	1½d.	to 2½d.	„	„
Endless rope (clip)	1d.	to 2d.	„	„
„ (bogie)	2d.	to 3d.	„	„
Endless chain haulage	½d.	to 1½d.	„	„

There can be no doubt that, with a well laid-out haulage plant and under suitable conditions, the endless rope system is preferable in most cases. Even in steep roads it will compare favourably with other systems of haulage. In fact, the author is of opinion that the endless rope system is the most suitable for underground work, whether the roads are flat or steep, and can be successfully worked if the roadways are properly laid out for such haulage.

In two instances,* with single rope and endless rope systems, one of the roads dipping 1 in 3 and the other 1 in 4, and each

* *Trans. Inst. Min. Engs.*, vol. x. p. 497.

being about 1200 yds. long, and yielding an output of 800 tons per day, the cost per ton in the former was 5d. and in the latter 1d. per ton. The cost of ropes by the endless system was 0·37d. per ton, which could not be any lower in any other system; for although a double length of rope was required, the difference in cost is always compensated by the reduced wear and tear. There will not, naturally, be the same disparity of cost in all cases, but, as a general rule, endless rope systems will be found to be cheapest.

Advantages of Endless Rope Haulage.—While the endless rope system has the disadvantage of requiring double roads, it has many advantages to compensate for this. Amongst others are—

The small number of attendants required for a given daily output.

The slow speed, which prevents any loss on the journey, reduces to a minimum the risk of breakages, and thereby obviates the mischief consequent on such accidents, such as those which may occur when a train of tubs is travelling at a high rate of speed in the main and tail rope system.

The wear and tear on machinery, tubs, ropes, etc., is a great deal less than with other systems of haulage.

By attaching the tubs at equal distances apart, the weight of the rope is carried along with less friction on the ground and pulleys.

A regular and continuous supply of tubs is brought to the pit-bottom.

In a road with varying degrees of inclination, the whole load is distributed over the whole length of the rope, which is a great advantage. The empty tubs going in-by also assist the engine to haul the full tubs out-by.

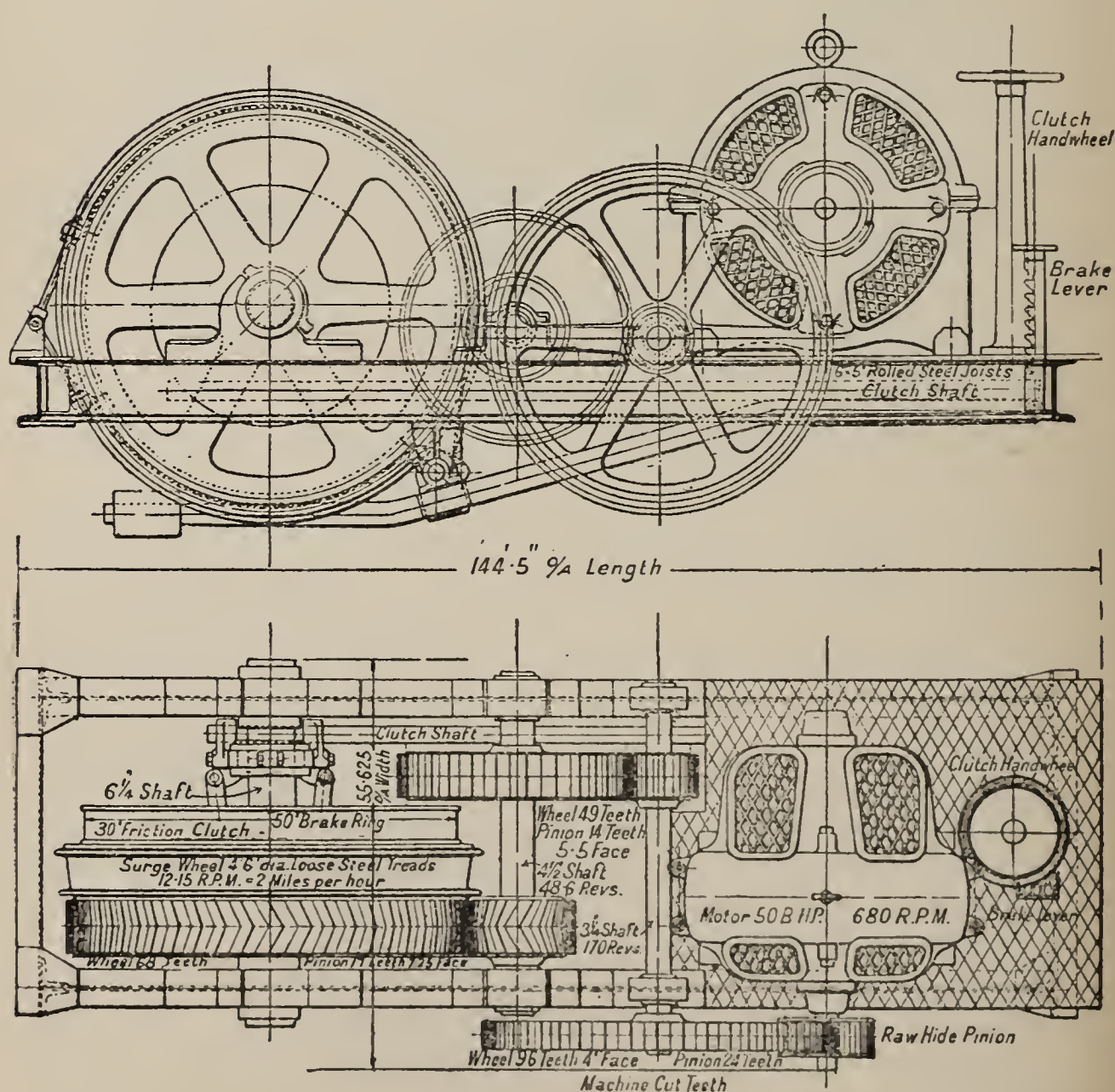
Ropes last longer (on an average three to four years), while with main and tail rope the average is probably not more than one to one and a half years.

Electric Haulage.—Within the past decade the installation of electrically driven haulages in mines has greatly increased, especially for main haulage plant placed underground, and for auxiliary haulages which have to be installed considerable distances from the shaft. Steam or compressed air driven haulages placed underground are being gradually, in some districts rapidly, superseded by electrically driven plant. Conveying steam or compressed air down deep shafts, or along underground roadways, is at the best inefficient and uneconomical, and while electric plant may be more expensive to install, it is more efficient and economical in working. There can be no doubt, too, that electricity is better suited than steam or air for dealing with varying loads, such as in underground haulage. Thus for long distances the power can be transmitted more efficiently and cheaply by electricity than steam or air, which require an expensive system of piping.

Haulage plants are driven either by belt, rope, or spur-wheel gearing, the latter being the most common, especially for the endless rope system. Figs. 483, 484 * show an electric endless rope plant of 50 brake horse power, and of the following dimensions :—

* *The Iron and Coal Trades Review*, 7th October 1910. See also *Proceedings of the National Association of Colliery Managers*, vol. viii. p. 284.

Length over all, 12 ft. 0½ in. ; width, 4 ft. 7½ in. ; height, 5 ft. 7 in. over all ; height from top of bed, 4 ft. 4 in. ; frame or bed, 8 in. × 5 in., rolled steel joints, well plated and angled ; motor (either continuous or alternating current), 50 B.H.P., 680 revolutions per minute ; raw-hide pinion on motor shaft working into cast-iron wheel with machine-cut teeth, both 1½ in. pitch, 4½ in. face, at the ratio of 4 to 1 ; second motion shaft, 3¼ in. diameter, running at 170 revolutions per minute ; second reduction gear wheels (plain teeth, shrouded and turned to pitch line), 1⅞ in. pitch, 5½ in. face, at the ratio of 3½ to 1 ; third motion shaft, 4½ in. diameter, running at 48·6 revolutions per minute ; third reduction gear wheels (double helical teeth, shrouded, and turned to pitch line), 2½ in. pitch, 7¼ in. face, at a ratio of 4 to 1 ; fourth motion (driving wheel) shaft, 6¼ in. diameter, running at 12·15 revolutions per minute. The surge wheel for haulage rope is 4 ft. 6 in. diameter on tread by 4½ in. wide, with renewable cast-steel treads ; friction clutch, 2 ft. 6 in. diameter, actuated by hand-wheel ; brake ring (turned), 4 ft. 2 in. diameter, of the all-round type, worked by foot-lever ; clutch hand-wheel and brake foot-lever placed close together.



FIGS. 483 and 484.—50 H.P. endless haulage plant.

Haulage Problems.

Question.—A train of ten tubs is ascending an incline of $4\frac{1}{2}$ in. rise per yard, each tub weighing with coal one ton. What power would be required, and what would the strain on the rope be?

$$4\frac{1}{2} \text{ in. per yd.} = \frac{3 \times 12}{4 \cdot 5} = 1 \text{ in. in 8.}$$

Weight of train = 22400 lbs.

$$\text{Friction taken at } \frac{1}{70} \text{ of the load} = \frac{22400 \times 1}{70} = 320 \text{ lbs.}$$

$$\text{Then } \frac{22400 \times 1}{8} = 2800, \text{ the force required estimated in lbs., and}$$

$$2800 + 320 = 3120 \text{ lbs.} = \text{the total strain on rope.}$$

Question.—Find the tension on a haulage rope with a load of 20 tons on an incline of 1 in 6.

Let W = load in lbs., H = vertical factor of rise, and L = horizontal factor of rise.

$$\text{Then tension} = \frac{W \times H}{L};$$

$$\text{Tension} = \frac{20 \times 2240 \times 1}{6} = 7466 \cdot 6 \text{ lbs., due to gravity alone,}$$

or allowing $\frac{1}{70}$ for friction,

$$\text{Total tension} = \left(\frac{20 \times 2240 \times 1}{70} \right) + 7466 \cdot 6 = 8106 \cdot 6 \text{ lbs.}$$

Question.—What engine power, expressed in foot-pounds per minute, is required to draw a load of eight tons up an incline of 1 in 5, at a speed of five miles per hour, excluding friction?

$$\text{Speed of rope} = \frac{5 \times 5280}{60} = 440 \text{ feet per minute.}$$

$$\text{Load} = \frac{8 \times 2240 \times 1}{5} = 3584 \text{ lbs.}$$

Engine power required (load in lbs. $\times$ speed in feet per minute) = $3584 \times 440 = 1,576,960$ ft. lbs. per minute.

Question.—What size of engine, length of stroke, etc., would be required to haul 400 tons per shift of nine hours, from a road 1000 yards long and dipping 1 in 12? The system of haulage to be main and tail rope, and the speed of rope six miles per hour, the diameter of the drum being 6 ft.

$$\text{Speed of rope per minute} = \frac{6 \times 5280}{60} = 528 \text{ ft.}$$

$$\text{Time per trip} = \frac{1000 \times 2 \times 3}{528} = 11 \text{ minutes.}$$

Time allowance for changing at each end, about 4 minutes.

Total time per trip, $11 + 4 = 15$ minutes.

$$\text{Load per trip} = \frac{400 \times 15}{9 \times 60} = 11 \text{ tons (approximately).}$$

Suppose the tubs hold 10 cwts. each, and to weigh 4 cwts. each when empty, twenty-two tubs will be required each trip, and the weight of the tubs will be $22 \times 4 = 88$ cwts. or 4.4 tons; therefore the gross load per trip will be $11 + 4.4$, or 15.4 tons.

$$\text{The load due to inclination} = \frac{15.4 \times 1}{12} = 1.28 \text{ tons.}$$

$$\text{The circumference, } C, \text{ of the rope required would be} = \sqrt{\frac{1.28 \times 8}{3}} = 1.84 \text{ in.}$$

$$\text{Weight of rope per fathom} = C^2 \times .9 = 3.06 \text{ lbs.}$$

$$\text{Total weight of rope} = 500 \times 2 \times 3.06 = 3060 \text{ lbs.}$$

Taking friction of load equal to 32 lbs. per ton, and friction of rope equal to $\frac{1}{20}$ th its weight; then

$$\text{Friction of full load} = 15.4 \times 32 = 492.8 \text{ lbs.}$$

$$\text{And friction of rope} = \frac{3060}{20} = 153.0 \text{ lbs.}$$

$$\text{Total friction is } 492.8 + 153 = 645.8 \text{ lbs.}$$

$$\text{Total load} = \text{full load} + \text{rope} + \text{friction} = 38201.8 \text{ lbs.}$$

$$\text{Total resistance to engine} = \frac{(15.4 \times 2240) + 3060}{12} + 645.8 = 3520.4 \text{ lbs.}$$

Now the work done in the engine is equal to the work done on the plane, and can be expressed by the formula :

$$D^2 \times .7854 \times P \times L \times N \times M = \text{load in lbs.} \times \text{circumference of drum in feet.}$$

Assume the effective steam pressure to be 45 lbs. per sq. in., and the modulus M , $\frac{2}{3}$ for single engine and $\frac{4}{3}$ for coupled engines, the length of stroke being $3\frac{1}{2}$ ft.

$$\therefore D^2 \times .7854 \times 45 \times 3.5 \times 2 \times \frac{4}{3} = 3520.4 \times 6 \times 3.1416$$

$$\text{After cancelling } D^2 \times 15 \times 3.5 \times .8 = 3520.4 \times 4$$

$$\text{or } D^2 \times 3.5 = \frac{3520.4}{15 \times .2} = 1061.3$$

$$\therefore D = \sqrt{\frac{1061.3}{3.5}} = 17.3 \text{ in.,}$$

the size of engines required.

This problem may also be worked out by the formula :

$$D^2 \times .7854 \times P \times L \times N \times M = (W \times h) + (\text{friction} \times \text{cir. of drum in ft.}),$$

$$\text{or } D = \sqrt{\frac{(W \times h) + (F \times \text{cir. of drum})}{.7854 \times P \times L \times N \times M}}$$

where as before,

D = diameter of cylinder in in.

L = length of stroke in ft.

M = efficiency of engine.

P = effective steam pressure in lbs. per sq. in.

N = number of strokes per revolution of the drum.

W = weight of loaded train in lbs.

F = friction of loaded train in lbs. and rope.

h = vertical height which the train is raised during one revolution of the drum.

$$\text{As } h = \frac{6 \times 3.1416 \times 1}{12} = 1.57 \text{ ft.}$$

Taking the same figures as before, we have,

$$D = \sqrt{\frac{(15.4 \times 2240 \times 1.57) + (645.8 \times 6 \times 3.1416)}{.7854 \times 45 \times 3.5 \times 2 \times \frac{4}{5}}} = \sqrt{334.98} = 18.3 \text{ in.}$$

Question.—Find diameter of cylinder required, length of stroke, etc., to haul 300 tons per eight-hour shift from a road dipping 1 in 20, laid with tram rails. Length of road 600 fms. Tubs to weigh 4 cwts. and hold 10 cwts. Size of driving wheel 6 ft. diameter. System, endless rope with a speed of two miles per hour.

Tons per hour = $\frac{300}{8} = 37.5$: speed of rope 3520 yds. per hour = 176 ft. per minute.

Tubs per hour = $37.5 \times 2 = 75$: distance tubs will have to be apart on rope = $\frac{3520}{75} = 46.9$ or 47 yds. (for simplicity).

Number of tubs on rope at once = $\frac{600 \times 2}{47} = 25.55$, approximately 26, *i.e.* 26 full and 26 empty.

Weight of loaded tubs, $= 26 \times 14 \times 112 = 40768 \text{ lbs.}$

“ empty “ $= 26 \times 4 \times 112 = 11648 \text{ „}$

Weight of rope, say at 4 lbs. per fathom = $600 \times 2 \times 4 = 4800 \text{ „}$

Total = 57216 „

Suppose the total friction is $\frac{1}{30}$ th $\therefore \frac{57216 \times 1}{30} = 1907.2 \text{ lbs.}$

Load due to inclination = $\frac{\text{full load}}{\text{inclination}} = \frac{40768 \times 1}{20} = 2038.4 \text{ lbs.}$

Total load to be overcome by engine = $2038.4 + 1907.2 = 3945.6 \text{ lbs.}$
And as before, work done by engine = work done on plane. Assume effective steam pressure at 50 lbs. per sq. in., and the piston speed = 250 ft. per minute.

$$\therefore D^2 \times .7854 \times 50 \times 250 \times \frac{2}{3} = 3945.6 \times 176,$$

$$\text{and } D^2 = \frac{3945.6 \times 176}{.7854 \times 50 \times 250 \times \frac{2}{3}} = 107.17.$$

$$\therefore D = \sqrt{107.17} = 10.35 \text{ say } 10\frac{1}{2} \text{ in.}$$

Assuming length of stroke = 3 ft., then the gearing would be found as follows:—

$$\text{Revs. of wheel or drum} = \frac{176}{6 \times 3.1416} = 9.34 \text{ per minute.}$$

$$\text{Revs. of crank} = \frac{250}{3 \times 2} = 41.66 \text{ per minute.}$$

$\therefore$ Ratio of gearing would be

$$\begin{array}{l} 9.34 : 41.66 \\ \text{or } 1 : 4.42. \end{array}$$

The size of engines required would be therefore 10.5 in. diameter, with a 3 ft. stroke geared 4.42 to 1.

This problem may also be worked out by the formula

$$D = \sqrt{\frac{(F \times \text{cir. of drum}) + (W - w)h}{.7854 \times P \times L \times N \times M}},$$

the letters having the same value as in preceding question, and in this case w = weight of empty tubs.

$$\text{Here } h = 20 \text{ ft.} : 18.84 \text{ ft.} :: 1$$

$$= \frac{18.84 \times 1}{20} = 94 \text{ ft.}$$

$$\begin{aligned} \therefore D &= \sqrt{\frac{\{1907.2 \times 6 \times 3.1416\} + \{57216 - (11648 + 4800)\} \cdot 94}{.7854 \times 50 \times 3 \times 4.42 \times 2 \times \frac{2}{3}}} \\ &= \sqrt{\frac{74272.64}{687.35}} = 10.38 \text{ in.} \end{aligned}$$

Coal Conveyors.—In seams which are too thin to allow of the tubs being taken along the face to be filled, a great deal of extra labour must be done to get the coal conveyed to the road-head or gate-end where the tram road terminates. The labour of doing so becomes greater and more difficult as the seams get thinner, and with the increase in the length of wall necessary to stow the débris got in the ripping. In a seam 2 ft. thick, and with walls 15 yds. in length, the coal may require to be cast—from the extreme end—three or four times before it reaches the road-head where it is filled into the tubs. In addition to the labour entailed in backing out the coal, the continual throwing of it causes much breakage and, therefore, increases very materially the production of small coal. It was to get over these difficulties that coal conveyors were introduced at the working face, and also to reduce the number of drawing roads required for a given length of face. When conveyors were first introduced the idea was to work a length of 100 to 300 or 400 yds., with a main roadway at each end where the tubs would be filled and afterwards conveyed to the mechanical haulage for transport to the winding shaft, thus doing away with the ripping and upkeep of large numbers of drawing roads. The installing of conveyors has made steady progress during the last few years, although not a few have been disappointed with the results, and have consequently discarded them. Where satisfactory results have been got it is generally due to a proper knowledge of the conditions necessary to ensure success. The causes of failure may be due to a number of reasons such as: (1) the workings where the conveyors have been installed were unsuitable for such mechanical apparatus; (2) a wrong method of timbering was adopted to keep the roof from falling on and burying the conveyor; (3) the type of conveyor selected has been unsuitable for the prevailing conditions. Another error that has been frequently made is to introduce conveyors into seams that are being under-cut by coal-cutters, the impression being that under such circumstances the best results will be obtained both as to labour

and economy. In a great many cases no greater mistake could be made, for it is only under certain conditions—which are not very usual in underground work—that a combination of coal-cutters and conveyors will yield good results. We are strongly of opinion that in a very large proportion of cases the best results are obtained from conveyors where they are used in seams that are cut by hand labour, and believe that there are many instances where coal-cutting machines are used without conveyors, and the holing is not too hard, if conveyors had been adopted instead of coal-cutters the saving would have been greater. Of course, there are exceptions to every rule, and we find instances where conveyors are working in conjunction with coal-cutting machines and giving highly satisfactory results. But to be successful with such a combination there must necessarily be a fairly good roof, as the width of roof which requires to be supported on timber—between the line of the coal-face and the packwalls—to accommodate the two sets of apparatus, coal-cutters and conveyors, will require to be 8 to 10 ft. at the least. That is a width that would be almost impossible to maintain with many roofs, especially where seams lie at a considerable depth from the surface. We do not say that it cannot be done, but the extra cost for timber required to support the roof properly would in many cases more than counterbalance the saving obtained by one of the installations, *i.e.* the conveyor or coal-cutter. Another mistake which has been made, but which is now being largely rectified, was to carry out the practice of having only two roadways, one at either end, of a long length of face. If this system could be carried out, it would certainly mean a very large saving in costs for ripping and upkeep of roads, but here again it would be impossible to do so in very many cases, and if attempted could only end in failure. Having only two roadways, it means that a length of 200 to 400 yds. of roof would only be supported at each end by a building or packwall 10 or 12 ft. in length. This would be a great strain on the intervening space of roof, which would require to be supported by timber alone. Of course, extra support might be got by taking into the face any débris which would be got from falls of roof in the roadways, but this would mean extra labour and expense. Additional support might also be obtained by building in timber pillars or chocks, but the extra cost of doing so might greatly reduce, or minimise altogether, any saving made by the decrease in the ripping and upkeep of additional roadways. There is also the further fact that with a large area of roof supported on timber alone there is always a feeling of insecurity among the workmen, for with only an open roadway at each end of the face, their egress might be cut off if the intervening roof was to collapse suddenly. By adopting such a system it is very much like supporting a very long beam at each end and loading it in the centre. This might be done with safety on the surface, where all the forces acting could be calculated, and the beam made of

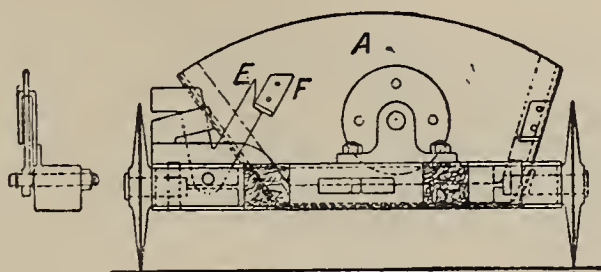
sufficient strength, but underground it is impossible, since it is exceedingly difficult to know all the forces that require to be contended with. To overcome the difficulties mentioned, and to ensure sufficient support to the roof, the practice in conveyor walls, at least in Scotland, is to have roadways at the regulation distance of 12 to 18 yds. apart. The roadways are ripped in the usual way, and pack-walls built in the face with the débris, the roads being left unsupported and abandoned as they are not required for drawing roads. There is thus a considerable saving in cost effected, as rails and sleepers do not require to be laid down, timber supports do not require to be set—unless in any of the roadways which may be required as an airway or outlet—and the cost of upkeep and repairs is done away. One or two other points may be noted if conveyors are to be successfully worked at the face: the labour necessary must be thoroughly and efficiently organised; the mechanical haulage must be kept close up to the point where the coal is delivered by the conveyor; and during the working shift there must be a continuous and sufficient supply of tubs to take away the coal.

Types of Conveyors.—The idea of transporting coal along the face by conveyors was, no doubt, evolved from the practice in use for many years in thin seam collieries in different parts of England and Scotland, of conveying the coal from the face to the road-head or gate-end by means of small bogies or slypes pushed or drawn by boys.

Mickley Conveyor.—This was one of the first modifications and adaptations of the bogie system, and is worked by manual labour. The conveyor is a long, shallow tub, the box being made of sheet-iron, 7 ft. long at the top and tapering down to $6\frac{1}{2}$ ft. at the bottom, 2 ft. 8 in. wide, 11 in. deep on the side next the goaf, and 7 in. deep on the side next the coal-face, this difference in height facilitating the filling of the coal into the box. The tub is fitted with wheels at each end, 6 in. diameter, which run on rails with a gauge of 2 ft. 10 in. The bottom of the conveyor is fitted with two sliding doors, which are drawn out when it is to be emptied. The conveyor is moved backwards and forwards along the face by means of an endless wire rope, fastened to each end of the tub, and passing round a wheel, 10 in. diameter, fixed at the far end of the face, and also round another, the driving wheel, 18 in. diameter, fixed at the road-head. To one of the spokes of this latter is fixed a handle about 12 in. long, which is turned by a lad when the conveyor requires to be hauled either way along the face, each conveyor working on a length of 150 ft. Each conveyor can deal with about 40 tons per shift in a seam varying from 19 to 28 in. in thickness, and was said to reduce the cost about 8d. per ton. This conveyor can only deal with a very limited quantity of coal, and has, therefore, not been largely employed.

Bothwell Conveyor.—This is probably the best and most efficient

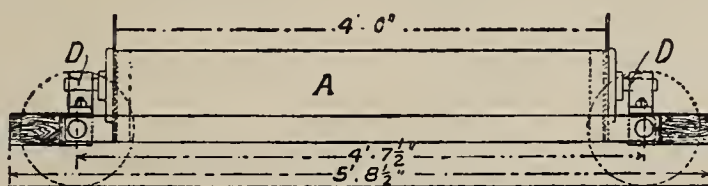
conveyor of the bogie type. It consists of a number—15 to 30 according to requirements—of small bogies, the body (A A, figs. 485–487) of the bogie being constructed of light sheet-iron set on an angle-iron frame. Each bogie holds $3\frac{1}{2}$ to $3\frac{3}{4}$ cwts. of coal and



End Elevation.

FIG. 485.

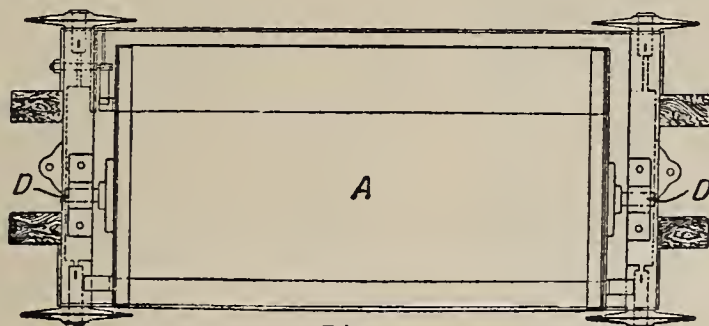
runs on four-edge wheels. The body of each bogie is set on a pair of trunnions (D D, figs. 486, 487), which are placed off the centre of gravity. At one end is a 'kicker' (E, fig. 485) working on a centre bolt, which fits into an angle piece F, fixed on to the end



Side Elevation.

FIG. 486.

of the bogie. This 'kicker' works automatically and holds the bogie in position while it is being moved along the face, and while it is being filled. The body of the bogie is so constructed that when it is empty the heavier portion is to one side, while the excess weight

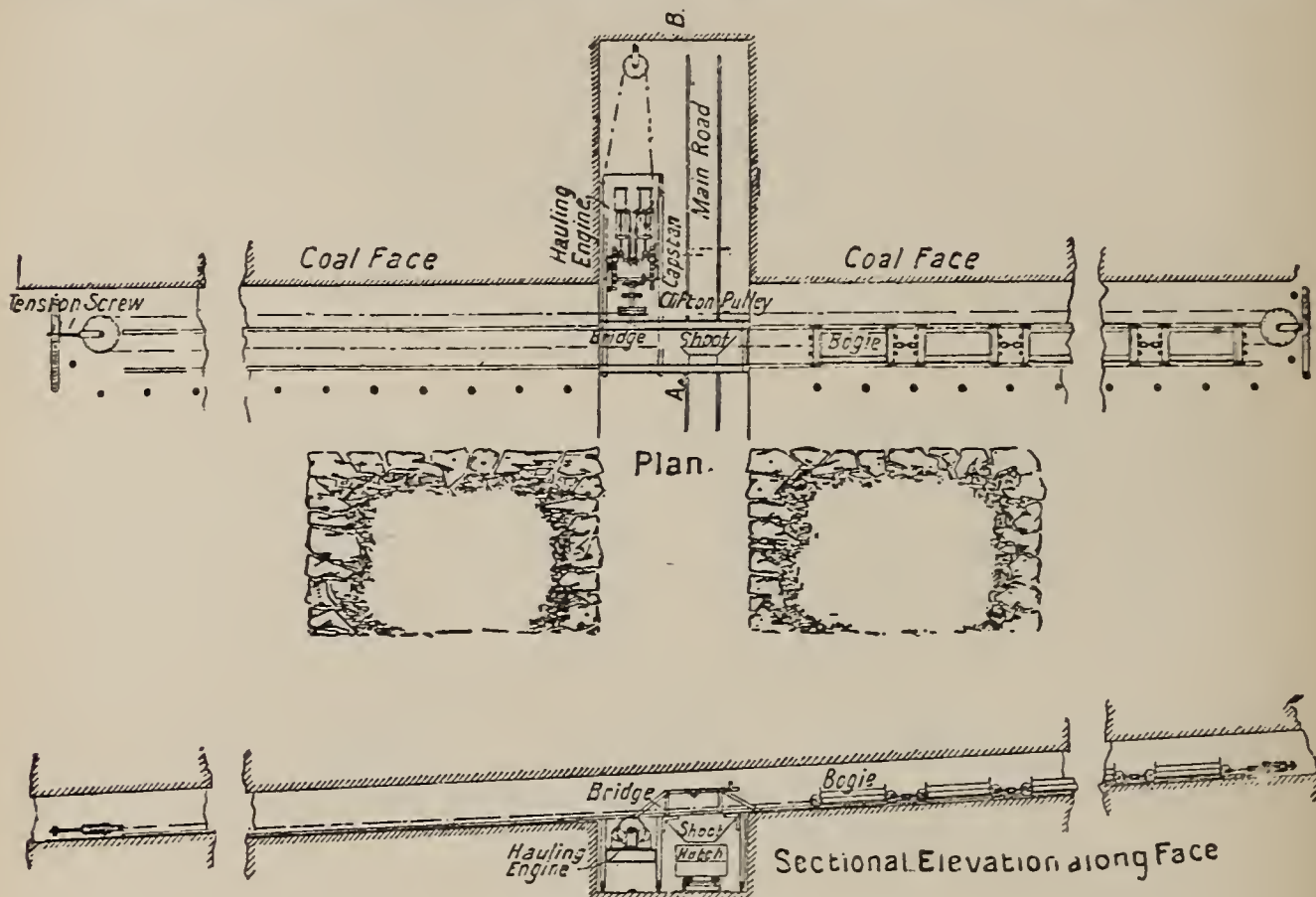


Plan.

FIG. 487.

is shifted to the opposite side when loaded, so that when the coal is emptied at the gate-end into the tubs the bogie assumes the upright position automatically, and the apparatus, therefore, works on the self-acting principle. The general arrangement of the working face, conveyor, hauling engine, etc., will be seen from figs. 488, 489.

A centre road (B, fig. 488), $8\frac{1}{2}$ ft. wide, is kept about 40 ft. in advance of the line of the working face. The total length of face on which the conveyor works may be 150 to 250 yds. At Bothwell Park Colliery near Glasgow, where this conveyor was first



FIGS. 488 and 489.

introduced, the conveyor was operated by a compressed air hauling engine with cylinders 6 in. diameter by 12 in. stroke, geared 5 to 1, and working a Clifton pulley 18 in. diameter, which actuates the hauling rope of plough steel, $\frac{1}{2}$ -in. diameter. The Clifton pulley

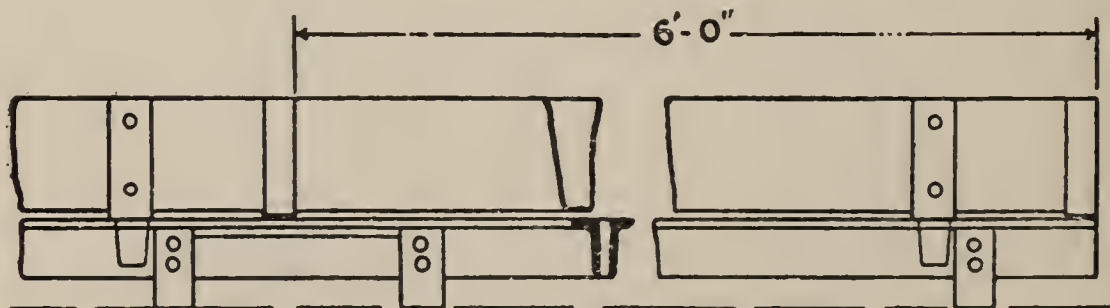


FIG. 490.—Troughs and frames.

is set so that the hauling rope is directly in line with the face. The rope after leaving the driving pulley extends along the face at each side of the advance centre road, and then passes round a pulley 18 in. diameter, fitted with a snatch-block frame with the bolt or holding end of frame passing through a 10 in. $\times$ 5 in. girder, of H-section, 10 ft. long, and in front of which is set two props. At

the end of each length of face one of the pulley frames is fixed to a tension screw which passes through the girder. The hauling of the bogies can be done on either the endless rope, main and tail rope, or single rope systems according to the conditions prevailing. The hauling engine and bridge are made of channel irons all combined in one, and is sitting on wheels so that these parts of the installation can be shifted when required in 5 to 10 minutes. When the engine requires to be shifted a snatch-block pulley is fixed to a prop at the advance road-face. On the engine is a small capstan on which is coiled a length of rope. The loose end of this rope is drawn out, passed round the fixed snatch-block pulley, and then brought back and fixed to the end of the movable platform on which the engine is carried. By setting the engine in motion, the bridge and engine are then drawn into the desired new position. The engine and bridge are usually shifted every second shift. The conveyor is made in three sizes, to suit varying thicknesses of seams, 17 in., 12 in., and 9 in. high, the latter being suitable for a seam 18 in. in thickness. It can deal with an output of 25 to 28 tons per hour.

The 'Blackett' Conveyor. — The 'Blackett' was the first successful coal-face conveyor to be put on the market, and there are now over two hundred and fifty in operation.

This conveyor consists of standardised 6-ft. sections of machine-pressed steel troughing carried on similar lengths of angle-iron framework. Each length can be erected or dismantled without bolt or nut, and each trough can be similarly lifted out for access to the chain beneath. The chain is constructed to carry the coal along the troughs. The lower half of the chain returns upon suitable runners, and a ratchet screw is provided at the tail-end for tightening or

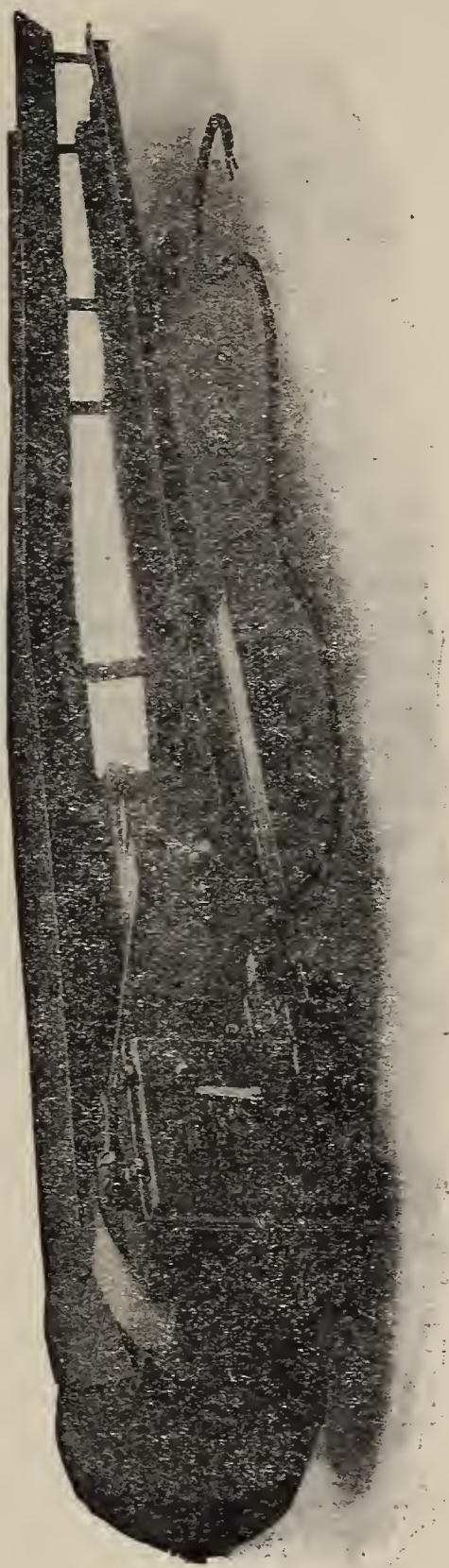
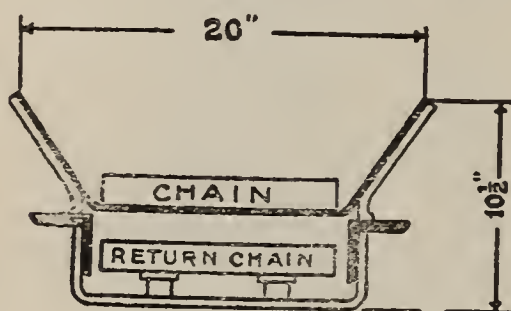


FIG. 491

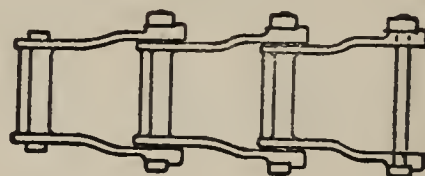
loosening the chain. The weight of the discharging end, comprising the motor-engine, dispenses with the need of any special mooring device, but at the tail-end of the conveyor a screw-tightening arrangement is provided with a mooring chain for securing to timber.

Electrically driven conveyors are fitted with totally enclosed motors of 8-10 B.H.P. The compressed air conveyor is driven by a small two-cylinder horizontal high-speed engine totally enclosed, and having self-oiling bearings and splash lubrication. Both the direct current motor and the compressed air engine are tucked away underneath the discharging-end trough. In the case of alternating current the motor stands on an extension of the framework of the discharging end, and in all cases the electric motor or air engine is specially designed and forms part of and moves with the conveyor.

The height of the conveyor over the troughs is about 10 in. The discharging and tail-ends are a little higher, but will easily go under 2 ft. The overall width is 20 in. along the face.



End view.



Chain.

FIGS. 492 and 493.

To prevent breakages in the event of anything getting fast a copper shearing key is inserted between two driving discs outside the gear case which can be readily replaced.

The Blakett conveyor can be worked to the best advantage in seams of not less thickness than 2 ft. with good roof and floor, the latter being free from pronounced undulations. The usual length is from 60-120 yds. A main gateway with double road is required at the discharging end of the conveyor, and this should be kept driven a few yards in advance of the coal-face to allow of a continuous supply of empty tubs. When the entire length of face has been cut by a Longwall machine, fillers are sent in to load the coal into the trough along which it is conveyed by the chain direct into the tub at the main or mother-gate. The conveyor is advanced with the face, being moved forward either in sections without disturbing the timber or (where the roof allows of several props being drawn) slued forward from either end, having sufficient flexibility to allow of this being done.

The 'Gibb' Conveyor.—This conveyor consists of a long, low, jointed carriage provided with mechanical end-discharge operated by the same motive power which draws the carriage to and fro along

the face. The dimensions of the carriage are adapted to the height of the seam for which it is required. Each carriage comprises a discharging end, as many intermediate sections as may be required, and a back end, containing the chain-returning gear. The carriage has a maximum height from the pavement of 20 in., and an inside width of 24 in. It consists of the two end sections and five intermediate sections, each 6 ft. long. The end-discharge is effected by scrapers attached to a pair of endless chains, which are returned under the bottom of the carriage. The discharging end contains the gear through which the scraper chains are operated by a rope, from an independently situated haulage gear, engaging with a rope wheel mounted on the side of the carriage. While the carriage is travelling the rotation of this wheel is prevented by a ratchet, but on arriving at the discharging position a simple trip arrangement releases the wheel, which is then free to rotate, and drive through spur and pinion the sprocket wheels which set the scraper chains in motion.

The intermediate sections are built of steel throughout, and are coupled to each other by a method

which permits the necessary freedom of movement. Fenders are fitted on the outsides, and these shroud the four flat tread wheels on which each section is carried, and guide the whole carriage between the props. The height of the goaf side of the carriage must leave sufficient clearance to avoid contact with the roof while the carriage is travelling, and the height of the face side must be such as to leave space between it and the roof for filling lumps of coal when the carriage is at rest at the filling position. The back end is built similarly to the discharging end, but is only fitted with the sprocket wheels for returning the scraper chains, and a shaft which carries at one end a rope wheel, and at the other end a ratchet. The sections of the carriage being loosely jointed, the carriage as a whole cannot be pushed, but must be pulled from the respective ends. The rope is therefore passed

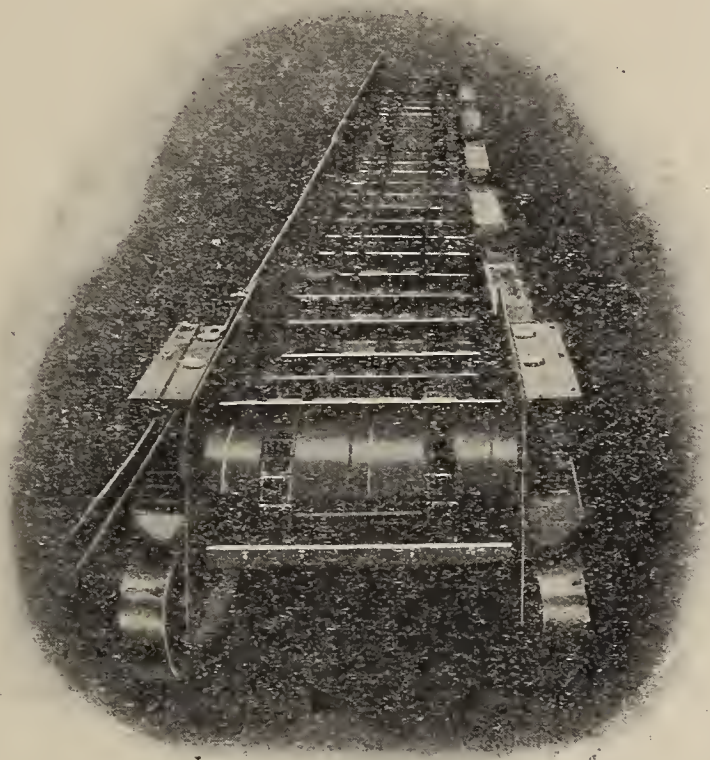


FIG. 494.

round the wheel referred to above, and this wheel is free to rotate idly while the rope is operating the discharge, but when the direction of the rope is reversed for returning the carriage, the ratchet fixes the wheel, and the whole carriage is drawn along the face to the desired position. The capacity of the carriage depends upon the thickness of the seam. In a seam with total headroom of 18 in., which may be considered the minimum, the capacity of a carriage with internal width of 20 in. would be about $\frac{1}{2}$ cwt. per foot run, and a carriage consisting of six intermediate sections would have a capacity of 22 cwts. In seams of 2 ft. 6 in. or greater thickness the increased permissible height of carriage raises its capacity to 50 cwts. or more. The capacity of a carriage can be readily increased by attaching plates to the sides and so increasing its depth.

Filling.—The number of fillers that can be simultaneously engaged in filling depends upon the length of carriage. With a carriage consisting of seven intermediate sections, eight men, including one in front and one behind, may be employed. During the absence of the carriage at the haulage road the coal is broken up and prepared, so that the actual operation of filling is quickly effected. On receiving the signal that the carriage is full the man at the roadway sets the conveyor haulage in motion, and the carriage is drawn along the face towards the road at a speed of about 150 ft. per minute. The props being set in a straight line prevent the carriage, which is provided throughout its length with rubbing strakes or fenders, from diverging from the forward direction. When the carriage reaches the 'stops' at the roadway, with its end overhanging an empty tram, it automatically operates a trigger which disengages the discharging gear. The discharge commences instantly on the arrival of the carriage at the 'stops,' and the endless rope which hauled the carriage continues its motion, operating the discharging gear through the rope pulley, which, being disengaged by the trigger, is free to turn. The controller of the haulage motor is placed close to the discharging position, so that the rate of discharge may be regulated, and reduced or stopped at the moment of changing trams. When the load has been discharged the direction of travel of the rope is reversed by operating the controller, the trigger re-engages and locks the discharging gear, and the carriage is returned to the fillers, who during its absence have been breaking down and preparing the coal for filling, and where dirt is present casting this into the goaf.

These four operations complete the cycle, which is repeated until the face is cleared.

The moving of the carriage to each new position as the face advances is a very simple operation. The sections are not disconnected as in the case of full-length conveyors, but the turn pulley of the conveyor haulage rope is moved laterally towards the face, and the timbers are so placed as to permit of the carriage being

pulled along the face in a slanting direction until it is guided into its new working position. The carriage can be shifted by the method described in about twice its own length. The complete operation, including setting the props, readjusting the haulage pulleys and rope, occupies four men for two to three hours. The endless rope through which power is transmitted to the carriage in all its operations is driven by a gear placed in an inset from the haulage road a few yards to the rear of the machine face as shown in fig. 495. This gear is moved forward in longer stages than the carriage and at correspondingly longer intervals of time. The tension pulley is fixed in the advance heading, 15 or 20 yds. in advance of the face, and it remains in this position until the forward movement of the face has overtaken it. The haulage gear is then moved forward to its original position relatively to the face, and the tension pulley is again advanced into the heading. The turn pulleys in the haulage road and at the end of the face are moved each time the carriage is shifted.

The haulage may be operated either by electric motor, or by compressed air engine, and either the air engine or the electric motor is controlled from the discharging position—in the former case by an auxiliary valve, and in the latter by a tramway type controller, fitted with self-contained resistance. A platform, consisting of two suitably connected flanged plates on to which the leading wheels of the carriage run against 'stops,' is provided for the discharging position. The rope is flexible, of plough steel similar to coal-cutter haulage ropes.

The 'Ritchie' Conveyor.—The 'Ritchie' conveyor is applicable to seams about 2 ft. thick and under, where a limited output is obtained. It is of the intermittent discharge type and will deal with any quantity up to 80-90 tons per shift during normal working. It can be successfully applied under conditions such as bad roof or undulating floor.

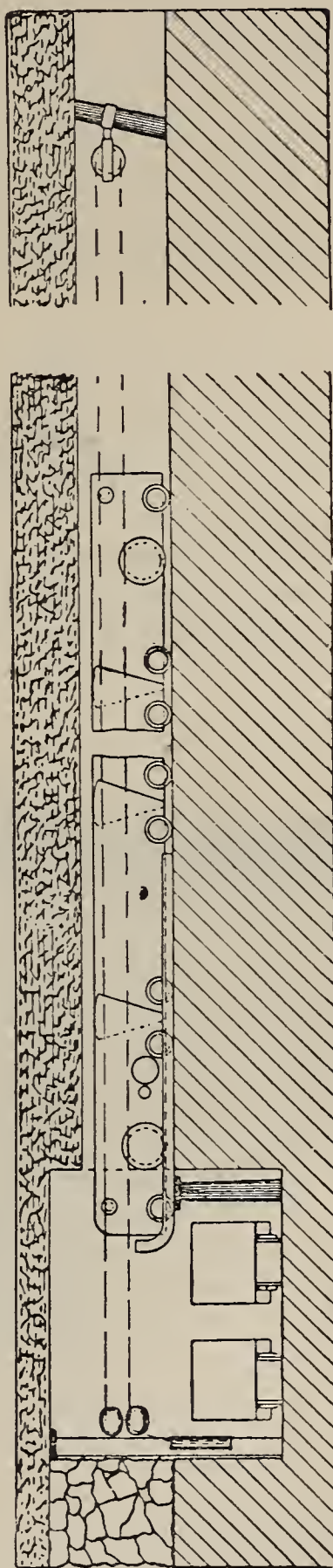
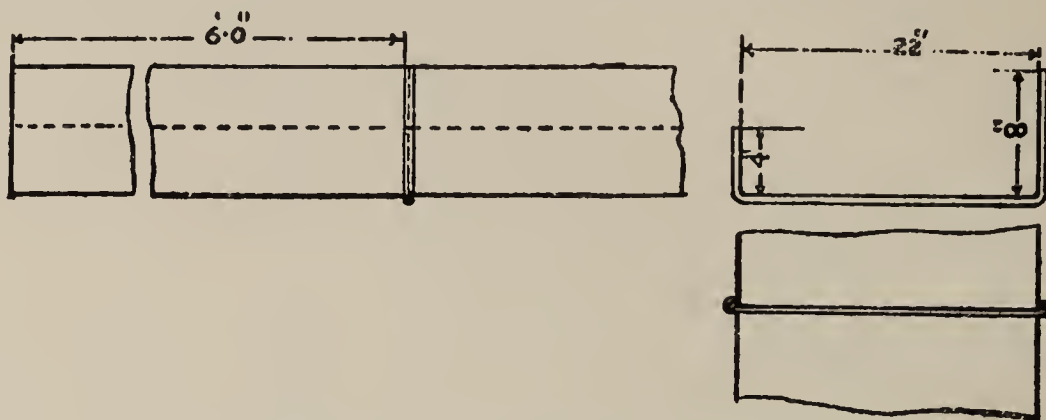


FIG. 495.—Sectional elevation through haulage road showing carriage at discharging position.

The conveyor is composed of standardised 6-ft. sections of light machine-pressed steel troughing. A belt of special construction, about one-third the length of the face, is hauled to and fro in the troughing by means of main and tail ropes driven by an electric



FIGS. 496 and 497.—Section of troughs.

motor or compressed air engine contained in the delivery end of the conveyor. The discharging end stands on the rails in the main gateway, is entirely self-contained, and is firmly secured in position by

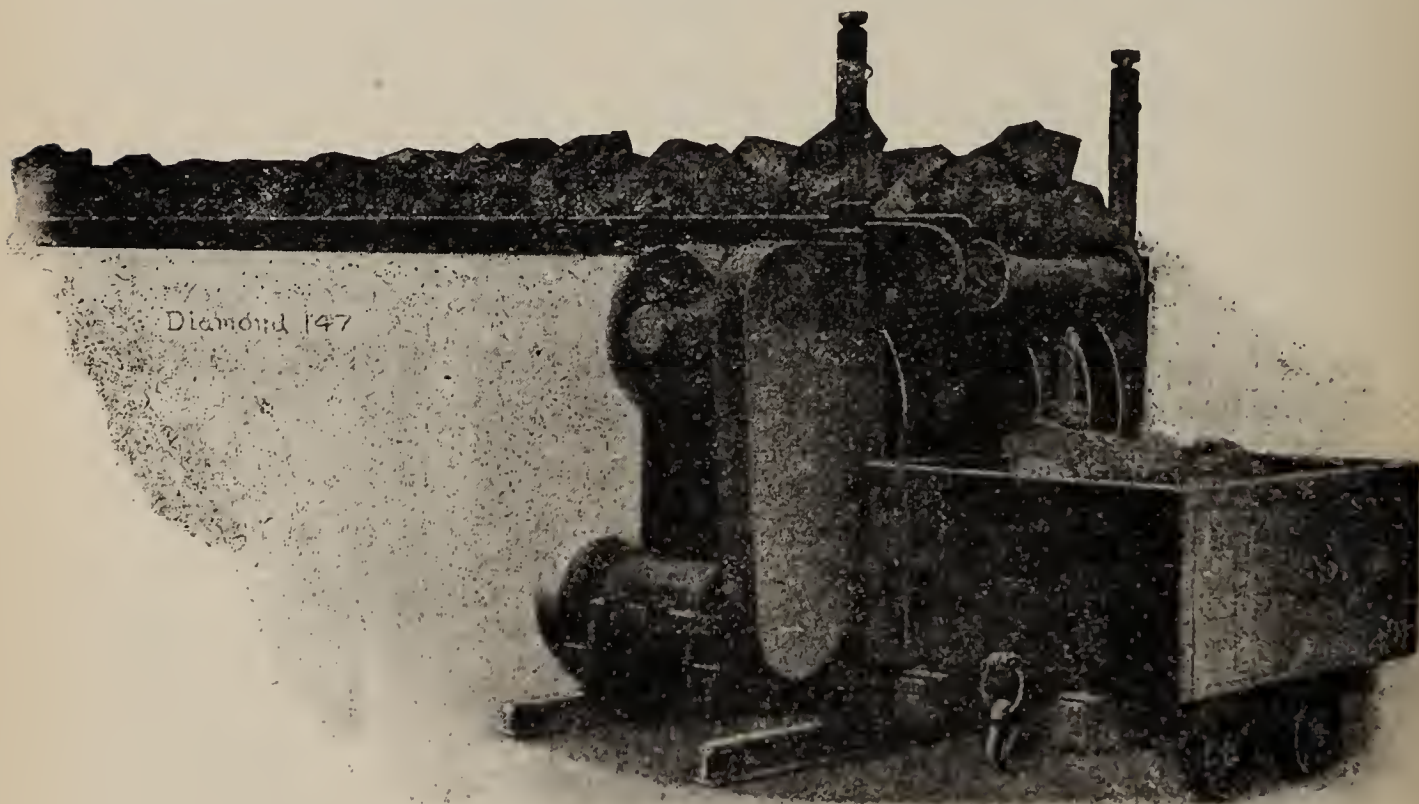


FIG. 498.—Discharging end of 'Ritchie' conveyor.

means of two short screw-jacks against the roof. All the gearing in the delivery end is completely enclosed. The tail rope is carried round a return pulley which is merely anchored to a prop in line with the troughs. The height of the troughs at the face side is only 4 in., and on the gob side 8 in. The overall width is 22 in. The

coal is delivered into the tubs as the belt is being wound on to the drum. The return speed of the belt is about three times the loaded speed, and the belt can, on its return journey, be used for carrying timber or packing material into the face. The construction of this conveyor enables it to be used in the thinnest workable seams, since the coal can be turned into it without lifting.

On account of the lightness of the pans and the facility with which they can be taken apart and assembled, the Ritchie conveyor can be moved forward very easily and rapidly. Under ordinary conditions 90 yds. of Ritchie conveyor can be moved forward in two hours.

The Thomson Conveyor.—This conveyor consists of a long tray made up in sections of L-section, and of a length equal to half the length of the coal-face, with haulage gear for reciprocating it along the coal-face and across a discharging bridge having special devices for automatically ejecting the coal into hutches underneath. The sections of the tray are about 6 ft. long, and are made of sheet steel riveted to two steel channels, having a retaining side 9 in. high formed on the side next the waste. They are attached to each other by a shackle and pin, the carrying part of each overlapping that of the next so as to form a continuous tray. The channels are designed to slide on the floor of the seam and across a bridge spanning the open hutchway or middle road. The bridge has mounted upon it standards into which are fitted blade-shaped implements placed slantwise across the tray, and arranged so as to automatically scoop the coal from the tray into the hutches as the tray is travelled from right to left or *vice versa*.

The conveyor is drawn to and fro along the coal-face by means of a steel wire rope guided by pulleys placed at each end of the face and in the airways, the haulage gear for actuating same being of the endless rope type, self-contained, and situated in the intake airway or other suitable place. The haulage gear can be provided suitable for operation either by compressed air or electric power. The centre road is brushed in the pavement about 10 ft. wide, and a double road for hutches is laid and maintained in it. In front of the conveyor flat plates are laid so that hutches may be easily shifted. This arrangement permits of the conveyor being worked continuously, so that one hutch can be brought forward to the conveyor while another is being filled, this in its turn being removed and a third brought forward while the second is being filled. The blades for automatically scooping the coal from the conveyor are of simple design, readily removable so as to fill a hutch opposite either set of rails, the standards being arranged so as to hold them in a suitable position for scooping the coal off the conveyor when travelling in either direction. An automatic device is provided on the haulage gear to stop the conveyor when it has reached either end of the coal-face, the arrangement of the whole operating gear being so

simple that the gear can be readily operated by a boy. The method of shifting the conveyor into a fresh line of face is to disconnect the rope from one end of the tray, the rope being then drawn up the

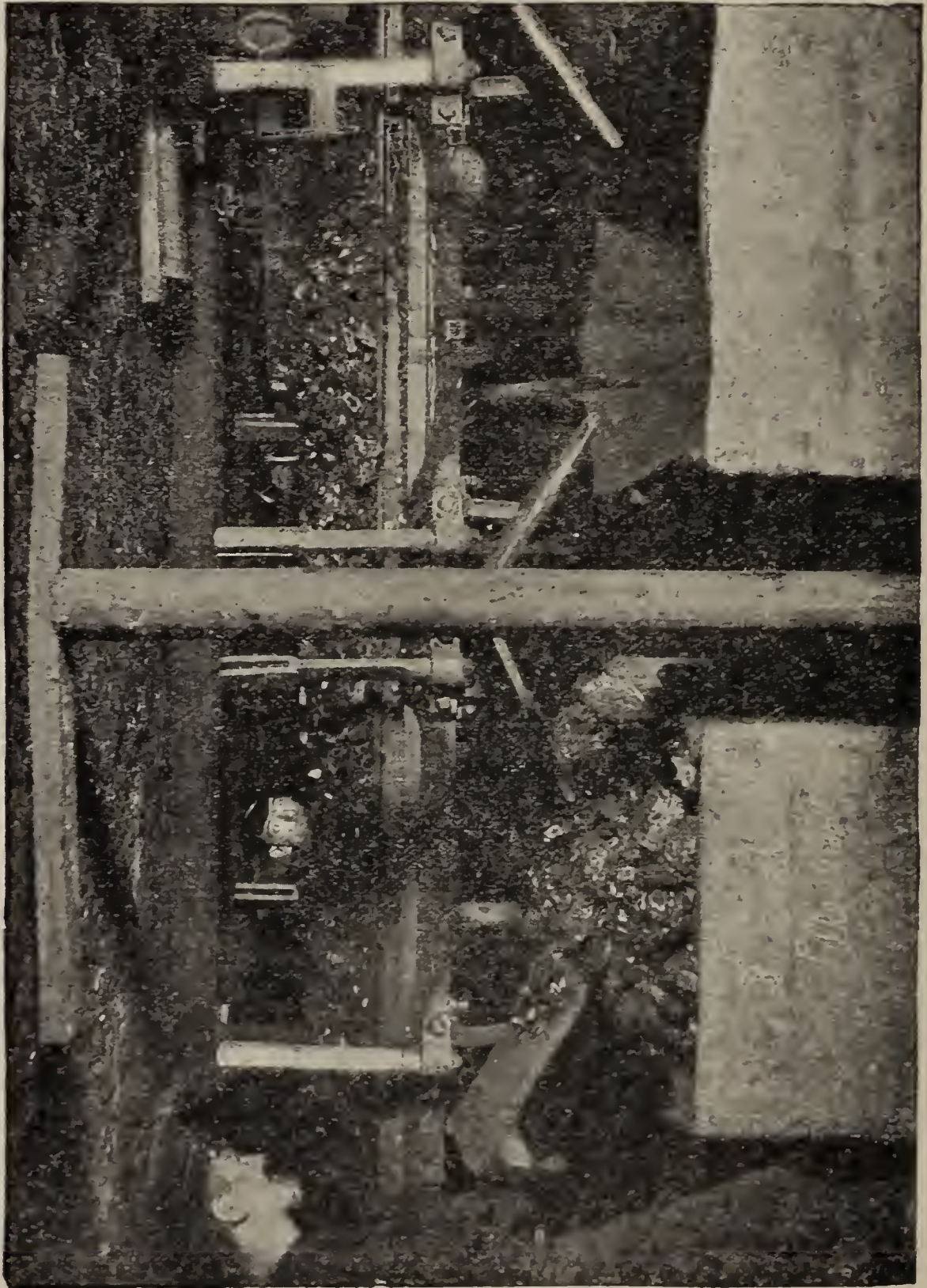


FIG. 499.—Bridge and discharge gear of the Thomson conveyor.

coal-face and brought back to the tray between the face and the props, the turn pulley being meanwhile shifted into the new position. Two or three props near the haulage road are removed temporarily, and, after the rope is reconnected to the conveyor, the haulage gear is put into operation and the conveyor is drawn into the new position.

The rope attached to the other end of the tray is now disconnected, drawn up the face, brought back between the face and the props and reconnected in like manner, the bridge and the turn pulley at this end being meanwhile shifted into their new positions. The rope is then tightened up, and the conveyor is ready for work in its new position.

Allardice Conveyor.—The conveyor *a* as constructed works on the pavement *b* and is continuous in its action so that the coal can, by means of the special construction of this conveyor, be loaded on to it at any part of the face. The pavement *b* shown at fig. 500, and which is of flat formation and not of trough shape, is formed of steel sheets of about 9 ft. lengths by 2 ft. 4 in. broad, and $\frac{1}{8}$ in. thickness. Each length is cut at one end (as shown at fig. 501) so that a portion is taken out of each corner of one end of each length *b*, whilst slits *d* are made in the other end, and the tongues thereby formed are made to slip in under the cut portion of the adjacent length, so that the several lengths are thereby joined to form a whole pavement, and a suitable stationary yet portable pavement *b* is made. These plates are only used in the event of the pavement being very soft or wet, or where it is very uneven.

On the top of the pavement *b* travels an endless chain comprising the conveyor proper, and composed of flat links *e*, having extending between them scrapers *f* spaced about 3 ft. or at any desired distance apart. The conveyor being so constructed is in itself a complete conveyor for the coal or other material, the links of the chain constituting the sides of the conveyor parallel to the face, and the scrapers constitute the front and rear sides, travelling over the stationary pavement to draw the coal (cut from the face) or other material along the pavement at right angles to the main road; these scrapers *f* discharging continuously (the coal cut) into a hutch *g* on the main road, which is bridged at this point as shown at figs. 501 and 502. The links *e*, which form an endless chain, are actuated by hexagonal pulleys or drums *h*, these links *e* passing over the pavement *b* as shown, and returning above the conveyor by passing over pulleys *i*.

Where the main road is on a level with the pavement of the working face the roof is brushed out over the main road, and the end part of the metal pavement *b* and of the conveyor carried at an angle such that the delivery end projects over the hutch, all as shown in fig. 500.

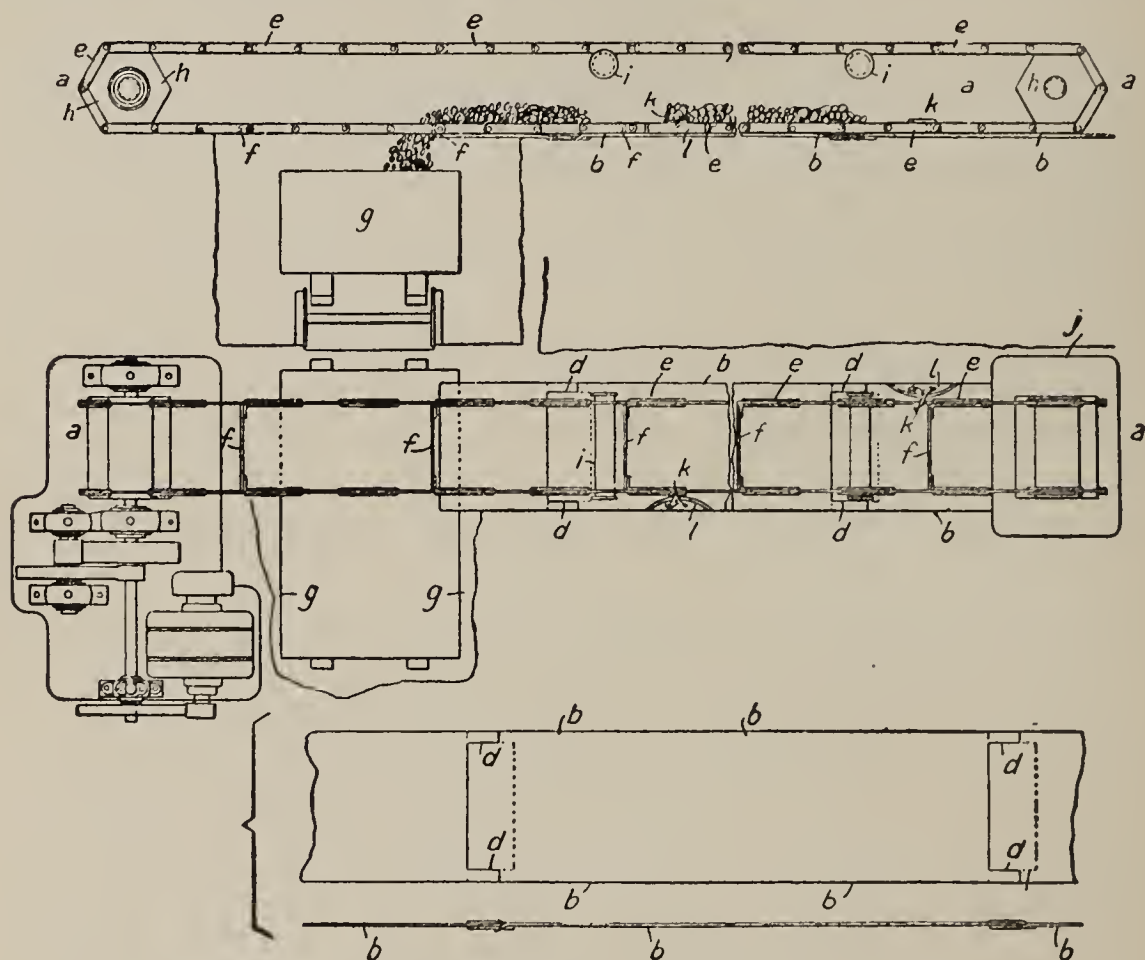
The hexagonal drums *h* are situated in a frame *j* carried on a bogie or skids, and the frame *j* is capable of being easily moved to any position required.

This continuous conveyor *a* is suitable for an undulating working, as suitable guides *k* can be fitted at the points required so as to keep the scrapers *f* down upon the pavement *b*, these guides being secured to wooden blocks *l* faced with iron plates as shown at figs. 500 and

501, the blocks *l* being kept down by pit props and acting to keep the conveyor in line.

The simplicity of construction of the conveyor *a* and accessories admits of easy manipulation of the pavement and scrapers constituting the conveyor from place to place according to needs by any ordinary workman, this arrangement of parts being specially suitable for use in a bad roof and a very low working.

The advantages secured by the use of a conveyor are its extreme



FIGS. 500, 501, and 502.

simplicity and cheapness of manufacture. The following advantages are also claimed for it:—

(1) That the conveyor travels on the pavement or on the ground when such ground is suitable.

(2) That the links of the chain comprising the conveyor act as sides to the conveyor whilst in motion and the disadvantage of having stationary sides is eliminated, thus dispensing with the necessity for the trough or the like heretofore employed.

(3) That the fact that the stationary sides are unnecessary in this conveyor admits of the greatest amount of coal being conveyed in any height of seam, particularly a low seam of 2 ft. or less than 2 ft.

(4) The disadvantage of a hard pavement brushing is by this construction and arrangement of conveyor eliminated.

(5) As the scrapers are bolted on to the links the breadth of the

conveyor may be varied at will. There is no doubt that this conveyor is one of the simplest, most efficient, easiest handled and worked, and most economical introduced for the conveying of coal at the face, and is specially suited for thin seams.

There are a number of other conveyors in use, the majority of which are modifications of those described, and for which space here is not available for descriptions of them. There can be no doubt but that the exhaustion of the thicker seams, and the increased costs for working of the thinner ones due to legislation and the higher wages now paid to the miners, will force colliery owners to adopt coal-cutters and conveyors more and more largely as the years go on. As a writer says: "The enforced concession of higher wages in the coal trade may be expected to result in a rapid increase in the number of coal-cutting and other labour-saving machines in mines. The rate of wages can only be forced up to a certain point. Beyond this point machinery steps in as labour's substitute." This is especially true of collieries having thin seams to work, and which have been hardly pressed by recent mining legislation and economic conditions. Where the conditions are suitable for conveyors, nearly all the advantages claimed for coal-cutting machines will apply equally well to them.

CHAPTER XII.

PUMPING.

IN all mines water is met with in greater or less abundance, shallow mines being, as a rule, more heavily watered than deep mines, owing to more frequent occurrence and greater width of the cracks or fissures in the overlying strata by which water can reach the workings, and to the absence therefrom of impermeable beds. In deep mines, which are seldom troubled with much water, such firm impermeable beds of strata are always present to some extent in the overlying strata, and prevent the water from entering the workings.

The different methods of draining mines of water are by means of adit levels, tanks or chests, siphons and pumps.

Adit levels can only be utilised under certain conditions, such as when the mine is situated on the side of a hill, or where the workings are at a higher level than some parts of the surrounding country. These conditions rarely exist in connection with coal-mining, unless in the opening out of virgin coal-fields in hilly countries. In metaliferous mining, adits are, however, much used, many of them being of great length and costing large sums of money. In the Freiberg district of Saxony there is an adit level $8\frac{1}{2}$ miles long, or, including branches, 25 miles in length, which cost about £360,000, the length of time occupied in this important work having been thirty-three years.* The Halkyn adit in Flintshire is about four miles in length, and drains a large area to a depth of 230 yds. It is estimated that this adit is now discharging 15 million gallons per twenty-four hours, or a total weight of 66,000 tons, the whole of this water being a natural flow.

Tanks or Chests.—If the quantity of water is not great and the pit is deep, it would be expensive to lay down a special pumping plant, while the water may be raised by water tanks or chests, especially if the winding engines are not fully employed drawing coal. The quantity of water dealt with in this way should not exceed 30 to 40 gallons per minute throughout the twenty-four

* *Ore and Stone Mining*, Sir C. Le Neve Foster, sixth edition, p. 462

hours, and this should be regarded as the maximum which should not be exceeded.

Where there is a considerable quantity of water to be dealt with, pumping will be found the cheapest method. In drawing water in tanks great injury is often done to the winding ropes, through part of the water falling back into the shaft and washing the grease or lubricating oil off them, and also by the great strain they have to undergo in lifting heavy tanks full of water. The load is often very much heavier than the usual load of coal. The dipping of the tank into the water causes 'slack,' while the vibration of the rope causes repeated bending to occur just above the capping, which tends to injure the rope, and if not carefully watched may result in breakage.

Siphons.—The siphon is not applicable in the same way that a pump is, since by the former water must always be delivered at a *lower* level than that of the receiving end of the pipes. We may, therefore, define a siphon as being *an apparatus for conveying a liquid from a higher to a lower level over an intervening height.*

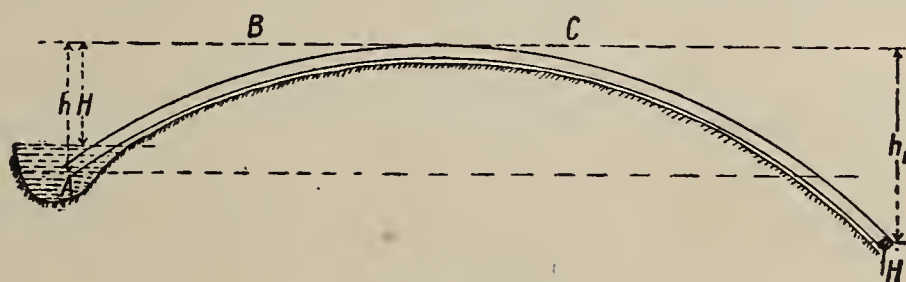


FIG. 503.—Siphon.

In construction the siphon is a simple piece of apparatus, and consists of a U-shaped pipe, one limb of which is longer than the other. The short limb dips into the liquid to be siphoned, and the other discharges it at a lower level. A vacuum or reduction of pressure being continually formed by the escape of water from the longer limb, the pressure of the atmosphere, acting on the free surface of the water into which the shorter limb dips, forces it up the latter, when, having reached the highest point of the column, it descends by gravity with a velocity proportionate to the difference of level between the outlet and the free surface of the source of supply.

Since the action of the siphon depends on the atmospheric pressure, it is obvious the height to which the water can rise will never be greater than that of the water barometer *at the time*, which at greatest is about $33\frac{3}{4}$ ft., no matter what the amount of fall may be at the discharge end. In practice the height to which the water will rise will not be more than 26 or 27 ft., the difference being due to the friction of the water in the pipes, but it will be better if the vertical height does not exceed 20 or 22 ft. The vertical height will, however, altogether depend on the length of the siphon column and its diameter, the longer the column and the smaller the diameter the less freely will the water rise.

To start the flow of water in the siphon, the two ends must be closed by plugs or taps. Water is then poured in at the highest point until the pipes are filled; this opening is then closed and the receiving and discharge ends are opened. The water in the pipe discharging produces a vacuum, thereby setting up a continuous flow. A better and more economical arrangement is to place a small hand-pump on the siphon at the highest point of the pipes, to pump the air out, and thus allow the water to rise.

The air and gases held in water are liberated on moderate reduction in pressure with great ease; and as nearly all water contains more or less dissolved gas, this will be liberated in the siphon at its highest point, and may accumulate there until the pressure equals that due to the acceleration head, when the siphon will cease to flow.

In laying down a siphon the greatest care should be exercised, so that it will have an opportunity of working under the most favourable conditions. The pipes should be laid with a regular gradient all the way to the highest point, and the pipes should be of sufficient diameter for the water to flow with a velocity that may reduce the friction to a minimum. The joints should be thoroughly air-tight; if the siphon has to continue working for any length of time, the joints ought to be run with lead, as this will be the most satisfactory and cheapest way in the end. A 'tail clack' should be put on at the bottom of the receiving end to prevent the pipes from getting empty when the siphon stops running.

Quantity of Water delivered by a Siphon.—The quantity of water which will be delivered by any given siphon depends upon: (a) the effective head or pressure; (b) the length of the siphon column; (c) the diameter of the pipes forming the column. A number of formulæ are given by different authorities for finding the quantity of water delivered under the conditions under which a siphon works, the following being probably the simplest:—

$$W = 4.71 \sqrt{\frac{D^5 \times H}{L}} \text{ (Etyelwein),}$$

where W = cubic feet of water delivered per minute.

D = diameter of pipe in inches.

H = total head of water in feet.

L = total length of column in feet.

4.71 = constant.

Example.—Let L = 2000 ft., H = 100 ft., and D = 3 in.

Then
$$W = 4.71 \sqrt{\frac{3^5 \times 100}{2000}} = 4.71 \sqrt{12.75} = 15.01 \text{ cub. ft. per minute,}$$

and $15.01 \times 6.25 = 93.75$ gallons per minute;

or by first finding the velocity of the water at point of delivery by the formula:

$$v = 48 \sqrt{\frac{dH}{L + 54d}} \text{ (Hawksley),}$$

where v = velocity at which water will issue in feet per second,
 d = diameter of pipe expressed in feet,
 H and L having the same value as before,

$$\text{then } v = 48 \sqrt{\frac{.25 \times 100}{2000 + (54 \times .25)}} = 48 \sqrt{\frac{25}{2013.5}} = 5.3 \text{ ft. per second,}$$

and gallons per minute = $v \times 293.7 d^2 = 5.3 \times 293.7 \times .25^2 = 97$ gallons per minute,
 which is a slightly different result from that given by Etyelwein's rule.

Head or Pressure necessary to overcome Friction in Siphon Column.
 —This has an important bearing in the proper working of a siphon, as in a long siphon column with pipes of insufficient diameter a large proportion of the head may be lost in overcoming the internal resistance of the water flowing through the pipes. This resistance may be calculated by the formula :

$$h = \frac{L}{r} \left(.0036 + \frac{.0043}{\sqrt{v}} \right) \frac{v^2}{2g} \text{ (Weisbach),}$$

where h = head in feet lost by friction. v = velocity of flow in feet per second.
 L = length of column in feet. g = gravity = 32.
 r = mean radius of pipe in feet.

The value of v may be found by the preceding formula, and taking the above example :

$$\begin{aligned} h &= \frac{2000}{0.125} \left(.0036 + \frac{.0043}{\sqrt{5.3}} \right) \frac{(5.3)^2}{2 \times 32} \\ &= 16,000 (.0036 + .0018) \frac{28}{64} \\ &= 16,000 \times .0054 \times \frac{28}{64} \end{aligned}$$

$$\therefore h = 38.8 \text{ ft.,}$$

or $38.8 \times .434 = 16.84$ lbs. pressure per square inch lost through friction.

This head or pressure lost through friction only holds true for a straight line of pipes ; if there are a number of bends or knee-joints the friction, and consequently the value of h , will be increased. The loss of head by friction in bends or knees may be calculated by the formula :

$$h = \frac{A}{180} \left\{ .131 + 1.847 \left(\frac{R}{r} \right)^{\frac{1}{2}} \right\} \frac{v^2}{2g} \text{ (Weisbach),}$$

where A = angle in degrees of bend or knee with forward line of direction of flow.

R = radius of centre line of bend in inches.

r = radius of pipe in inches = $\frac{1}{2}$ diameter.

h = head of water necessary to overcome friction of bend or knee.

v and g = same value as in previous formulæ.

Or the value of h may be found as follows :—

$$\text{For knees, } h = .0155 v^2 K \text{ (Molesworth).}$$

The value of K is as follows for different angles :—

A° =	20°	40°	60°	80°	90°	100°	120°
K =	·046	·139	·364	·74	·98	1·26	1·86

For bends, $h = .0155v^2\left(\frac{A}{180} L\right)$ (Molesworth)

Values of L with various ratios of the radius of bend to radius of bore :—

When $\frac{r}{R} =$	0·1	0·2	0·3	0·4	0·5	0·6	0·7	0·8	0·9	1·00
L =	·131	·138	·158	·206	·294	·44	·66	·98	1·4	2·00

The loss of head due to bends may be found approximately from the formula : $h = c \frac{v^2}{2g}$.

Values of c are—

- For elbow ; $c = 1$.
- Right-angle bend, radius = diameter of pipe ; $c = 0·3$.
- Right-angle bend, radius = 3 diameters ; $c = 0·14$.

Pumps.—The best and most usual method for raising water from mines is by pumping. The first point of importance is the capacity of the plant required. In deciding this it is necessary to ascertain, as nearly as possible, the maximum quantity of water likely to be met with both in the shaft and in the workings. In sinking a shaft in a new and untried district it is impossible to do so, but in districts which have been well opened out it can often be done without much difficulty. The plant should be capable of raising a larger quantity of water than any ascertained maximum, so that a sudden inflow could be dealt with, if necessary, to prevent the flooding of the workings.

Choice of Pumping Plant.—The choice of the proper and best type of pumping plant is a matter of great importance, and worth the most careful consideration, especially where large quantities of water have to be dealt with in mines. The three essential qualities required in mine pumps are: (1) efficiency, (2) economy, and (3) freedom from breakdowns. Dealing with water at collieries may be considered in two divisions, viz.: (a) raising water in the shafts; (b) pumping water in the underground workings.

Shaft Pumps.—A decade or so ago practically the only method adopted for raising water in shafts was by means of the ordinary bucket or ram pump, operated by rods which were worked by a horizontal or vertical steam engine situated at the surface. In these

days the question of economy was not of such importance as it has now become, as steam was often raised by the burning in the boilers of unsaleable small coal. With the introduction of coal-screening and coal-washing plant, the greater part of the small stuffs is now made more valuable, commanding a ready market at good prices. The introduction of coal-cutting machinery has in addition reduced the percentage of small coal produced, and hence the question of economy in fuel burning for steam raising is now a question of greater importance, as the price of coal has gone up. There can be no question that many of the steam-driven pumping plants at collieries were very wasteful as regards the consumption of fuel, and, therefore, highly uneconomical. It was not at all uncommon for 10 to 12 per cent. of the total output to be consumed in raising steam for the various engines at a colliery, with a consumption of 9 to 10 lbs. of fuel per indicated horse-power per hour. All steam pumping engines should not be placed in this category, for there can be no doubt but that the old single-cylindereed beam engine of the Cornish type was, and probably is, one of the most economical of steam engines, especially with a low steam pressure, say not exceeding 40 lbs. per square inch. Engines of this class have given the high duty of 80,000,000 to 100,000,000 foot-pounds per cwt. of small coal burned, compared with the duty of 30,000,000 to 50,000,000 foot-pounds per cwt. given by many modern steam and electrical pumping plants.

Shaft pumps may be divided into the following classes: (1) Single or compound vertical engines placed on the surface, operating the pumps by means of rods; (2) single or compound horizontal engines, geared or non-geared, placed at the surface, and transmitting power through rods to the pumps; (3) steam pumps, usually of the horizontal type, placed at the bottom of the shaft, or insets in the shaft, and forcing the water direct to the surface; (4) compressed air or electrically driven pumps placed at insets or the bottom of the shaft, and forcing the water direct to the surface; (5) hydraulic pressure pumps placed and worked as in (3) and (4).

Underground pumps for the in-by-e workings are usually of the following types: (a) Horizontal steam pumps, geared or non-geared; (b) compressed air- or electrically driven pumps; (c) hydraulic pressure pumps.

Descriptions and details of types of these pumps, both for shafts and in-by-e workings, are given further on in this chapter and need not be further referred to here.

Pumps for raising water in mines are generally of the reciprocating type and may be classified as: (1) Plunger or ram pumps; (2) piston pumps; and (3) bucket or lift pumps. Other kinds of pumps are also employed, such as the centrifugal and Fontigaine pumps, but the latter can only be adopted for limited lifts.

Conditions affecting the Working of Pumps.—The working of pumps is influenced by various conditions, such as: (1) The height at which

they are placed above sea level, *i.e.* atmospheric pressure conditions. (2) The temperature of the water. (3) The size and length of delivery and suction pipes. (4) The area, weight, and lift of valves.

These conditions have all to be carefully considered in designing or deciding upon the necessary pumping plant for a given position either on the surface or underground. The chief requirements mining pumps are expected to fulfil are: (a) They should be capable of working for long periods with little repair, packing, or adjustment. (b) They should be capable of being operated under water (a particularly desirable feature in sinking pumps). (c) They should be capable of passing sandy or dirty water, and sometimes acid water, without too rapid deterioration or corrosion. (d) Their speed and capacity should be easily adjustable to suit the varying inflow of water. Whatever type of pump is installed, it ought to be of such a capacity that it will be able to do the whole of the work necessary in a portion of the twenty-four hours.

Pump Fittings. Pipes.—The pipes used in connection with pumps may be made of wood, cast iron, or wrought iron or steel.

Wood pipes are seldom if ever used in Britain for pumping, but in some parts of the United States, and where the climate is extremely variable, they are much used for certain descriptions of work, when the pressures are light. For pressures not exceeding 85 to 90 lbs. per sq. in., or a head of water equal to about 200 ft., they are said to be economical. The advantages claimed for them are that they contract and expand to only a small extent, and are therefore well suited for climatic changes, while they offer little resistance to the flow of water, and do not decay readily if water is kept constantly flowing through them. They are built up with staves, much like ordinary barrels, and are strongly bound with wrought-iron hoops.

Cast-iron pipes were formerly almost exclusively used for pumping, and are so still to a very large extent, but for certain classes of work steel or wrought-iron pipes are displacing them. The great disadvantage of cast-iron pipes is, that where they have to be of a large section, they are very heavy and difficult to handle. The joints in this class of pipe are usually made with a common flat flange, and an india-rubber ring inserted between them, the joint being well secured by nuts and bolts. For deep lifts and heavy pressures the top flange should have a groove cut in it, while on the bottom flange a rib should be cast to fit into this groove. An india-rubber joint of circular section is fitted into the groove and the rib on the other pipe fitted on it, the two ends being well screwed down with nuts and bolts in the usual way.

Wrought-Iron and Steel Pipes.—Pipes of this class are now extensively used for pumping purposes, and possess considerable advantages over those made of cast iron. They are more easily handled, and also cheaper, while sections of any reasonable length can be readily cut off and fitted wherever required. It has been

found, however, in practice, that when dirty or acid water has to be dealt with, that cast-iron pipes are best. Pipes of large section should be made of mild steel, which is more homogeneous and possesses greater strength than wrought iron. The joint used for this class of pipe is somewhat different from that used in cast-iron pipes. Leaded joints are sometimes used when the pipes are permanently fixed, such as in siphons, but they are not suitable for shaft work.

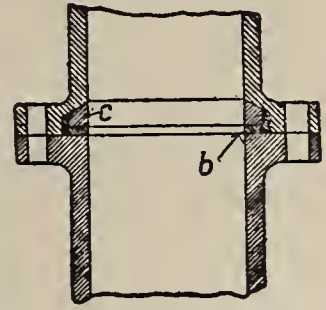
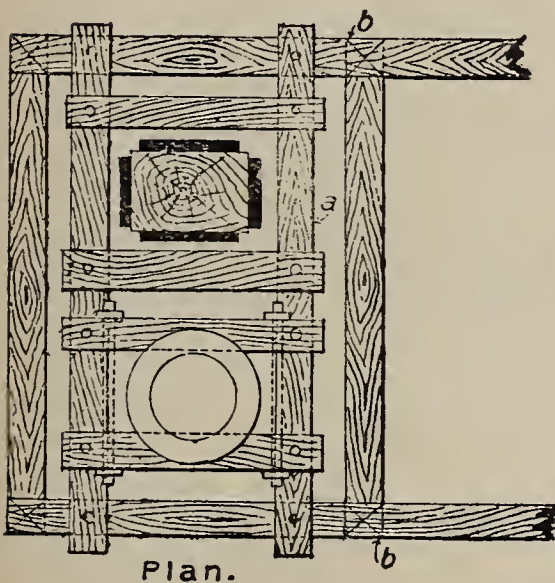
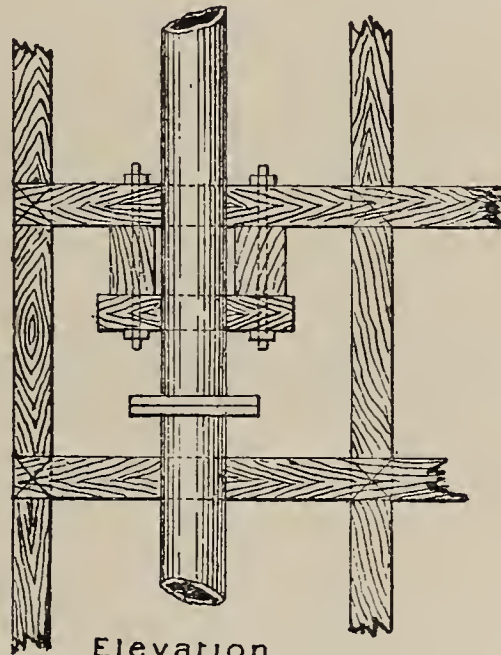


FIG. 504.

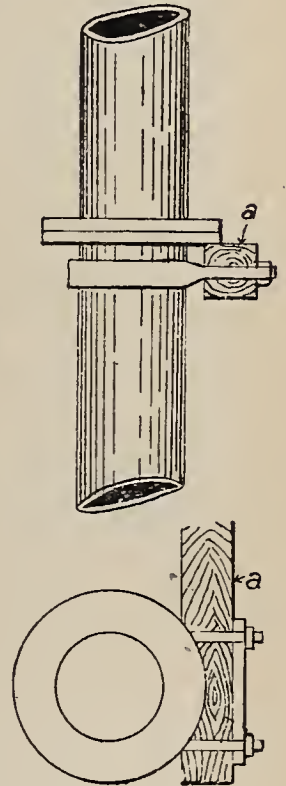
Eadie's joint, which is somewhat similar, is also much used, both for water and compressed air. At Kladno, Bohemia, a flange packing has been successfully adopted for a head of water of 1700 ft. One of the flanges is recessed to admit a ring of rubber or metal of L shape, this ring being held in position by a rigid metal ring, which gives great security and tightness to the joint (fig. 504).



Plan.



Elevation.



FIGS. 505, 506, 507, and 508.—Pipe supports.

Expansion Joints.—Where the column of steel or wrought iron used is very long in either shafts or inclines, expansion joints should be used to prevent 'elbowing.' The commonest forms are merely joints with an ordinary stuffing-box, containing hydraulic packing. The pipe entering the stuffing-box should be perfectly smooth, and kept well greased.

Pipe Supports.—Pipes in a vertical shaft ought to be properly supported to keep them in a vertical line. It is also necessary to contrive some support for the weight of a long column of pipes. A support ought to be put in every eight or ten fathoms at least.

A common method is to lay timber pieces across the shaft, and bolt them firmly down to the buntons at one end, and fix them into the strata at the other.

Sometimes the rods are supported by the method shown in figs. 505, 506. Two cross pieces *a a* are laid across the pieces *b b*, the two sets of timbers being firmly bolted together with hanging and horizontal bolts. When there is little room in the shaft, the method employed is to use a gland, bent to the circle of the pipes, with two screwed ends, over which a plate of iron is fixed. A piece of wood *a*, like a saddle, may be fixed between the pipe and the support, and the gland then tightened by the nuts (figs. 507, 508).

Size of Pipes.—The diameter of pipes will, of course, depend on whether they are connected to a lift or force pump, and on the velocity at which the water is required to flow. For ordinary lift pumps the pipes ought to be $\frac{1}{2}$ in. larger than the working barrel, so that the bucket can be drawn through them if necessary, and changed at the top of the lift or on the surface. With force or plunger pumps this is unnecessary, and therefore, for this class of pump, the diameter of the delivery pipes may be smaller.

The velocity of the flow of water in pipes varies as the area and $\text{area} \propto D^2$, therefore velocity $\propto D^2$.

$$\therefore d_1^2 : d_2^2 :: \sqrt{1} : \sqrt{2} \text{ or } \frac{d_1^2}{d_2^2} = \frac{\sqrt{1}}{\sqrt{2}} ; \text{ where } d_1 = \text{diameter of plunger.}$$

$$\begin{aligned} d_2 &= \text{diameter of rising main.} \\ \sqrt{1} &= \text{velocity of water in rising main.} \\ \sqrt{2} &= \text{velocity of plunger.} \end{aligned}$$

If the maximum velocity of the water in the rising main is not to exceed say 150 ft. per minute, and the speed of plunger 60 ft. per minute, diameter of plunger 15 in., what should be the diameter of the rising main?

$$\text{Here } d = 15 \text{ in., } \sqrt{1} = 150 \text{ ft. per min., } \sqrt{2} = 60 \text{ ft. per min. } \therefore \frac{15^2}{d_2^2} = \frac{150}{60}$$

$$\therefore 150 d_2^2 = 60 \times 15^2 \therefore d_2 = \sqrt{\frac{60 \times 225}{150}} = \sqrt{90} = 9.48 \text{ in. So that pipes about}$$

$9\frac{1}{2}$ in. diameter would be suitable for a 15 in. diameter plunger under the above conditions. A practical rule is to make the rising main about $\frac{3}{4}$ the area of that of the plunger.

Area of Suction Pipes.—For the efficient working of pumps it is necessary that the suction pipes should be of such an area as will ensure the pump barrels being completely filled at each stroke. Many pumps do not work satisfactorily owing to the suction pipes being of insufficient area. This is especially so with quick-running pumps where the water is moving at a fairly high velocity. It should be remembered that the frictional resistance increases rapidly when the water is propelled through the pipes at a high speed. This is specially of importance in underground pumps where the

suction pipes may be of considerable length. With velocities up to about 3 ft. per second the frictional resistance is simply proportional to the speed, but at higher velocities the friction may increase as much as the square of the speed, or even in a greater ratio. It is calculated that when water is forced through a 4-in. diameter pipe at a velocity of $1\frac{1}{4}$ ft. per second, or 75 ft. per minute, the loss in pressure due to friction is less than $\frac{1}{10}$ lb. per square inch per 100 ft. of piping, whereas at a velocity of 19 ft. per second the loss will rise to over 14 lbs. per square inch per 100 ft. of piping. In actual practice with pumps having a piston or ram speed of 60 to 100 ft. per minute it is usual to employ suction pipes of such dimensions that the velocity of the water through them shall not exceed from $4\frac{1}{2}$ to 6 ft. per second, or 270 to 360 ft. per minute. It would be more satisfactory, however, if the speed of water in the suction pipes did not exceed the speed of the piston or ram.

The loss of head due to friction may be found from the formulæ

given on p. 222, or from the formula: $h = k \frac{lv^2}{d^2g}$,

where h = loss of head in feet.

l = length of pipe in feet.

d = diameter of pipe in feet.

v = velocity of water in feet per second.

k = constant: .02 to .03 depending on smoothness of pipe.

The area of pipe can be found from the formula: $A = \frac{G}{2.6 \times v}$.

A = area of pipe in square inches.

v = velocity of water in feet per second.

G = gallons per minute.

Example.—What size of pipe would be required if 400 gallons of water is raised per minute, the velocity of the water in the suction pipes to be 250 ft. per minute?

$$A = \frac{400}{2.6 \times \frac{250}{60}} = 38.92 \text{ sq. in.}$$

$$\therefore d = \sqrt{\frac{38.92}{.7854}} = 6.85, \text{ say 7 in. diameter,}$$

and allowing 10 per cent. for friction,

$$d = \frac{7 \times 100}{90} = 7.77 \text{ in.}$$

Thickness of Pipes.—The thickness of pipe required will vary according to the pressure to be sustained and the constancy or otherwise of the pressure. If the pressure is uniform, the pipes may be made much lighter than where it varies, by the stopping and starting of the water column, as in a single-acting pump. Similarly, if the water is corrosive the pipes will require to be extra strong.

Pipes are liable to rupture either transversely or longitudinally. Their strength may be calculated according to the following formula:

Rupture longitudinally : $P \times D = 2t \times f$.

Rupture transversely : $P \times D^2 \times .7854 = t \times D \times 3.1416 \times f$.

Where P = pressure in lbs. per square inch, t = thickness of pipes in inches.

D = internal diameter of pipes in inches, f = coefficient of metal employed
(cast iron = 16,000, wrought iron = 50,000, steel = 70,000).

Example.—Given a set of pipes 10 in. diameter (internal) and $\frac{5}{8}$ in. thick : find the pressure required to rupture them (a) transversely, and (b) longitudinally.

(a) *Longitudinally* :

$$P \times D = 2t \times f$$

$$P \times 10 = 2 \times \frac{5}{8} \times 16,500$$

$$\therefore P = \frac{2 \times 5 \times 16,500}{10 \times 8} = 2000 \text{ lbs. per square inch.}$$

(b) *Transversely* :

$$P \times D^2 \times .7854 = t \times D \times 3.1416 \times f$$

$$P \times 10 = 4 \times \frac{5}{8} \times 16,500$$

$$\therefore P = \frac{4 \times \frac{5}{8} \times 16,500}{10} = 4000 \text{ lbs. per square inch.}$$

For safety, the working pressure would be taken at $\frac{1}{3}$ th to $\frac{1}{5}$ th of the above pressures.

The thickness of pipes required can be obtained from the formula deduced from above, $t = \frac{PD}{2f}$ (f , the safe working pressure for cast iron, should be taken at not more than 1500 lbs. per square inch).

Example.—Given a head of water of 600 ft., and internal diameter of pipe 10 in. : what thickness would the pipes require to be?

$$t = \frac{600 \times .434 \times 10}{2 \times 1500} = 0.86 \text{ in.}$$

Where the pipes are exposed to shock as in mine pumps, the practice is to make them with a greater thickness of metal than would be brought out by the foregoing formulæ, which are based on the assumption that the pressure exerted is constant, and which takes no account of defects in casting, etc. The following rule will be found to coincide more closely with practice :—

$$t = (.00022 Pd) + .15\sqrt{d} \text{ (Molesworth).}$$

Where t = thickness of metal in inches or decimals of an inch.

P = pressure in lbs. per sq. in. due to head of water = $h \times .434$.

d = dia. of pipe in inches.

Taking the example as already worked out :—

$$t = (.00022 \times 600 \times .434 \times 10) + .15\sqrt{10} = .5728 + .4740 = 1.046 \text{ in.}$$

For working barrels $t = (.00017 Pd) + 1.26$.

To find the necessary thickness for wrought-iron pipes with a given head of water :—

$$t = .0000288 hd$$

$$\text{or } t = .0000665 Pd$$

Weight of Cast-iron Pipes.— $W = 2.45 (D^2 - d^2)$

Where W = weight in lbs. per ft. of pipe.

D = external diam. of pipe in inches.

d = internal ,, ,,

2.45 = constant.

The weight of two flanges may be taken as approximately equal to one foot of pipe.

In practice the thickness of pipes used is somewhat greater than the calculated thickness, to provide for wear and other contingencies. The following thicknesses of metal are used for cast-iron pipes of different diameter and for different heads of water :—

Inside Diameter of Pipe.	50 ft. Head or 21.65 lbs. Press.	100 ft. Head or 43.3 lbs. Press.	150 ft. Head or 64.85 lbs. Press.	200 ft. Head or 86.6 lbs. Press.	250 ft. Head or 108.25 lbs. Press.	300 ft. Head or 129.9 lbs. Press.
	Thickness of Metal.	Thickness of Metal.	Thickness of Metal	Thickness of Metal.	Thickness of Metal.	Thickness of Metal.
ins.	ins.	ins.	ins.	ins.	ins.	ins.
8	0.422	0.450	0.474	0.498	0.522	0.546
10	0.459	0.489	0.519	0.549	0.579	0.609
12	0.491	0.527	0.563	0.599	0.635	0.671
14	0.524	0.566	0.608	0.650	0.692	0.734
16	0.580	0.604	0.625	0.700	0.748	0.796
18	0.589	0.643	0.697	0.751	0.805	0.859
20	0.622	0.682	0.742	0.802	0.862	0.922
24	0.687	0.759	0.831	0.903	0.975	1.047

Pump Valves or Clacks.—Valves for pumps used in mines are of various types, their design and construction depending upon whether the water is clean or gritty, acid or otherwise corrosive, and whether the temperature is high or low.

There are three types of valves generally used, viz. :—

Hinged valves, commonly called clacks.

Straight lift valves, which rise vertically on their seats.

Flexible valves, which alter their form on opening.

In ordinary pumps the valves mostly used are of the hinged type, with either a single or double lid (see figs. 509, 510, 513).

For direct-driven pumps, straight lift valves are very largely used. Figs. 511, 512 show such a valve, which is commonly used for any pressure up to 500 lbs. per square inch. Double flap or 'butterfly' lid valves are almost entirely used for buckets and clack valves of lifting sets. Thus the latter valve is very suitable for lifts which do not exceed 30 to 40 fms.; but for lifts over 30 fms., iron hinges should be used instead of leather. Figs. 514, 515 show this type of valve.

The valves are usually faced with leather, but where acid water has to be pumped, rubber is to be preferred, and sometimes vulcanite or brass is used instead of iron for mounting. If the water is hot, a rubber composition is used for facing them.

Hinged valves are more liable to leakage than straight lift valves, as they wear more unequally; but the tightness of a valve depends a good deal on the pressure per square inch on its face, while the less bearing it has, the more difficult it is to keep it tight. On the other hand, the larger the bearing surface, the greater the wear. Leakage

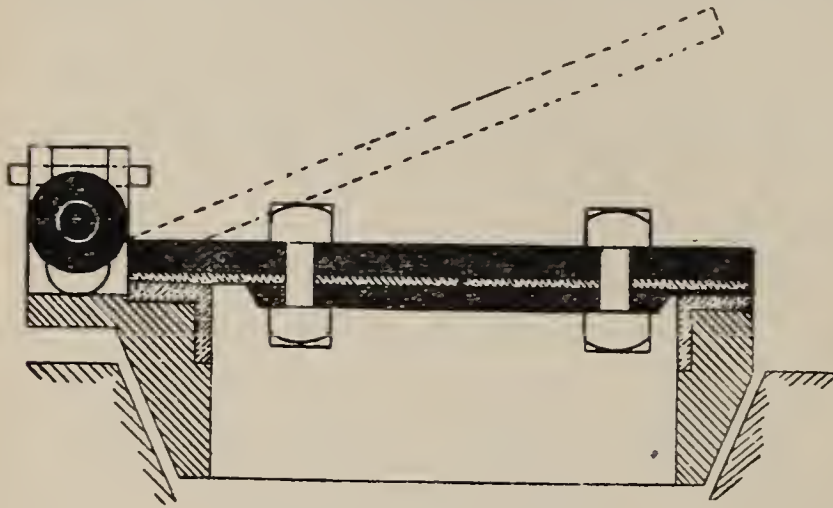


FIG. 509.—Elevation.

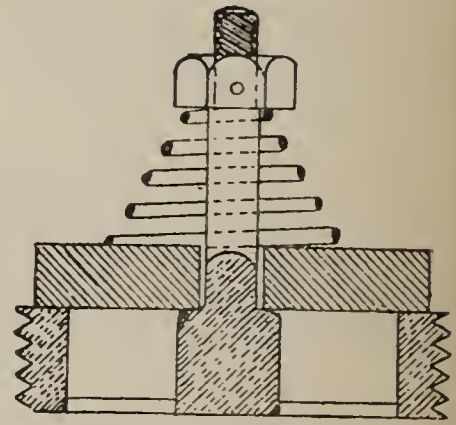
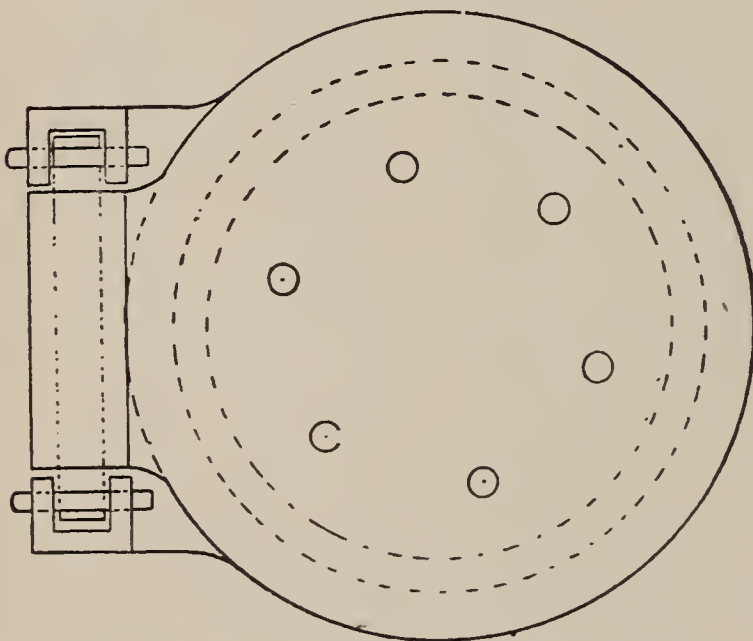
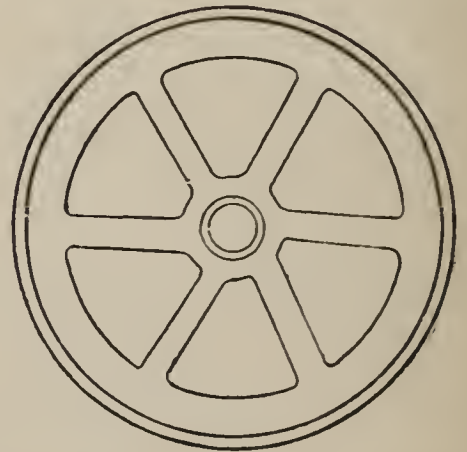


FIG. 511.—Elevation.

FIG. 510.—Plan.
Common hinged clack valve.FIG. 512.—Plan.
Straight lift valve.

with hinged valves will probably amount to 10 or 12 per cent., and for lift valves 5 to $7\frac{1}{2}$ per cent.

A valve should not be larger than 10 in. in diameter. If larger outlets than this are required they should be made double, for when the valve is very large it is impossible to prevent it from 'hammering,' owing to the weight of water above. Good valves should be simple in construction, and not liable to get out of order. They should present as little resistance as possible to the flow of water, and should close as quickly as possible on the completion of the stroke.

Buckets.—Buckets are usually constructed of iron with leather mountings, the shell being often composed of gutta-percha, or a composition containing this. They are provided at the top with two flap lids opening from the centre. Their construction will be understood from figs. 516, 517, 518. The leather and other mountings should be of the very best quality, otherwise they will rapidly wear, and cause much trouble and annoyance. For very high lifts the

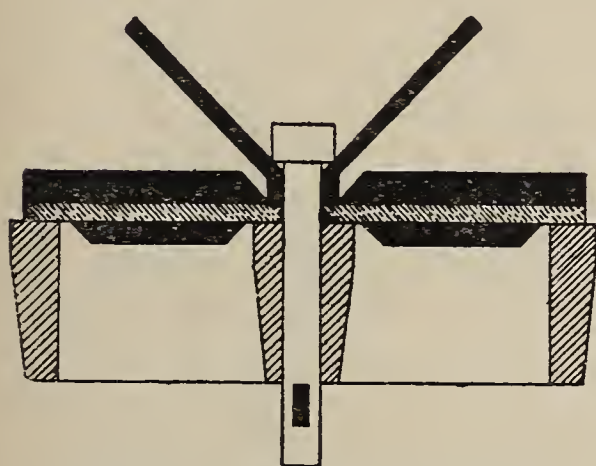


FIG. 513.—Double hinged valve.

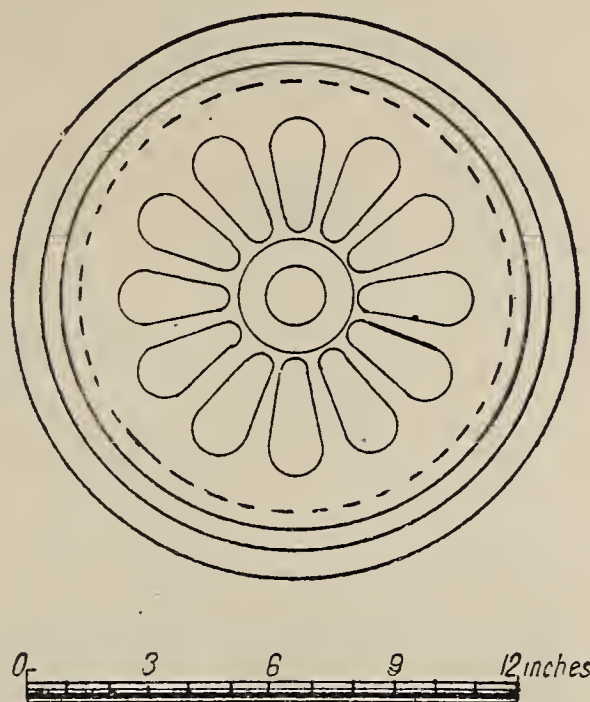


FIG. 515.—Flexible valve plan.

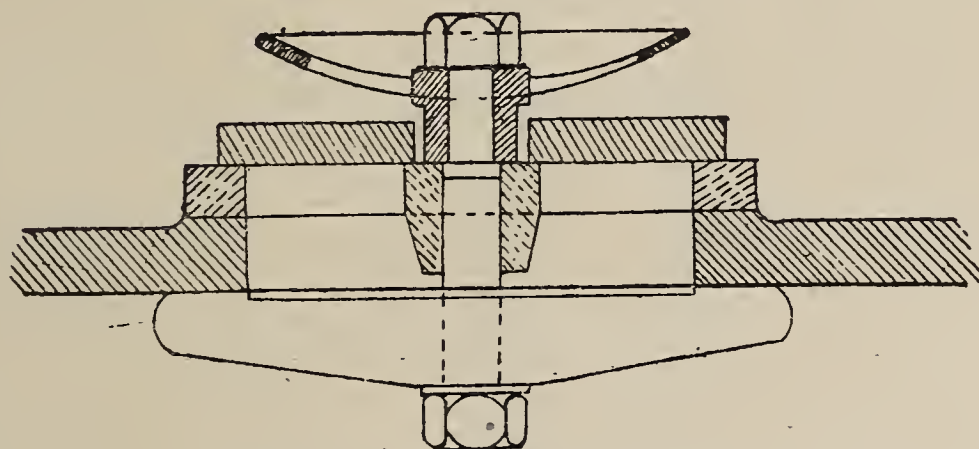


FIG. 514.—Flexible valve elevation.

bucket is sometimes made entirely of metal. Fig. 519 shows such a bucket constructed of steel, and simply made to fit sufficiently well to prevent leakage, and not occasion too much friction.

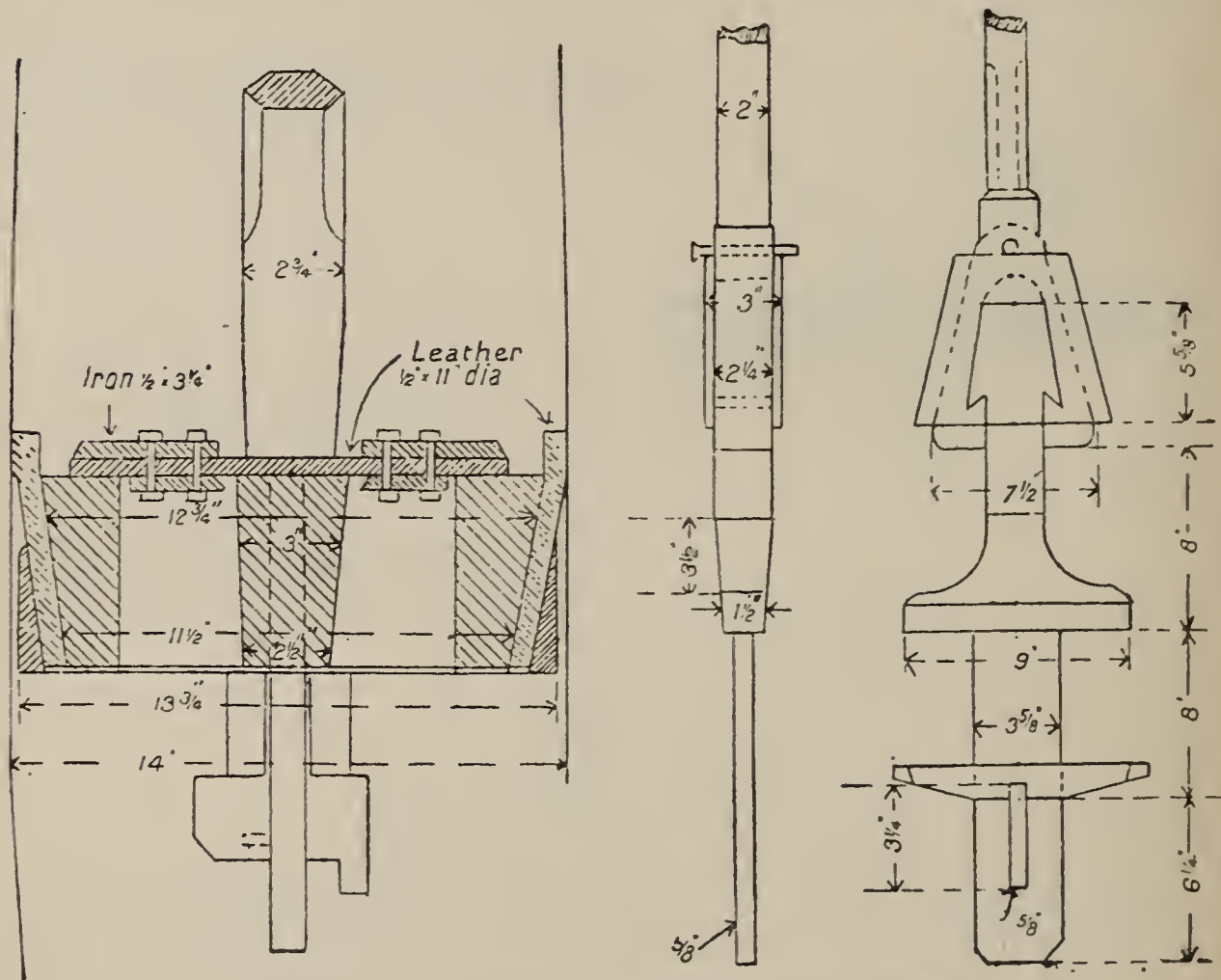
The bodies of buckets of this class are made a good deal longer than ordinary sorts, which increases their efficiency against leakage. Grooves are sometimes turned in the body of the bucket, and brass rings inserted to make them work smoothly.

Plungers or Rams.—Plungers for ordinary pit pumps are usually made of cast iron ; for small pumps, or where the water contains much

acid, they are sometimes made of brass or lined with it. The plunger is usually made hollow, and connected to the rods by the methods shown in figs. 520, 521. In large hollow plungers the wood rod is sometimes simply driven in firmly to give it sufficient gripping power to prevent it from slipping. The plunger should be accurately turned and finished smooth, so as to work with as little friction as possible.

Plungers should be kept well greased at the top of the stuffing-box, as this will reduce friction and tend to easy working.

Pump-rods or Spears.—Pump-rods are usually made of pieces of



FIGS. 516, 517, and 518.—Bucket and connections.

pitch pine or oak, 18 to 45 ft. long, 30 ft. being a common length, and square or rectangular in section, joined together by strong iron plates to withstand the heavy strains to which they are subjected. Iron rods are sometimes used, but not very extensively, at least in Britain. They are more difficult to make and to put together, get more easily out of order, and are more difficult to repair than wooden rods. Speaking of iron pump-rods, Callon says: "The lightness of rods is not usually a point to be desired, because we are often led, on the contrary, to give them additional weight. We may fairly conclude that this is one of those cases more frequent in practice than we think, in which the word *modification* need not necessarily signify *improvement*, and that metallic rods have no decided superiority over wooden ones."

On the whole, rods made of pitch pine have been found to be in every way better suited for pumping. The size of section will vary according to the size of pump and length of lift. The pieces are joined one to another either by a common square joint or a scarf joint, and held together by iron plates and bolts (see figs. 522, 523,

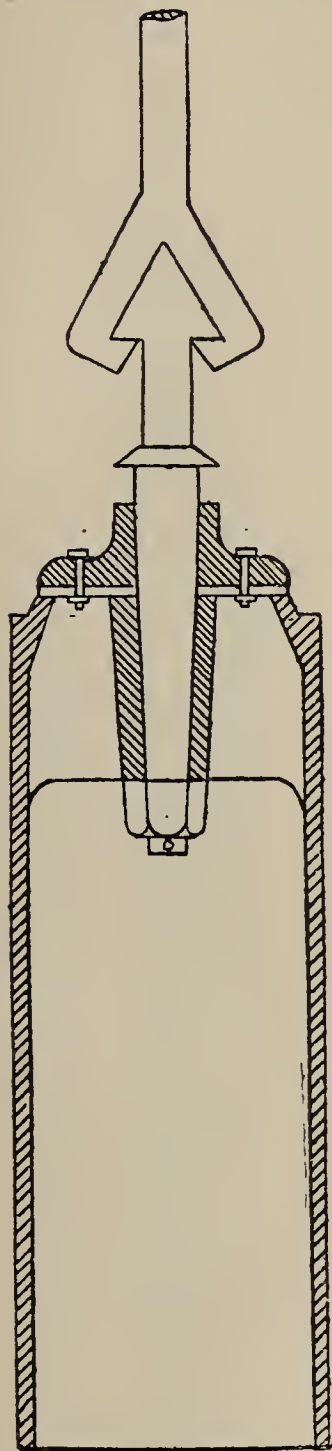


FIG. 519.—Steel bucket.

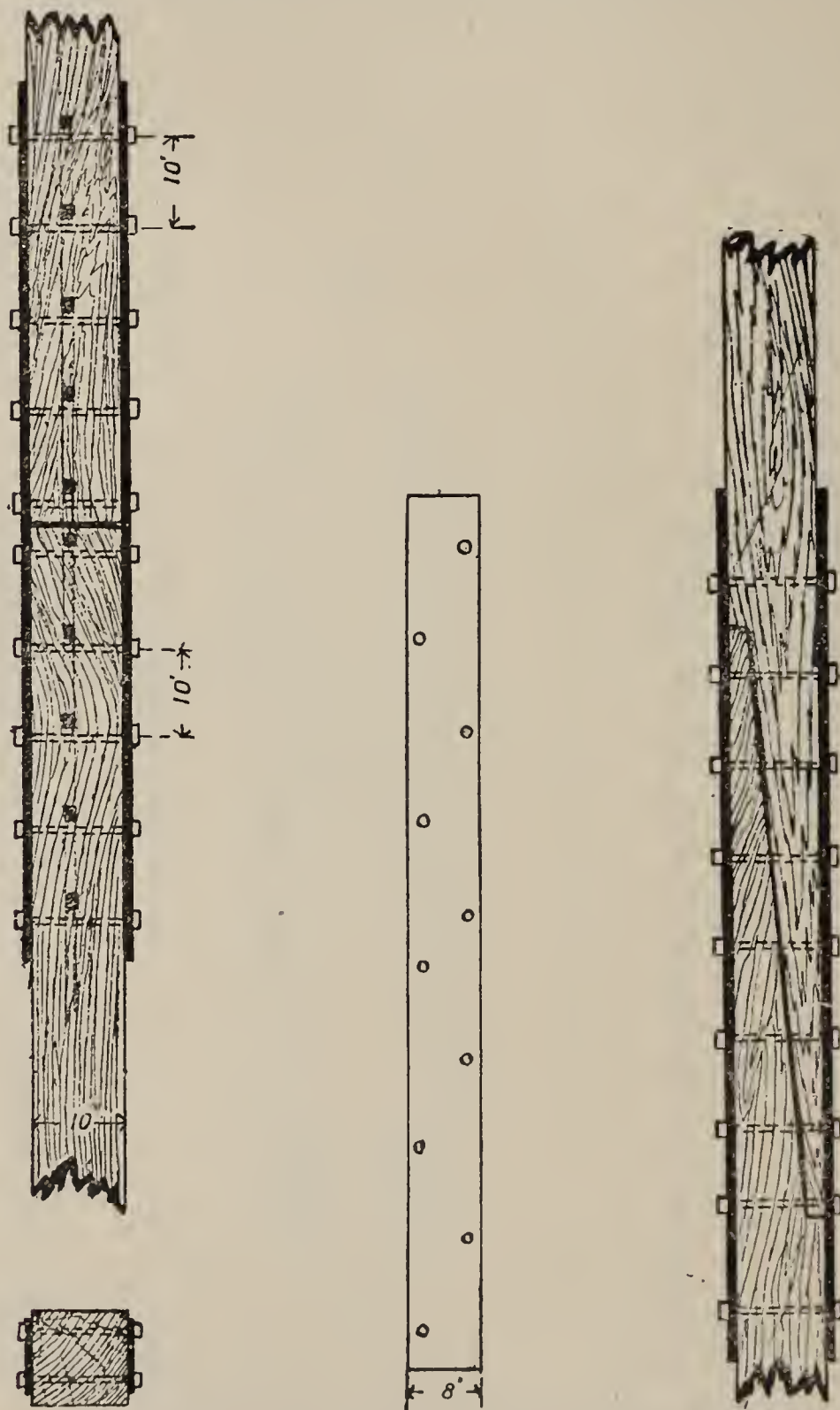


FIGS. 520 and 521.—Plunger connections.

524). For plunger or force pumps the joint should be made square, as the whole of the pressure or work done is practically on the down stroke; but for bucket or lift pumps, where the tension is greatest on the up stroke, the joint is often scarfed in a zigzag fashion which is found to give better results.

For rods up to 8 in. or 9 in. square, two straps or plates and

'dumb' or 'clink' bolts, put in at right angles to the plate bolts and a little above them (fig. 522), will be sufficient to prevent the other bolts from tearing along the grain of the wood. The bolt holes in the plates should be bored in zigzag fashion, otherwise if they are



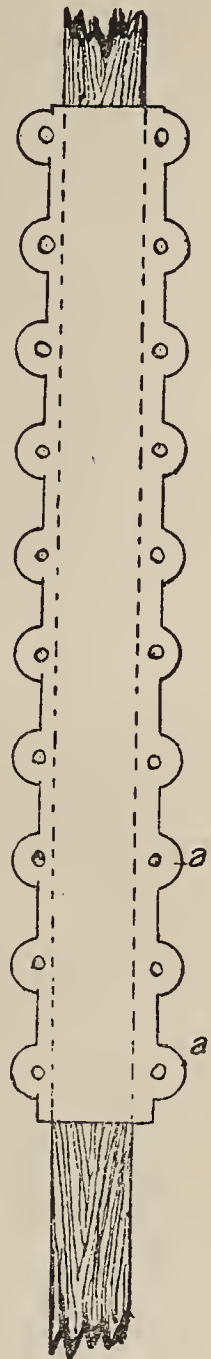
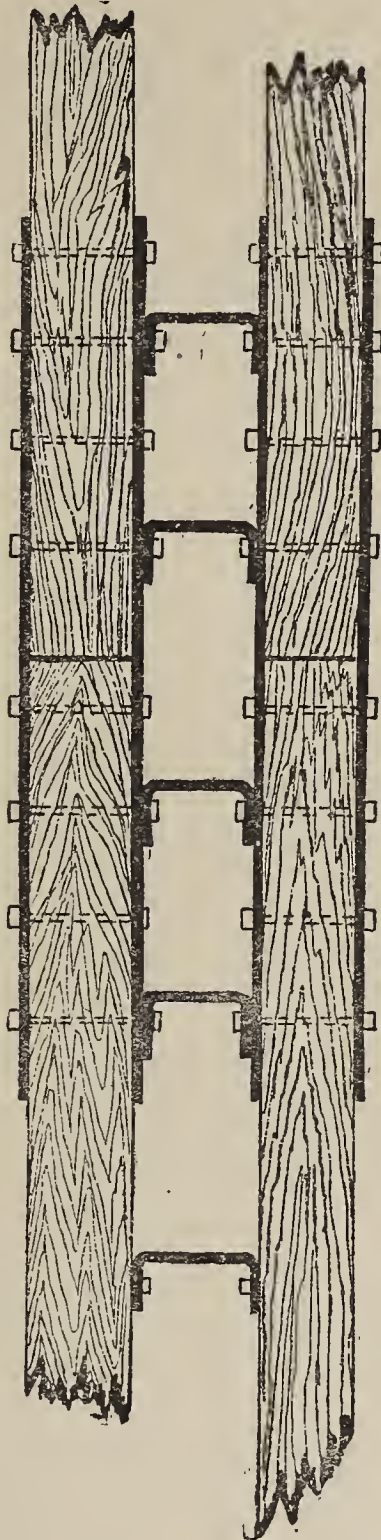
FIGS. 522, 523, and 524.—Pump-rods or spear joints.

placed in a line they will have a greater tendency to split the rod. For rods over 9 in. square four plates will be necessary to give the requisite strength. The plates used on the joints are from 6 to 18 ft. in length, according to the size of the rods and the length of lift;

they may be either uniform in section or somewhat tapered at each end. Additional strength may be obtained by overlapping the ends of the rods, under the jointing plates, by a separate piece fitted in, and holding the ends by means of steel keys (*a*, fig. 525).



FIG. 525.



FIGS. 526 and 527.

When the rods required for a pump have to be very large in section, and would be too large to be handled easily, two rods of smaller section are sometimes employed, braced together by iron stays and bolts, as in fig. 526.

Sometimes the lengths of pump-rods are joined together by plates having projections for the bolts (fig. 527), to obviate the necessity of boring bolt holes in the rods themselves. This plan is said to give additional strength to the rods, and saves labour in boring holes, and the plates are more easily put on. The same object may be attained by using two long wooden straps at the joint and bolting them firmly on the rod.

In both these methods great care should be taken that none of the bolts get slack, as the rods would be liable to drop out from between the plates.

The sizes of rods used, as already mentioned, will depend on the size and speed of pump, and length of lift or head of water. The following sizes are often used in practice, viz. :—

- 8-in. bucket. Lift 30 to 40 fms. Rods 4 in. square, plates $\frac{3}{8}$ in. $\times$ $3\frac{1}{2}$ in. $\times$ 8 ft. Rod bolts $\frac{3}{4}$ in. diameter. 'Dumb' bolts $\frac{1}{2}$ in. diameter. Bolts put in 10 in. centre to centre.
- 10-in. bucket. Rods 5 in. square, with same sizes of plates and bolts as for 8-in. bucket.
- 12-in. bucket. Rods 6 in. square, plates $\frac{1}{2}$ in. $\times$ $\frac{1}{4}$ in. $\times$ 9 ft. Rod bolts $\frac{7}{8}$ in. diameter, dumb bolts $\frac{5}{8}$ in. diameter, and five on each side of joint.
- 16-in. bucket. Rods 8 in. square, plates $\frac{1}{2}$ in. or $\frac{5}{8}$ in. $\times$ 6 in. $\times$ 9 ft. Rod bolts 1 in. diameter, dumb bolts $\frac{3}{4}$ in. diameter.

With a 20-in. bucket the rods would require to be 10 in. square, and four plates would have to be used instead of two, the plates being about $\frac{3}{4}$ in. $\times$ 8 in. $\times$ 12 ft., and the bolts $1\frac{1}{4}$ in. to $1\frac{1}{2}$ in. diameter. Pitch pine sawn into lengths for pump-rods costs about 2s. 6d. per cubic foot. Plates bored and ready for putting on cost 8s. per cwt., and bolts 10s. 6d. per cwt.

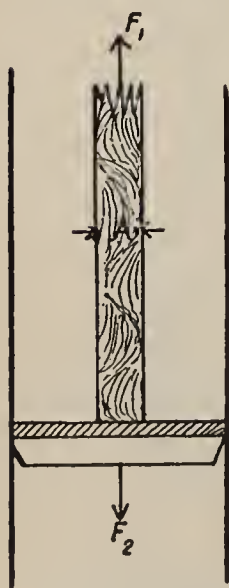


FIG. 528.

In calculating the size of rods and plates to be used for any given size of pump and length of lift, different formulæ are used, giving somewhat different results, but a simple plan is to equate the opposing strains or pressures to which the rods are subjected.

In an ordinary lifting-pump as illustrated (fig. 528), let f_1 be the tensile strength of pitch pine (12,000 lbs. per sq. in.) and A the sectional area of rod in square inches. Then the total *upward* strain on rod will equal $A \times f_1$.

If a is the area of bucket in sq. in. ($D^2 \times .7854$), and p the pressure in lbs. per sq. in., due to head of water (head in ft. $\times .434$), then $a \times p$ will be the total *downward* strain.

Now $A \times f_1$ should equal $a \times p$ or $F_1 = F_2$.

This rule would give the data required for calculating the size of rod necessary to exactly balance the downward strain, but in actual practice a large margin would have to be allowed for safety, this factor depending on the speed and size of pump. The inertia of the water column and of valves would also have to be taken into account.

"In large forcing pumps, with the engine placed on the surface, all the excess of the weight of the rods above that of the column of water, plus the friction, should be balanced, save a little surplus which is left to make the downward stroke begin distinctly. Not infrequently the main line of rods is over-weighted

on purpose, but at the same time balanced, in order to increase the mass set in motion. The object of this is to produce a moderate acceleration only at the commencement of the upward stroke when the steam has its full pressure and the power is much greater than the load, for it is on the rate of this acceleration that the reactions of the force of inertia which are got up in the various parts depend, and these reactions again increase the strain on the entire system."*

It is generally allowed that the useful weight of the main rod ought to exceed by $\frac{1}{10}$ th to $\frac{1}{8}$ th the weight necessary for raising the delivery valves in plunger pumps. For all practical purposes we may take the power necessary to overcome the friction of the water in moving through the pipes and valves, and other contingencies, at about $\frac{1}{5}$ th of the power required to set the water column in motion, *i.e.* the total pressure exerted on the ram. To cover this the factor of safety should be comparatively high, say 30 to 40 for quick-running, intermittent pumps, where the strokes are more frequent and the strain greater than in large slow-moving pumps, where the factor need not exceed 20 to 30.

Example.—What size of (a) rod and (b) plates should be used in a 20-in. lifting set, the head of water being 40 fms., and the factor of safety 30?

(a) $A \times f = a \times p \times M$.

$A \times 12000 = 20^2 \times .7854 \times 240 \times .434 \times 30 = 81.8$ sq. in. (area of rod required), or about 9 inches square.

(b) The size of iron plates may be found in the same way, by substituting 50,000† for 12,000, the factor of safety being taken at 10. If two plates are put on each joint and each plate is 6 in. broad,

$$A \times 50,000 = 20^2 \times .7854 \times 240 \times .434 \times 10;$$

$$5 A = 32.64. \therefore A = 6.52 \text{ sq. in. area; hence}$$

$$\text{Thickness of plate} = \frac{6.52}{2 \times 6} = 0.54 \text{ in. (about } \frac{5}{8} \text{ in.)}$$

The plates would therefore require to be $\frac{5}{8}$ in. $\times$ 6 in. $\times$ 12 ft. with bolts $1\frac{1}{4}$ in. diameter. For plunger pumps the compressive strain (6000 lbs. per sq. in.) of pitch pine would require to be taken instead of tensile strain.

The following empirical formulæ are also used for determining the dimensions of pump-rods, plates, and bolts, where D=diameter of pump in inches.

Bucket Pump-Rods:

$$\text{Sectional area of rods} = \frac{\text{area of bucket}}{3\frac{1}{2}} \quad . \quad . \quad . \quad . \quad (a).$$

$$,, \quad ,, \quad \text{plates} = \frac{\text{area of bucket}}{D+7} \quad . \quad . \quad . \quad . \quad (b).$$

$$,, \quad ,, \quad \text{bolts} = \frac{\text{area of rod}}{10} \quad . \quad . \quad . \quad . \quad (c).$$

Plunger Pump-Rods:

$$\text{Sectional area of rods of lowest lift} = \frac{\text{area of pump}}{1\frac{1}{2}} \quad . \quad . \quad . \quad . \quad (d).$$

$$,, \quad ,, \quad \text{second } ,, = \frac{\text{area of pump}}{2} + \text{area of bottom lift} \quad . \quad (e).$$

$$,, \quad ,, \quad \text{third } ,, = \frac{\text{area of pump}}{3} + \text{area of second lift} \quad . \quad (f).$$

$$,, \quad ,, \quad \text{plates} = \frac{\text{area of rods}}{10} \quad . \quad . \quad . \quad . \quad (g).$$

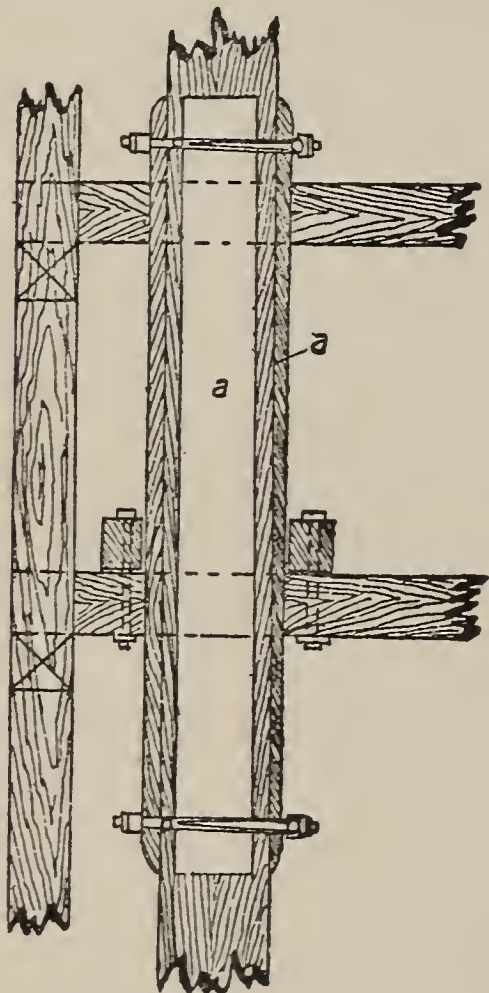
$$,, \quad ,, \quad \text{bolts} = \frac{\text{area of rods}}{10} \quad . \quad . \quad . \quad . \quad (h).$$

These rules are based upon the assumption that the lift is not greater than 60 to 80 yards.

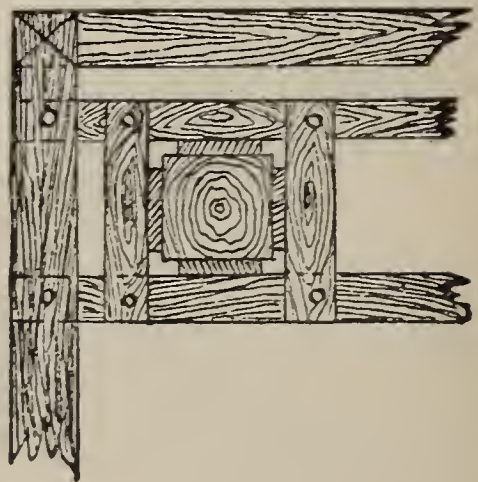
* *Lectures on Mining*, by M. Callon, vol. ii. p. 363.

† Tensile strength of wrought iron = 50,000 lbs. per sq. in.

Guiding the Pump-rods.—When the pump-rods are in operation, they require to be kept from twisting or vibrating in the shaft; this is done by ‘collaring’ or guides. The collaring is composed of pieces of wood nailed or bolted to the cross buntons at certain fixed distances apart. They ought to be placed sufficiently near each other to keep the rods from ‘buckling’ under compressive strains, and they ought to be kept in line or as nearly so as possible. Pieces of hard wood, oak, or beech, called cleats, are fixed to the rods (*aa*, fig. 529) either by nails, counter-sunk bolts, or by glands at the top and bottom.



Elevation.



Plan.

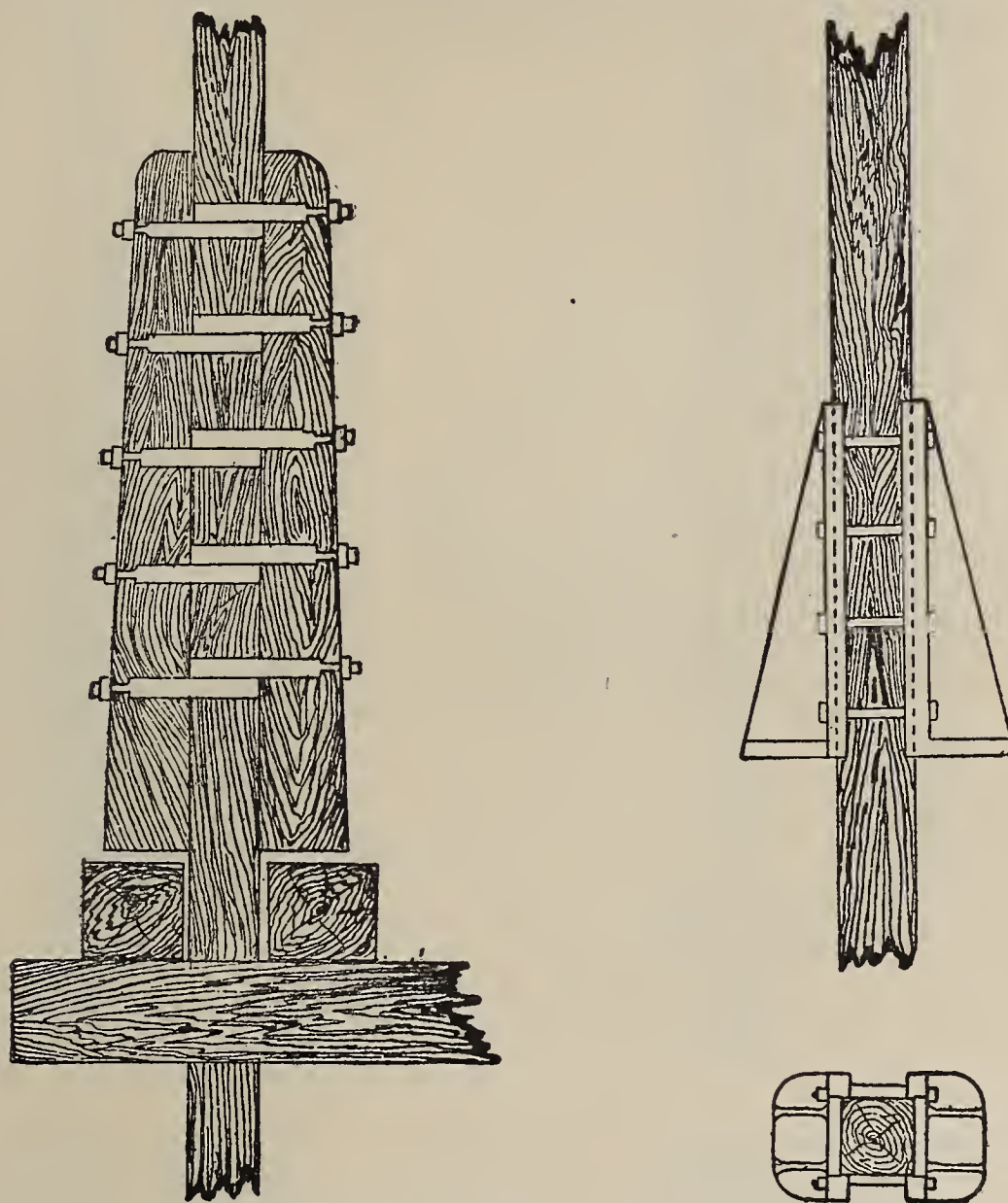
FIGS. 529, 530.—‘Collaring’ for rods.

These cleats are 2 in. to 3 in. thick, tapered at the ends, and made a little longer than the stroke of the pump. They are put on to prevent the rods from wearing too rapidly, and to prevent breakage. Figs. 529, 530 show the methods of collaring.

Sometimes a piece of wood, square in section, is fixed to the rods and works in a ‘shoe’ fixed on the cross buntons or the guides, or the rods may be made of iron and a ‘shoe’ fitted to suit, which is much more substantial.

All these cleats or rubbing pieces should be kept well lubricated, to reduce friction and wear to a minimum. The rods working inside the pipes (called ‘wet’ rods) must also be provided with cleats to prevent the bolts or plates from rubbing against and wearing the pipes.

'Bang' Pieces.—The rods should also be provided with 'bang' pieces, by means of which, in the event of the rods breaking or getting detached from the engine for repairs, they may be caught and rested on strong beams placed across the pit (called 'horse trees'). The 'bang' pieces (figs. 531, 532, 533) may be made of either wood or iron clamped to the rods. It is best to secure these catches to the rods without the bolts passing through the latter, because then



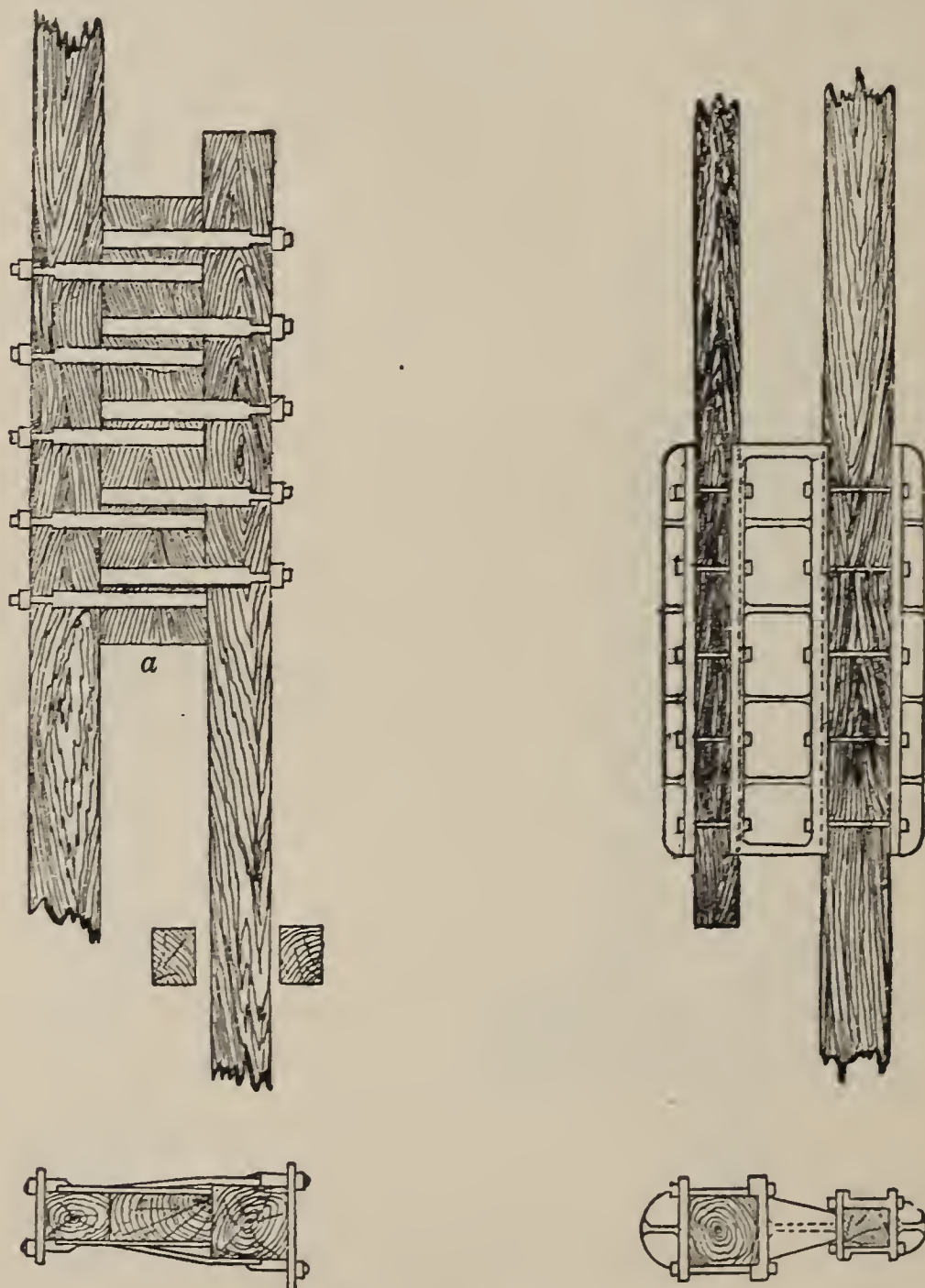
FIGS. 531, 532, 533.—'Bang' pieces.

they can slip a little, and so lessen the shock if they happen to fall away. With the same object a 'cushion' is sometimes used, made of thin boards placed immediately above the 'horse trees.' The cross beams or 'horse trees' require to be sufficiently strong to withstand the shock of the column of rods falling on them suddenly.

There ought to be at least two of these supports, one immediately below the 'bell cranks,' and the other at the bottom of the rods.

Offsets or Aprons.—The usual method of operating two sets of

pumps by one column of rods is generally done by means of offsets called aprons. The usual way is to insert a square block of wood, called an 'apron,' between the two sets of rods, so as to carry the one set (those of the lower lift) clear of the working barrel of the other set (figs. 534, 535). Cast-iron aprons are also used for making



FIGS. 534, 535.—Offset or 'apron.'

FIGS. 536, 537.—Cast-iron 'apron.'

offsets (figs. 536, 537), and are better than wood, as they are less liable to shrink, and the clamps less likely to get loose. They are, however, more expensive and more difficult to fix.

Guides or collaring should be placed as near where the offset is made as possible, both above and below.

The offset may also be effected by using a 'cross-head,' which is to be preferred to the ordinary apron, as the set of rods operated

below the cross-head is in direct line with the rods working above, so tending to reduce friction and strains (see fig. 538).

Balancing the Rods.—In most pumping arrangements where wooden rods are used some balancing arrangements are required, as the rods are nearly always heavier than the column of water they have to raise.

Callon says: "This excess of weight requires the expenditure of a certain amount of motive power to raise it, and as soon as the rod in its descent has opened the clacks it becomes useless and even

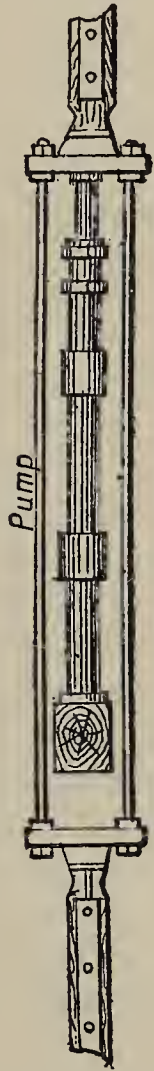


FIG. 538.—Cross-head.

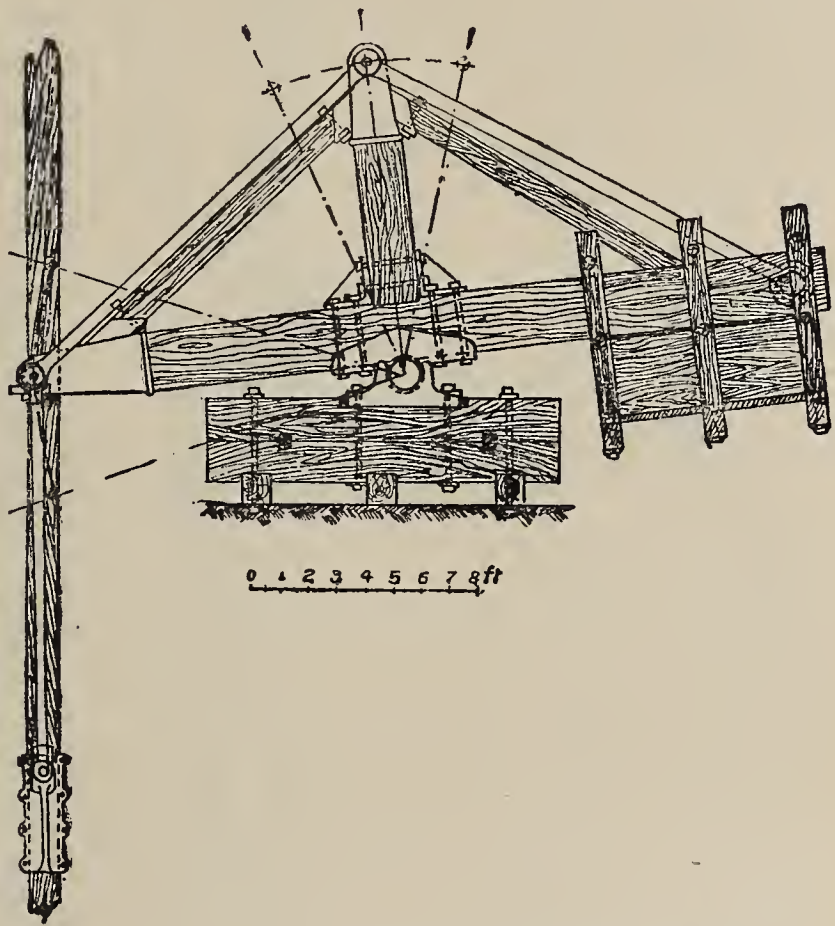


FIG. 539.—Balance beam.

injurious. In fact, it becomes necessary to counterbalance it, so that the rod may not acquire an accelerated velocity." *

There are different methods of applying a counterbalance, but the commonest one is to use a balance on the 'bell-crank' at the surface, or an auxiliary bell-crank in some part of the shaft.

A double bell-crank is sometimes used for the counterbalance, and at one end a heavy weight is placed, generally a large box filled with scrap metal, which can be adjusted whenever required (fig. 539).

Hydraulic or loaded pistons are also used for balancing the rods,

* *Lectures on Mining*, M. Callon, vol. ii. pp. 347-8.

and compressed air, where used, can also be applied for this purpose, and is to be preferred to the hydraulic piston.

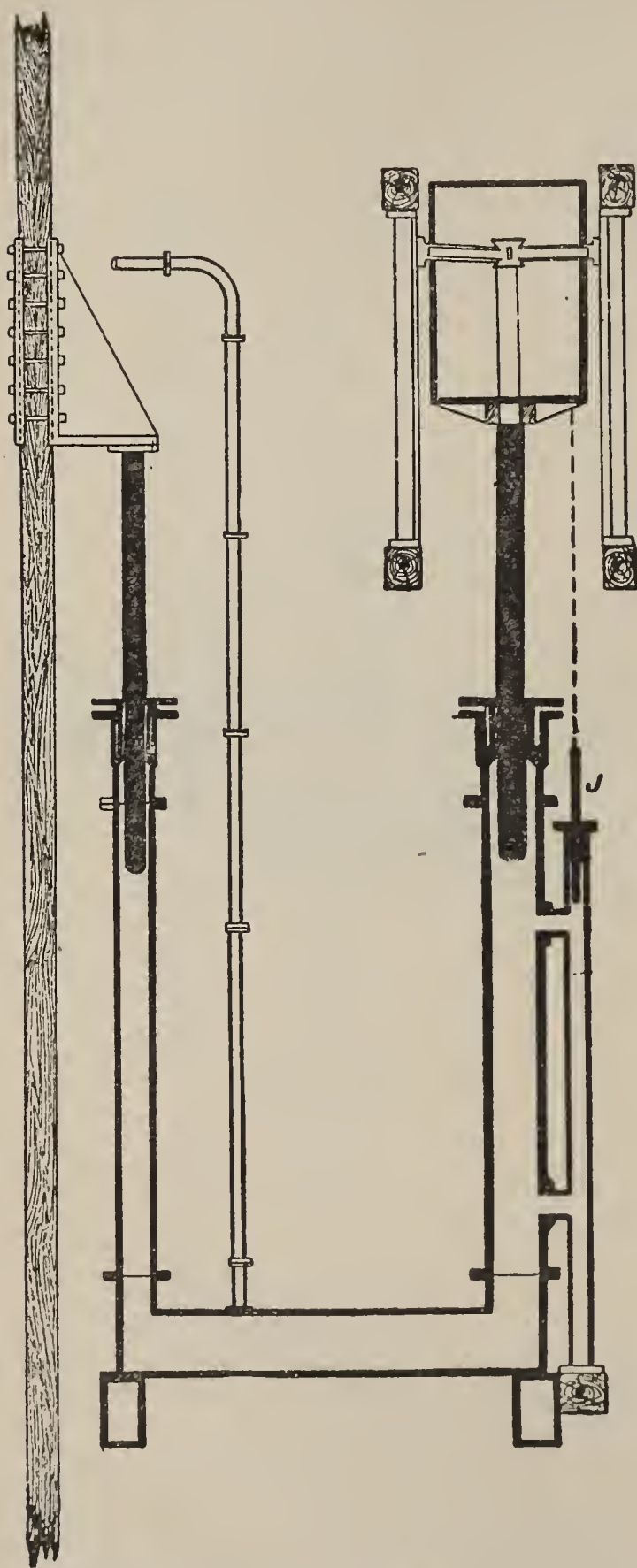


FIG. 540. — West and Darlington's balance.

In West and Darlington's hydraulic balance an auxiliary piston is used, worked off the main rods (fig. 540). The loss of water is made up by a pipe which communicates with a cistern placed in the shaft. When this cannot be done a small plunger J draws up and forces in the necessary supply. The same arrangement can be adopted with compressed air.*

Bucket Pumps. — These pumps are largely used for raising water in shafts. They are similar to ordinary well pumps. A working barrel *a* (fig. 542), smoothly bored out, works the bucket *b*. Immediately below the working barrel is the clack piece *c*, which contains the suction or receiving clack *d*, and dipping into the water is the wind-bore or snore-pipe *e*, which is a pipe closed at one end with a number of holes bored in it for the admission of the water.

Action of Pump. — Suppose the bucket to be at the end of the down stroke, and to be now raised. This upward motion tends to produce a vacuum below it, and the air contained in the wind-bore opens the suction valve and rushes into the working barrel. The elastic

force of this air being thus diminished, the atmospheric pressure

* For fuller description of this balancing arrangement, see *Ore and Stone Mining*, Sir C. Le Neve Foster, sixth edition, p. 476.

will cause the water to rise in the pipes until the pressure of the column, increased by that of the air inside, exactly counterbalances the pressure of the external atmosphere. At the next stroke of the bucket a fresh quantity of air will escape, and the water will rise higher in the pipes, this process being repeated until the water reaches the bottom of the bucket, provided the distance is not greater than 34 ft. In practice this height is, however, never attained, 20 to 25 ft. being the practical working distance between the bucket and the surface of the water. The pipes being now filled with water up to the bottom of the bucket, and the pump-rod falling at the down stroke, the suction valve will be closed and the valves on the top of the bucket will be open, allowing the water, which on the up stroke will be carried the full length of the stroke, to rush through. This process is repeated until the water reaches the surface. It will be seen that nearly the whole weight of water is raised on the up stroke.

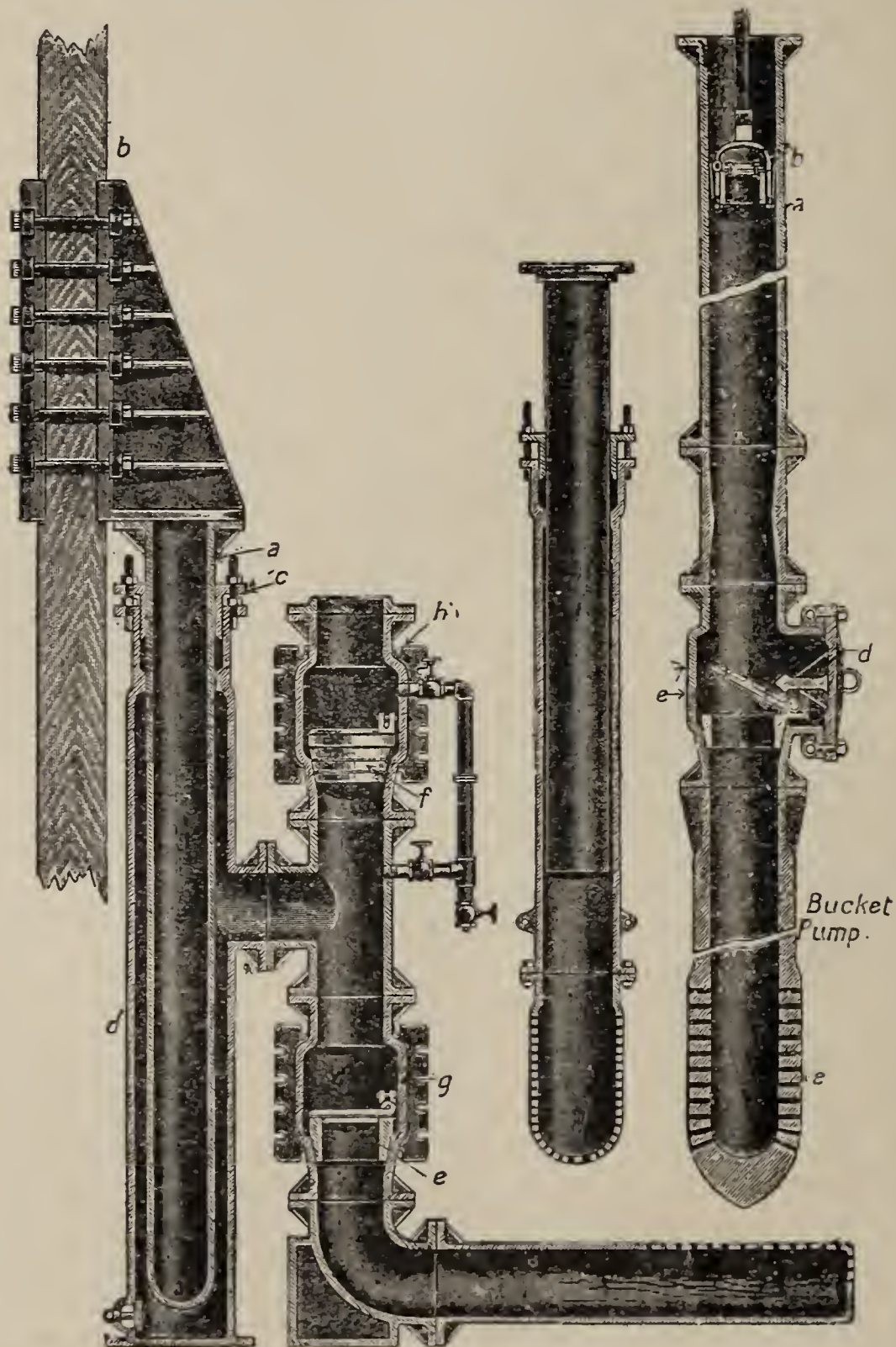
Plunger Pump.—The construction of the plunger pump is somewhat different to that of the bucket pump. In this pump there is a long solid piston or plunger *a* (fig. 541) connected to the pump-rods *b*, working through a stuffing-box *c*, in the barrel or ram casing *d*. Instead of a single valve, as in the bucket pump, there are two, a suction valve *e* and a delivery valve *f*, situated in the two clack chambers *g* and *h*.

Action of Plunger Pump.—The principle of this pump is exactly like that in the bucket pump, but it is worked somewhat differently. On the up-stroke of the plunger the water forces the suction clack open and fills the space between it and the bottom of the plunger; on the down stroke of the plunger the suction valve is closed and the water is forced up through the delivery clack the full length of the stroke, the same process being repeated at each stroke until the water is delivered at the surface. Thus the whole of the work done is practically carried out in the down stroke.

In a bucket pump the work is more unequally divided than in a plunger lift; but with a bucket pump the rods last much longer unless the water is acid and destroys the iron connections at the joints. The latter has also the advantage that in case of a sudden inflow of water in the shaft, the bucket may be drawn through the pipes and changed at the top of the lift, if, as is usual in fitting up bucket lifts, the pipes are a little larger than the working barrel. It is, therefore, generally the rule where there are a series of lifts in the shaft, to make the bottom one a bucket lift. It is usual to arrange bucket lifts with two bell-cranks, so that when one of the lifts is making an up stroke the other is making a down stroke. In this way they are more or less balanced, and the work done by the engine is better distributed.

Compared with a bucket lift the advantages of plunger pumps may be thus stated:—(1) The rods or spears being working outside

the pipes, can be more easily examined and got at for repairs when needed. (2) Any leakage at the plunger is more easily seen and repaired. (3) The delivery pipes may be smaller. It is usual to determine the size of the delivery pipes by the velocity at which the



FIGS. 541, 542.—Plunger and bucket pumps.

water can travel without excessive friction. (4) The power is more evenly distributed during the up and down strokes.

Length of Lifts.—The length of lift depends somewhat on the point at which the water is coming into the shaft, and the suitable-

ness of the strata for a seat for the column of pipes and also for a lodgment for the water. For bucket pumps it is found that the most suitable length of lift is from 30 to 35 fms., and for plunger pumps 40 to 80 or 90 fms., according to the type and size of engine used to operate the pumps. With engines of the Cornish type, the lift is not often more than 45 fms., while with direct-acting engines of Bull type the lift may be 70 or 75 fms. With such long lifts the speed must be slow and the stroke long, so as to avoid too severe shocks.

With the power engine placed at the bottom of the shaft the water may be forced up any height of lift between 100 and 1500 ft.; but with very long lifts the joints at clack chambers and between the pipes will be very difficult to keep, and may give much trouble.

Speed of Pumps.—This will vary much, according to the size and type of pump used. For either plunger or bucket pumps working with rods, a good average speed is 60 to 80 ft. per minute, which may be increased a little with pumps below 12 in. in diameter. For pumps working direct without rods, the speed may be 100 to 150 ft. per minute, according to the size of the engine. In all pumps great care ought to be taken to avoid sudden shocks.

Length of Stroke.—This will also be determined by the size and type of pump used, and may vary from $3\frac{1}{2}$ ft. for 8 in. or 10 in. diameter pumps, and small engines up to 12 or 13 ft. for 20 in. to 30 in. diameter pumps and large direct-acting engines.

Quantity of Water delivered by a Pump.—The quantity of water delivered by a pump can be calculated from the formula,

$$G = \frac{D^2 \times .7854 \times L \times N \times 6.25}{144} \quad \text{or} \quad G = D^2 \times .034 \times L \times N \left\{ \frac{.7854 \times 6.25}{144} = .034 \right\}$$

Where D = diameter of pump in inches.

L = length of stroke in feet.

N = number or effective strokes per minute.

G = Gallons delivered per minute.

6.25 = number of gallons in 1 cub. ft. of water.

These formulæ will give the theoretical quantity delivered, 5 per cent. to 12 per cent. of which should be deducted for slip at the valves and bucket or plunger.

Example.—How many gallons could a pump 15 in. diameter, with a 6 ft. stroke, and making eight effective strokes per minute, deliver per minute, allowing 10 per cent. for slip?

$$\begin{aligned} G &= D^2 \times .034 \times L \times N \\ &= 15^2 \times .034 \times 6 \times 8 \\ &= 369.2 \text{ gallons per minute without slip.} \end{aligned}$$

$$\therefore G = \frac{369.2 \times 90}{100} = 332.28 \text{ gallons per minute allowing for 10 per cent. slip.}$$

The theoretical quantity delivered per foot stroke is approximately found by the expression $G = \frac{30}{D^2}$.

The diameter of pump required for a certain quantity of water may also be derived from the above formula,

$$D^2 = \frac{G}{.034 \times L \times N} \quad \text{or} \quad D = \sqrt{\frac{G}{.034 \times L \times N}}.$$

Example.—What size of pump would be required to deal with 10,000 gallons of water per hour? Engine to work 124 hours per week, and 10 per cent. to be allowed for leakage.

10,000 gallons per hour = 166.6 gallons per minute.

124 hours per week = 17.7 hours per day.

$\therefore \frac{166.6 \times 24}{17.7} + 10 \text{ per cent.} = 248.48 \text{ gallons per minute.}$ If the effective speed

of pump is 30 ft. per minute, then $D^2 = \frac{248.48}{.034 \times 30} = \sqrt{243.6} = 15.6 \text{ in. diameter.}$

The size of pump required would therefore be 15.6 in. diameter, say with 5 ft. stroke, going at six strokes per minute.

Plant Required for Working Pumps.—The engines used for operating pumps may be worked with steam, compressed air, hydraulic pressure, or electricity.

For operating pumps in shafts, engines worked by steam are almost universally used, especially when they are situated on the surface and worked by rods. Steam engines for working pumps by means of rods may be divided into three classes: rotative, non-rotative, geared.

The non-rotative types may be either direct-acting or indirect-acting. Direct-acting engines are those in which the piston-rod is in line and connected to the pump-rods. The Bull engine is the most familiar type of this class of pumping engine.

Sometimes a direct-acting engine is defined as a “machine for raising water, so constructed that the pump is worked by the motive-power cylinder, without the intervention of beams, connecting rods, cranks, or fly-wheels.”

Direct-acting engines are not so largely used now as formerly, unless for large quantities of water, or deep shafts, because the cylinder usually obstructs the mouth of the shaft. This is often inconvenient, unless there is plenty of room, or a shaft is wholly set apart for pumping. This type of engine also requires very heavy and expensive foundations, and it is also wasteful of power, owing to the uncertainty of the point at which the stroke is completed, and the necessity for thus working with a large ‘clearance’ space in the cylinder.

Geared pumping-engines are very largely used for actuating pumps with rods, and are receiving more attention from engineers than formerly. The great advantage of this class of engine is that the speed can be easily altered to suit any conditions and that the work per stroke can be varied much more easily than in other types.

It must not be forgotten, too, that steam, being a much more elastic body than water, can be moved more rapidly without danger, and hence in a geared engine the steam may be moving very rapidly,

while the water may be moving very slowly. This class of engine can likewise be erected more cheaply and disposed of more readily than large direct-acting engines, if not required for, or readily adapted to, other work than pumping.

Special Types of Pumping Engine:—*Bull Engine.*—The Bull engine may be termed a Cornish engine, with a beam working under the cylinder instead of over it (fig. 543). The cylinder is placed directly over the shaft in a line with the pump-rods, these being a continuation of the piston-rod, and acting direct.

The great objection to this class of engine is, that it blocks up the shaft and requires large and expensive foundations. On the other hand, the first cost is less than for an ordinary Cornish engine.

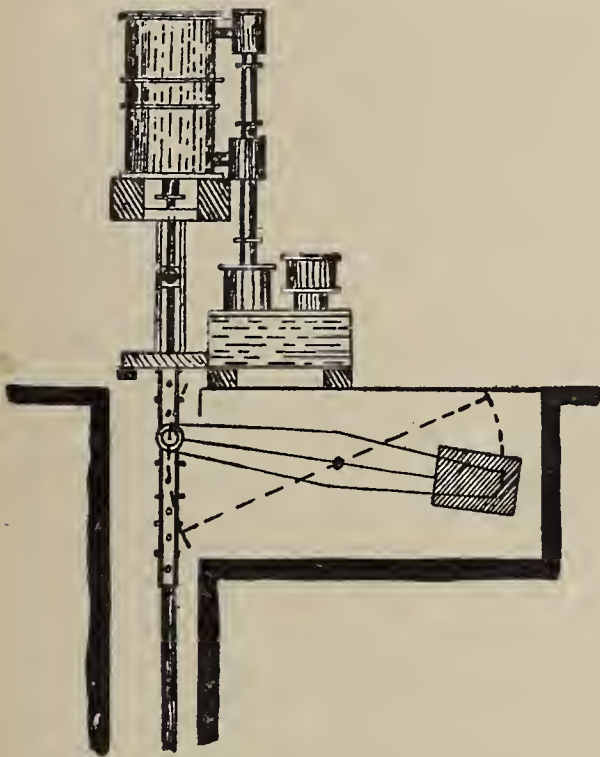


FIG. 543.—Bull engine.

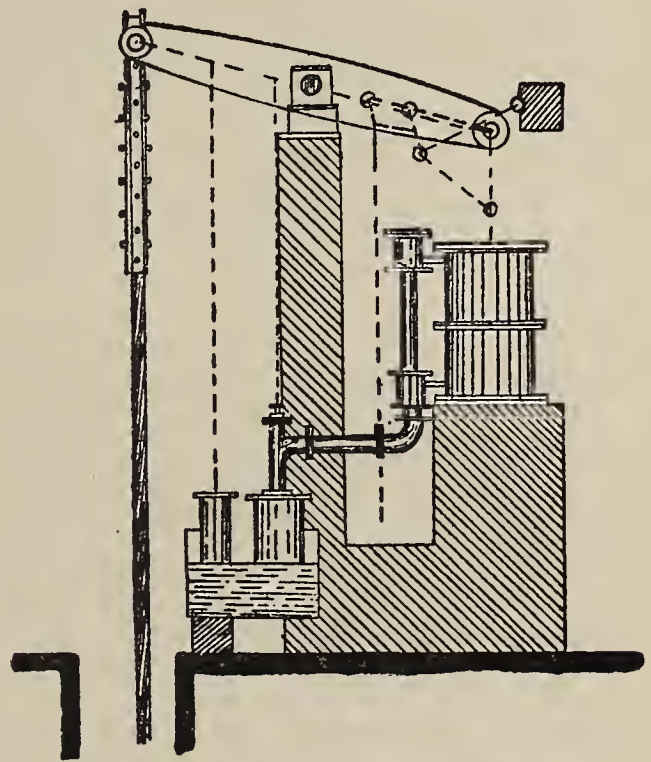


FIG. 544.—Cornish engine.

A modification of the Bull engine is sometimes used, in which, to avoid the cylinder being placed directly over the shaft, it is placed some distance back and bell-cranks introduced between the piston and the pump-rods. In this way the space in the shaft is not encroached upon to the same extent, and less expensive foundations are necessary.

Sometimes these engines are worked on the compound condensing principle, with high- and low-pressure cylinders. The steam is regulated in the same way as in the Davey engine, by an auxiliary piston and tappet valves, and is not permitted to act by expansion alone, but by *wire-drawing* to a considerable extent.

Cornish Engine.—This is one of the oldest of pumping engines, the type having been first devised by Watt, and such engines are still used to a considerable extent for pumping in mines (fig. 544). The Cornish engine is generally worked as a single acting, high-pressure

condensing engine. It consists of a vertical cylinder, having the piston-rod connected to one end of a large 'walking' beam, and the pump-rods to the other end. The beam usually works on a fulcrum near its centre, thus causing the stroke of the engine to be somewhat longer than that of the pumps, and the flow of water in the pipes is therefore less than the speed of the steam piston, in order to lessen the shock of the water in the pipes on the sudden closing of the valves at the reversal of the stroke. In the working of the engine steam is admitted at the top of the cylinder, this causing the piston to make its down stroke. The steam is then cut off and an equilibrium valve opened by means of a tappet-rod, which allows the steam to pass to the under side of the piston, thus equalising the pressure and causing a pause in the movement. The weight of the rods alone is sufficient to cause the down stroke and force the water up. The steam is then exhausted into the condenser, where a vacuum is formed by the air-pump. This process is repeated at each stroke. The engines are best suited for slow speeds and long strokes. They are very expensive at first, but once set up they give little trouble, require little repair, and give a high efficiency.

Barclay Engine.—This is another well-known type of the beam pumping engine, which is much used for dealing with large quantities of water. In construction it is a modification of the Cornish engine. It consists of a single steam cylinder placed vertically, the piston-rod being connected to the overhead beam. To one end of the beam the pump-rods are connected, while the other end works on a fulcrum which takes the form of a rocking pillar. Near to the centre of the beam a rod connection is attached for working the air-pump and condenser hot-well. This type of engine, like the Cornish, is best suited for large quantities of water and where a long stroke can be adopted. Owing to the arrangement of the beam the stroke of the engine is less than that of the pumps: for instance, the piston stroke may be 8 ft., while the pump stroke may be 10 ft. Engines of this type are made with cylinders 50 to 100 in. diameter and 8 to 12 ft. stroke, moving at two to four strokes per minute.

Another type of engine much used for pumping in the Fifeshire coal-field, where large quantities of water are met with, is one in which a pair of cylinders, high- and low-pressure, working compound and condensing, are placed vertically directly over the shaft, the piston-rods of the cylinders being connected direct to the ends of the bell-cranks which operate the pump-rods (fig. 545). These engines are well suited for raising large quantities of water, 1000 to 1500 gallons per minute, from depths of 150 to 250 fms. A plant of this type, with cylinders 58 and 100 in. respectively, is at work raising 1200 gallons per minute from a depth of 350 fms. The objections to this type of pumping engine, like the Bull engine,

are that the cylinders are placed directly over the shaft, and they take up a great deal of space, necessitating the sinking of a very

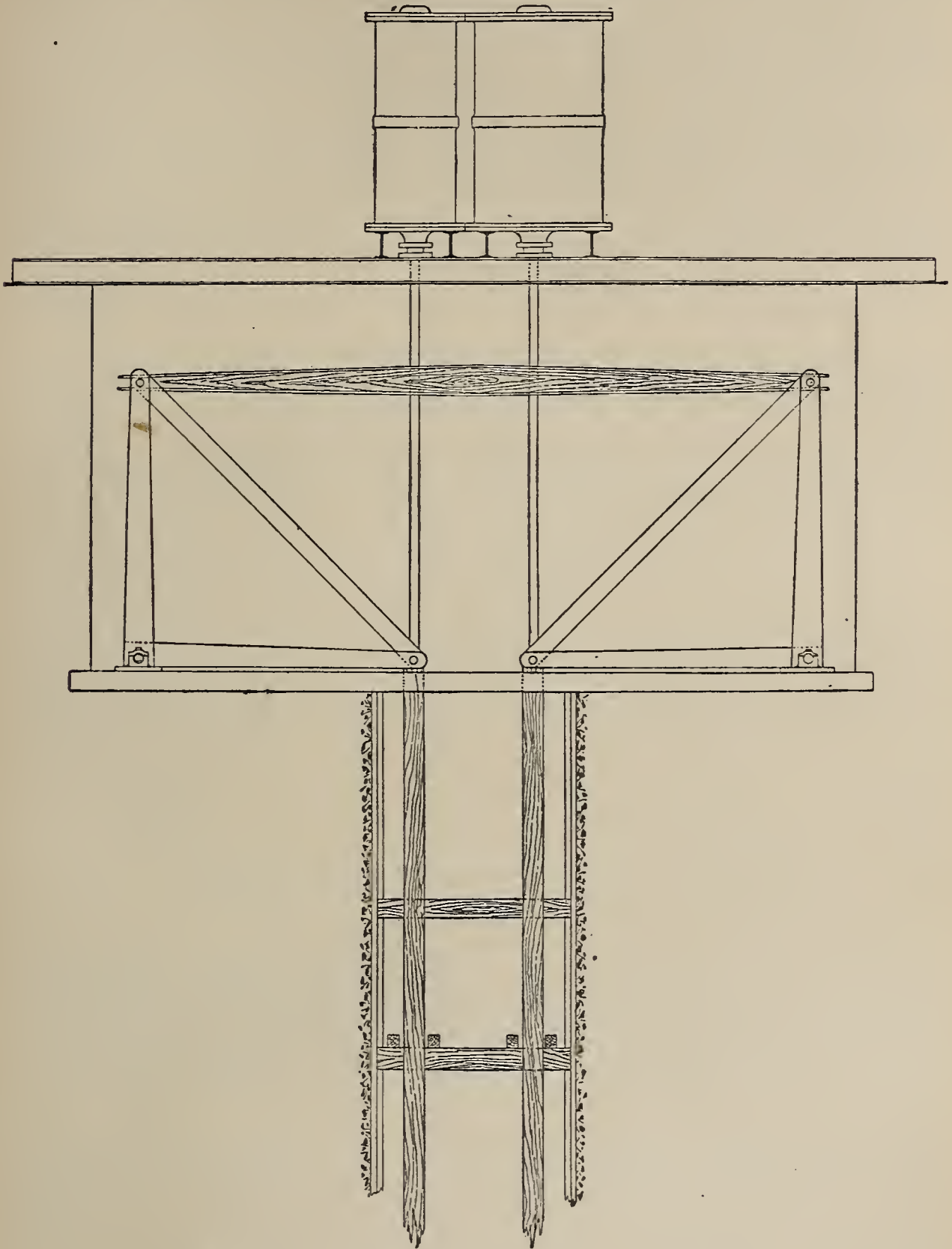


FIG. 545.—Compound vertical pumping engine.

large shaft, or devoting a shaft entirely to pumping. They are best suited for large rectangular shafts where the necessary space can be set apart for them at one end of the shaft.

Wherever large pumping engines of the types just described are employed it is almost imperative that a duplicate plant be installed to deal with the water in the event of any serious breakdown. This duplicate plant may consist of a direct-acting pumping engine, such as the Riedler pump, placed underground at the bottom of the shaft to force the water direct to the surface.

Horizontal Pumping Engines.—A typical pumping plant of this description is at work at Boghead Colliery, Bathgate, N.B. The plant is designed to deal with 500 gallons per minute from a depth of 160 fms. It consists of high- and low-pressure cylinders placed tandem (fig. 546), 38 in. and 76 in. diameter respectively, each

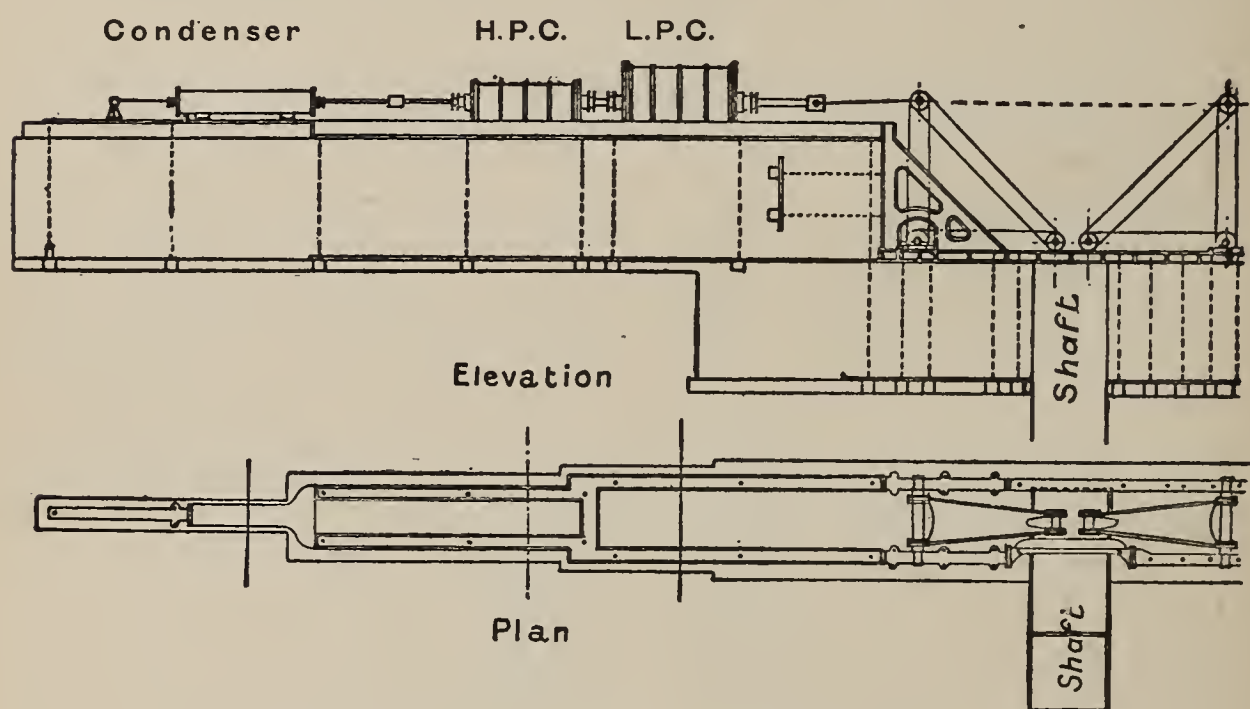


FIG. 546.—Direct-acting horizontal pumping engine.

cylinder having a stroke of 9 ft. The piston-rod of the low-pressure cylinder is continued forward and connected to the bell-cranks, while the piston-rod of the high-pressure cylinder is continued backwards to the condenser and air-pump. In this case no fly-wheel is used.

The pumps are worked by a double set of rods attached to the bell-cranks at the surface, each set of rods working two plunger and one bucket lift at three different stages in the shaft. At the top and mid lifts the offsets are made by means of cross-heads bolted to the rods and connected to each other by iron side rods far enough apart to clear the pumps. The details of the pumps are as follows:—

Two bucket lifts at pit-bottom, 160 fms. from surface, each set $14\frac{1}{2}$ in. diameter, lifting 30 fms.; two plunger lifts at 130 fms., each $19\frac{1}{2}$ in. diameter, lifting 56 fms.; two plunger lifts, each 20 in.

diameter, at 74 fms., lifting to the surface. The sizes of pump-rods for the various lifts are:—bottom bucket lifts, rods 6 in. square; second lift, rods 12 in. square; top lift, rods 14 in. square; the iron strapping plates at the joints for these two latter are $11 \times \frac{3}{4}$ in. $\times$ 17 ft. The available steam pressure is 90 to 100 lbs. per square inch, plus $12\frac{1}{2}$ lbs. vacuum from condenser.

A good type of pumping engine is one in which high- and low-pressure cylinders are used. The connecting rods to the bell-cranks are in a direct line and form a continuation of the piston-rod. A back piston-rod works a fly-wheel which imparts steady motion to the engine. This class of engine has much to recommend it for pumping purposes, as it works very smoothly, and can be easily regulated to any required speed.

Geared pumping-engines are very largely used for pumping water at collieries, and, as has already been pointed out, are peculiarly fitted for such work, and have many advantages to recommend them. They may be constructed either with one, two, or four cylinders, and may be condensing and compound. The gearing can be adapted to suit any speed and any size of pump. This easy adaptability to work under varying conditions makes these engines very suitable for mining, and gives them many advantages over the larger types of vertical engines. An engine constructed in this way gives a high efficiency and a small consumption of steam, and works very smoothly at almost any reasonable speed. Fig. 547 shows the arrangement of a geared engine with low- and high-pressure cylinders, as used at the Priory Colliery, Blantyre, where the water has to be raised from a depth of over 200 fms. The high-pressure cylinder is 20 in. diameter, and the low-pressure cylinder 34 in. diameter, with 5 ft. stroke geared to about $2\frac{1}{2}$ to 1. The cylinders are steam-jacketed, and work with a very small consumption of steam. They can be regulated to any speed, as low as $1\frac{1}{2}$ strokes being attainable.

At Holm Colliery, Kilmarnock, the pumping engine is constructed with quadruple cylinders, two high-pressure and two low-pressure ones being supplied. The high-pressure cylinders are $16\frac{7}{8}$ in. and $15\frac{7}{8}$ in. diameter respectively, while the low-pressure cylinders are $24\frac{1}{4}$ in. and 26 in. diameter respectively, the stroke being $3\frac{1}{2}$ ft. and the gearing $5\frac{1}{2}$ to 1. The water is raised the first 68 fms. by two rams 15 in. diameter and 5 ft. stroke, the lower lift of 13 fms. being two buckets 10 in. diameter and 5 ft. stroke. The speed is five strokes per minute, and the water raised 420 gallons per minute.

Direct-acting Steam Pumps Underground.—Direct-acting pumping engines placed underground are very largely superseding surface engines with heavy pump-rods working in the shaft. The greater simplicity of construction and cheapness combined, make them desirable for pumping purposes, and although, as a rule, they require

much fuel, this is compensated for in other ways. These engines are nearly always double-acting, and are directly coupled to the pump. They may also be made to work geared, but this arrangement is not so much employed underground as the direct-acting.

Double-acting pumps have generally a comparatively short stroke, with very much smaller masses to move them than pumps actuated

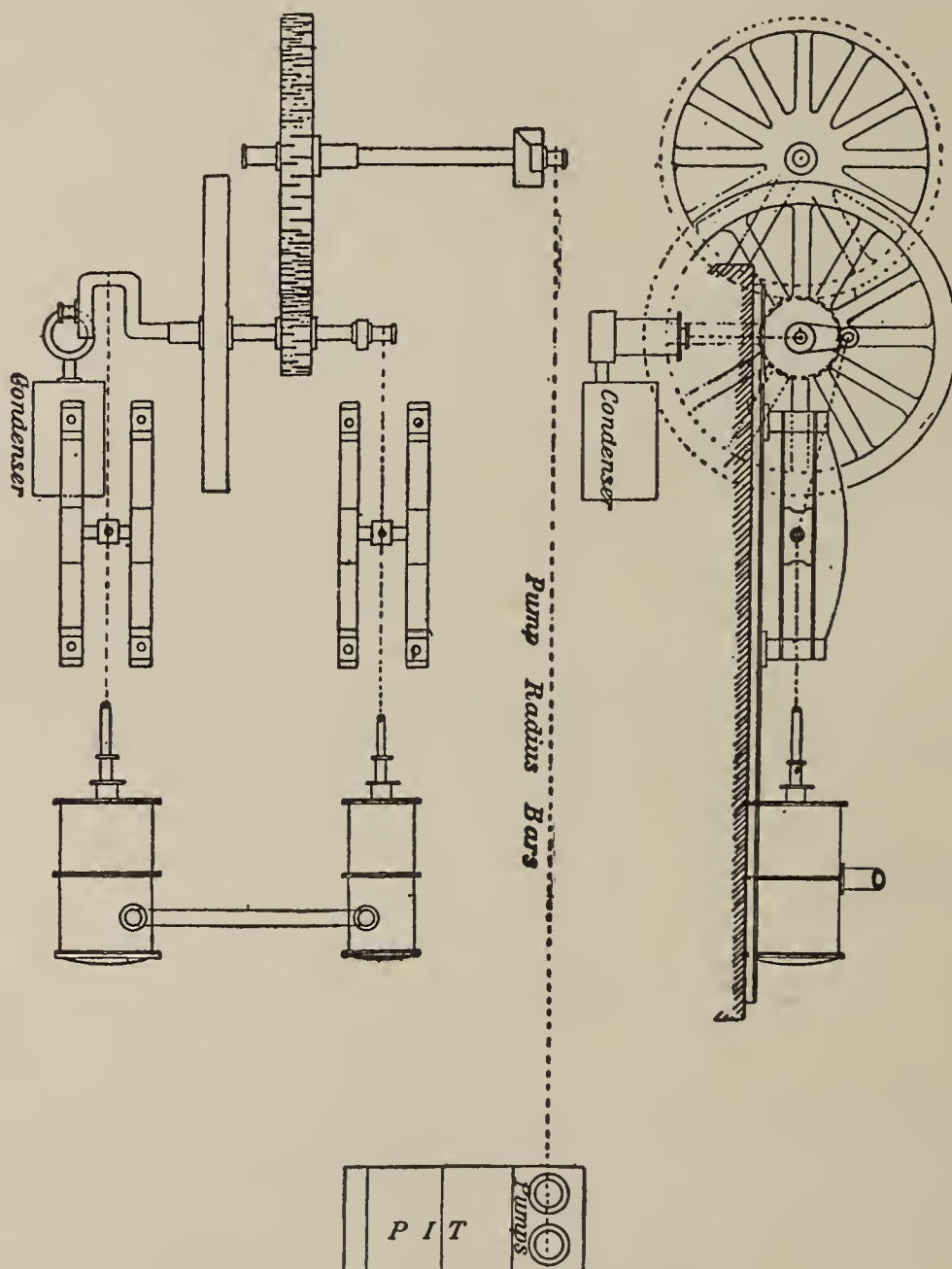


FIG. 547.—Geared pumping engine.

from the surface with rods, and they can therefore be driven at much greater speeds.

The higher speeds at which they can be driven, combined with their smaller size, are the main features which commend this class of pumps for use underground. Direct-driven steam pumps are now used for heads of water varying from 200 to 1800 ft.

Where engines are placed underground, a large lodgment, sufficient for, at least, a week's water, ought to be provided, to give time for repairs on the engine should they be necessary. With engines placed

underground, the following are some of the main advantages to be gained :—

The necessity for heavy pump-rods, which are expensive at first cost and for upkeep, is done away with.

As no pump-rods are required, less shaft space is occupied.

The engines can be worked at high speeds, as there are no heavy rods to put in motion at the commencement of each stroke, and the work is evenly distributed.

The flow of water being continuous, smaller pipes can be used in the shaft.

On the other hand, there are the following disadvantages to be kept in view :—

Steam must be taken underground or boilers fitted up in the workings, both of which methods are more or less objectionable; especially the latter.

The engine is liable to be drowned by a sudden inflow of water (the same, however, may be said of any other type of pump operated from the surface).

Loss of steam by condensation and difficulty of dealing with the exhaust steam.

Increase in temperature of air if placed near the intake, and consequently injurious effects from the moisture to the roof and timber of roadways.

The use of steam in confined places is attended with danger.

Piston Pumps.—This type is largely used for direct-acting pumps working underground without rods, and is invariably double-acting.

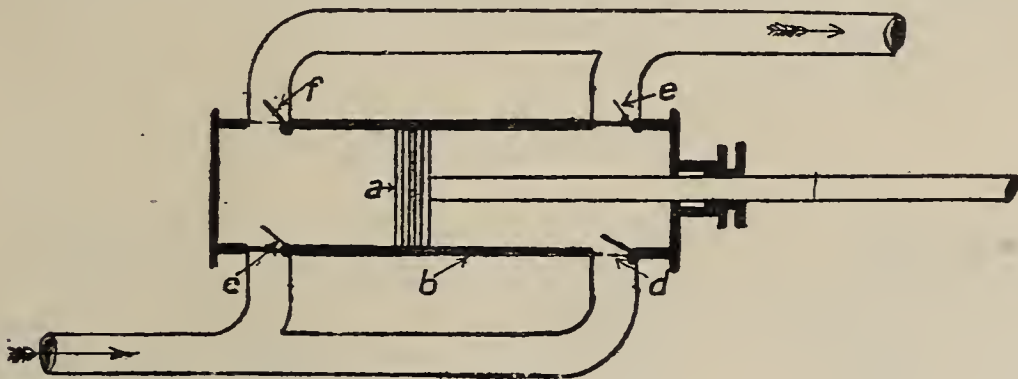


FIG. 548.—Piston pump.

The working of the pump will be understood from fig. 548. A piston *a* works in the barrel *b*, to which are four openings fitted with valves, two receiving and two discharge. The action of the pump is as follows :—When the piston moves outward, the clacks or valves *c* and *e* will, supposing the pump is full of water, be open, water being received through the valve *c* and discharged through the valve *e*. On the return or inward stroke the valve *d* will be open, receiving water, and the valve *f* open discharging, so that there is a continual flow of water during both strokes.

There are so many steam pumps in the market, all equally suitable for underground work, that it is difficult to select any particular

make for detailed description. The general principle of a double-acting steam pump will be seen from fig. 549, in which the steam cylinder is connected direct to, and is in line with, the double-acting piston pump, provided with an air vessel on the delivery column.

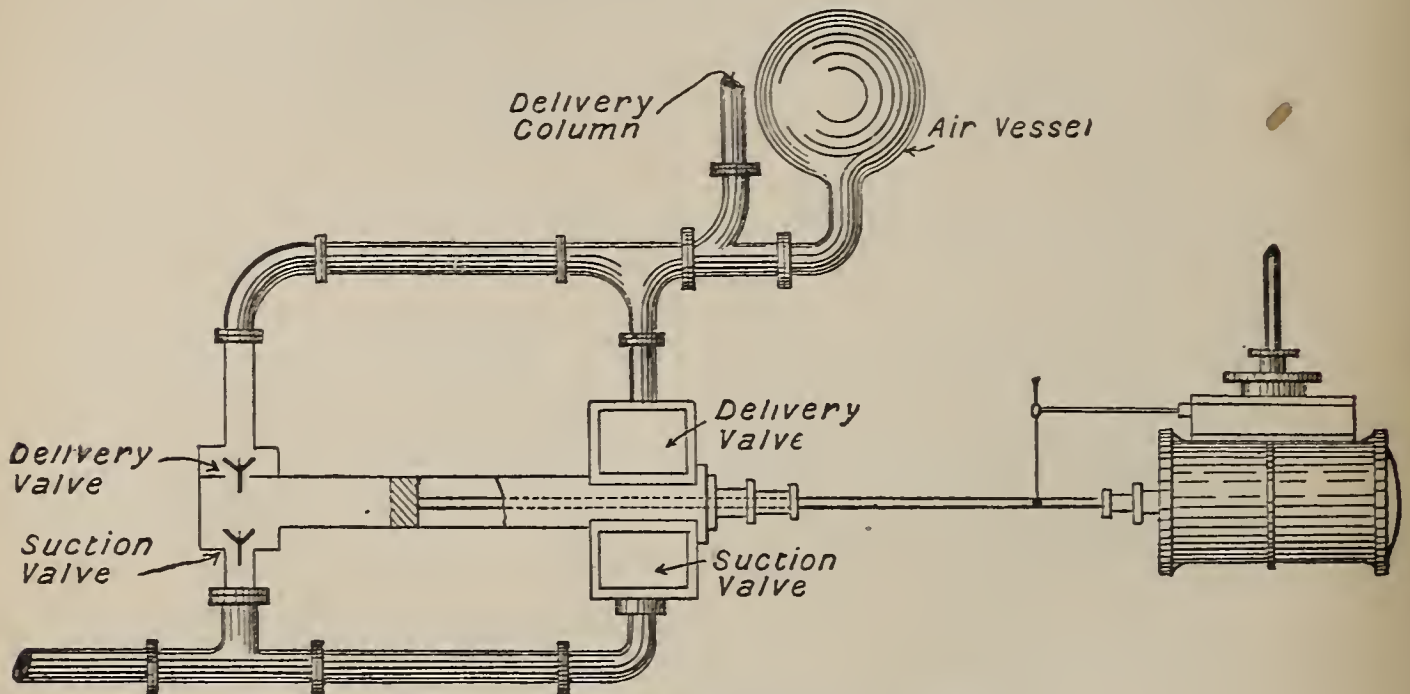


FIG. 549.—Double-acting steam pump.

Worthington Pump.—This is a well-known type of direct-acting steam pump for underground work. It consists of a steam cylinder with piston (A, fig. 550), the piston-rod of which is continued and

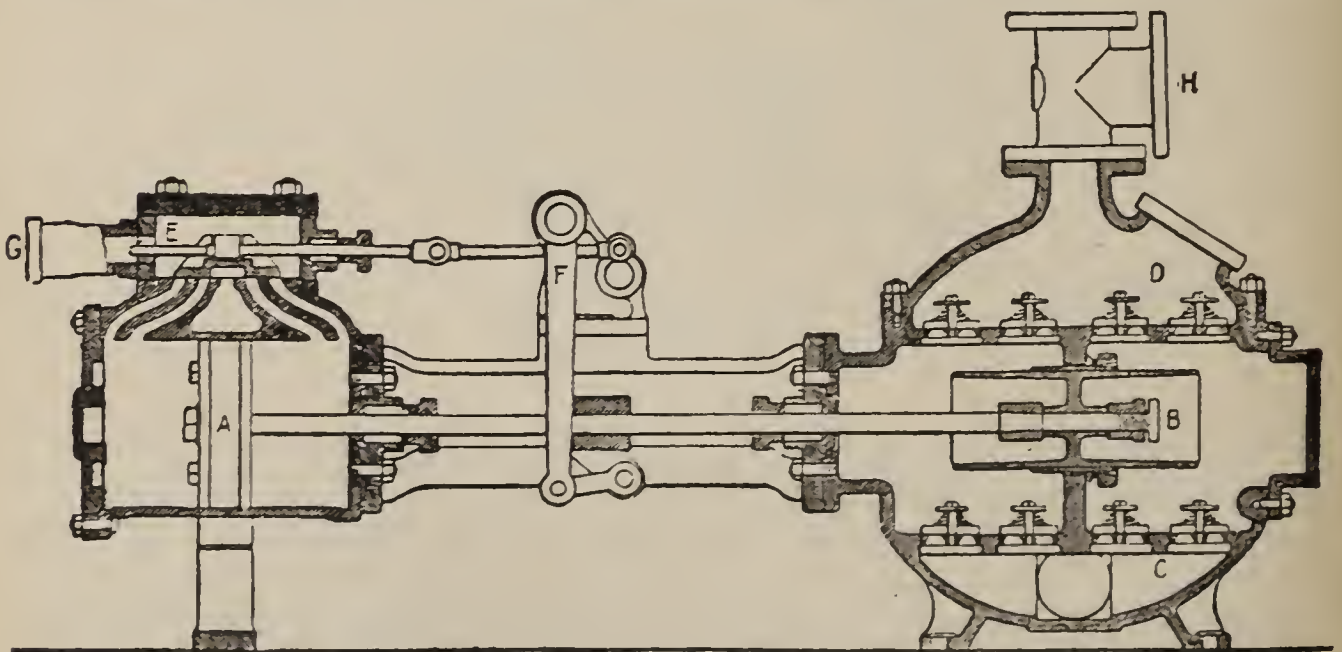


FIG. 550.—Worthington steam pump.

directly connected to a double-acting plunger B, in the water cylinder or pump. The steam is admitted to the cylinder by an ordinary slide valve E, working upon a flat face over the port-holes. The valve receives its motion from a vibrating arm F, which swings through

the whole length of the stroke, with long and easy leverage. As the moving parts are always in contact, the blow inseparable from the tappet system is avoided. The double-acting plunger B works through a deep metallic packing ring, bored to an accurate fit. Both the ring and the plunger can be quickly taken out, and either refitted or exchanged for new ones; and if it is desired at any time to change the proportions between the steam pistons and pumps, a plunger of smaller or larger diameter can be readily substituted. The water enters the pump from the suction chamber C, through the suction valves, then passes partly around and partly by the end of the plunger, through the force valves, nearly in a straight course, into the delivery chamber D. The valves usually consist of small

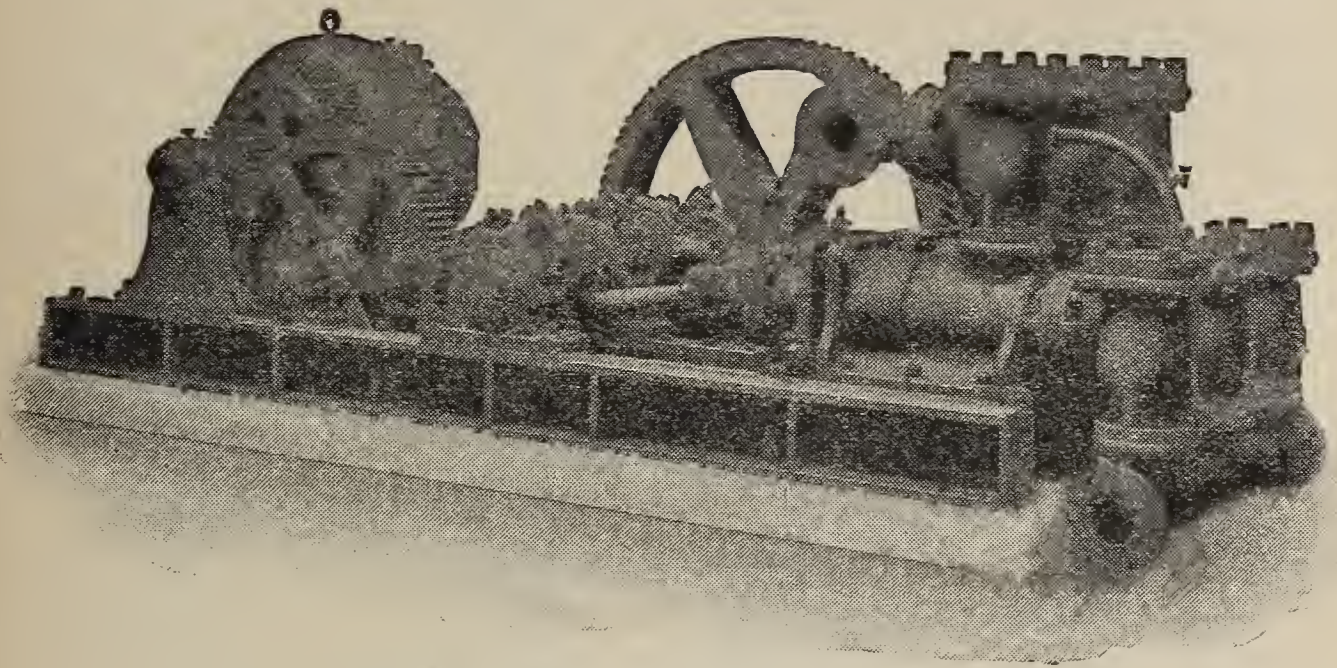


FIG. 551. — Electric three-throw ram pump.

discs of rubber or other suitable material. The plunger is located some inches above the suction valves, to form a settling chamber, into which any foreign substances may fall below the wearing surfaces. Two steam cylinders and two pumps are usually cast together to form one machine. The right-hand division moves the steam valve of the left-hand one, and *vice versa*; under this arrangement one pump takes up the motion when the other is about to lay it down, thus keeping up a uniform delivery of water. Steam ram pumps are now being gradually displaced by compressed air or electrically driven pumps, unless for where the pumps are situated in the shaft, or close to the shaft bottom; and even in these positions compressed air and electrically driven plants are now being very largely used. There are few cases in new and modern collieries where steam is now used as the motive power for driving pumps underground, as much better results are got from the use of electricity, not in efficiency alone, but in the ease with which it can be applied and taken to any desired point. All of the ram pumps

described here can be equally well operated by an electric motor, and many of what were formerly steam pumps are now being made to suit this form of power.

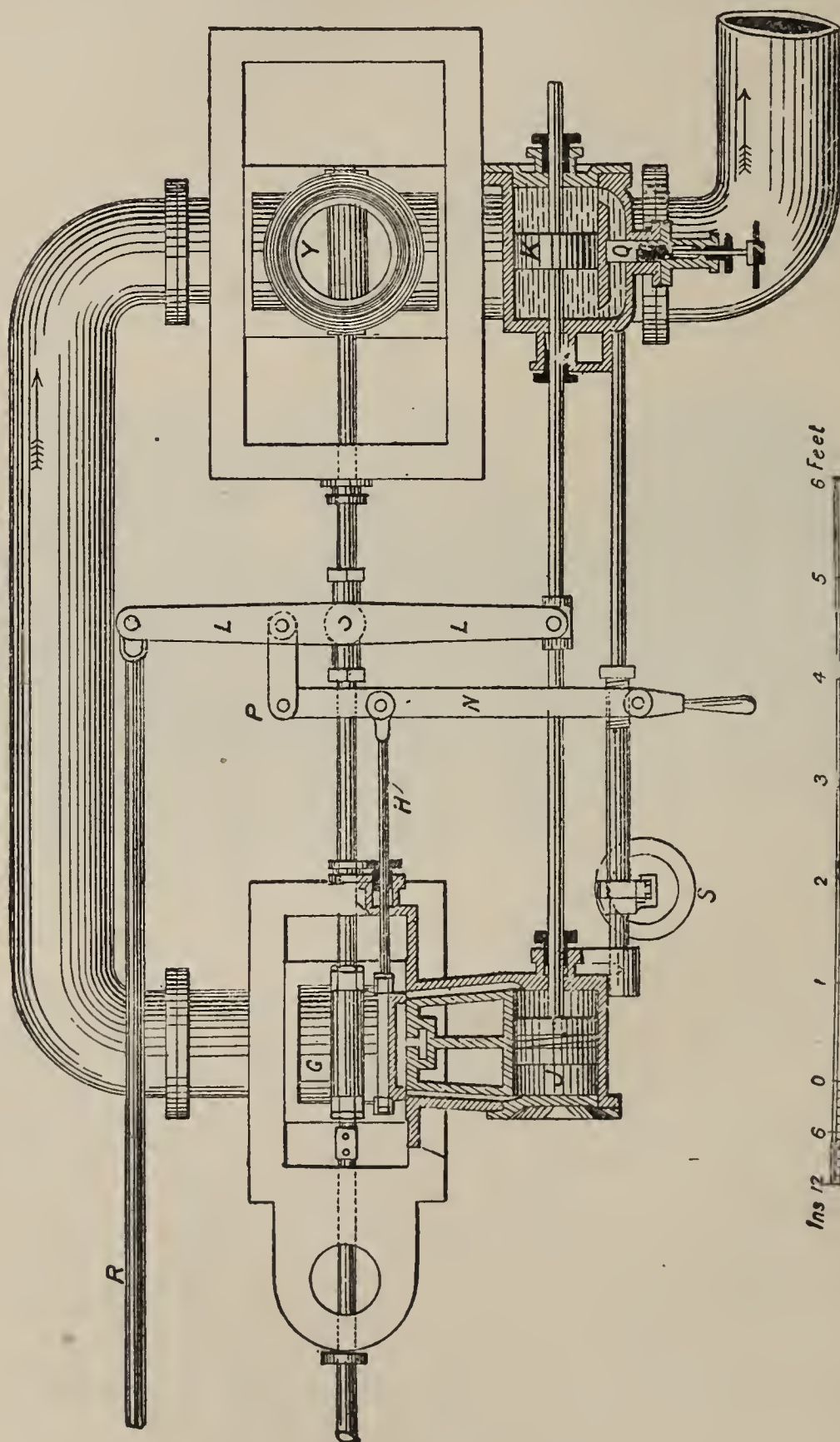


FIG. 552.—Davey's differential engine.

Davey's Differential Engine.—This engine is also largely used for pumping purposes. The main requirements to be satisfied in a good pumping engine are economy in consumption of fuel, safety in working, and immunity from stoppages.

The distribution of steam should be effected in such a way as to cause no shock or slip in the pumps ; and in the event of a sudden easing of the load, the engine should be safe ; in short, it should be self-governing under extreme variations of conditions. To secure this, the Davey Differential Valve Gear, which admits steam to the engine in proportion to the resistance to be overcome, and, in case of a sudden total loss of load, reverses the steam to catch the piston, was designed.

The action of the gear will be understood from fig. 552.

The main slide valve G is actuated by the piston-rod through a lever H working on a fixed centre, which reduces the motion to the required extent and reverses its direction. The valve spindle is not coupled direct to this lever H, but to an intermediate one, L, which is jointed to the first lever at one end ; and the other is jointed to the piston-rod of a small subsidiary steam cylinder J, which has a motion independent of the engine cylinder ; the slide valve being actuated by a third lever N, coupled at one end to the intermediate lever L, and moving at a fixed centre P at the other end.

The motion of the piston in the subsidiary cylinder J is controlled by a cataract cylinder K on the same piston-rod, by which the motion of this piston is made uniform throughout the stroke, and the regulating plug Q can be adjusted to give any desired time for the stroke. The intermediate lever L has not any fixed centre of motion, its outer end being jointed to the piston-rod of the subsidiary cylinder J ; the main valve consequently receives a differential motion compounded of the separate motions given to the two ends of the lever. Thus the cut-off can be suited to different loads which may be on the engine.

The illustration in fig. 553 shows the details of a valve made for a large Riedler pump (2200 gallons per minute) working against 1700 ft. head. On the left is the valve seat with its face marked A, the spindle of same terminating with a cap marked H. Next is the valve proper with its surface marked B. The shank of the valve is marked D, through which passes the stem. F is a rubber buffer fitting over shank of G, which in turn fits over valve shank D, and is prevented from slipping off same by the cap nut E. That portion of the valve bonnet extending into the valve chamber is marked P. This bonnet contains the packing, through which extends the spindle K, having on its end the forks J. Keyed to the spindle K is lever L, through which the spindle receives its motion from the pump wrist plate. The bronze pins M have taper ends which bear on the taper on edge of valve seat, and hold the valve seat down in position. The pins are forced in from the outside, thus doing away with the necessity of work on the inside of pump body should it be necessary to remove the valve and seat. The glands O prevent any leakage around the pins M. Groove Q is used for hydraulic packing. V refers to wedge-shape metal vanes cast on

the valve arms. These are for the purpose of giving the valve a slight turning motion on each opening, so as to ensure a good bearing at all times between the valve and its seat. It also prevents the severe cutting which might occur in pumps working against high heads, due to grit getting between the valve and the seat.

Riedler Differential Pump.—A differential pump is practically a double-acting pump with only two valves. By an arrangement of the parts, an equal amount of work can be done on each side of the steam piston during one revolution. In a double-acting pump four valves are required, viz., one suction and one discharge valve for

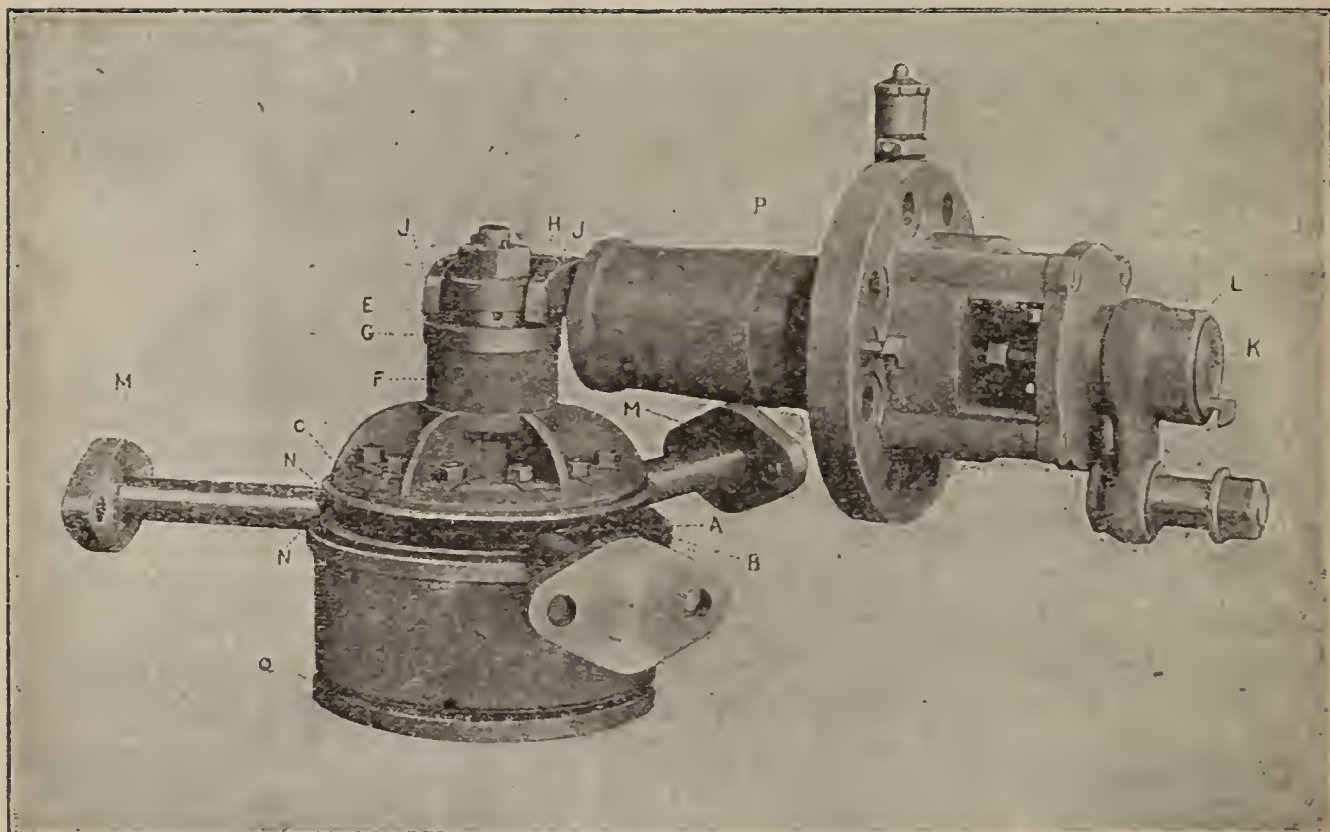


FIG. 553.—Riedler pump valve.

each end of the double-acting plunger. A differential pump has the advantage of always being primed, as will be seen by referring to fig. 554, where the column pipes D, the discharge space C, and the differential plunger chamber are always in connection. Thus the total pressure due to the water column is always on the differential plunger H. In other words, as long as there is water in the pipe, the pumping engine will always have resistance to overcome, even should suction be deficient. The arrangement, therefore, prevents undue severe hydraulic stresses on the different parts.

As it has only half the number of valves, this form of pump is simpler than the double-acting pump, and is used, in all cases, until the capacity becomes too great, so that the valves are cumbersome. When this condition obtains, it is better to use a double-acting pump instead, which would be half the size of a differential pump of the same capacity.

In working, the water enters the suction pipe A, passes into the suction air-chamber, and thence into the suction funnel B. When the main plunger J moves towards the steam cylinder, it draws in

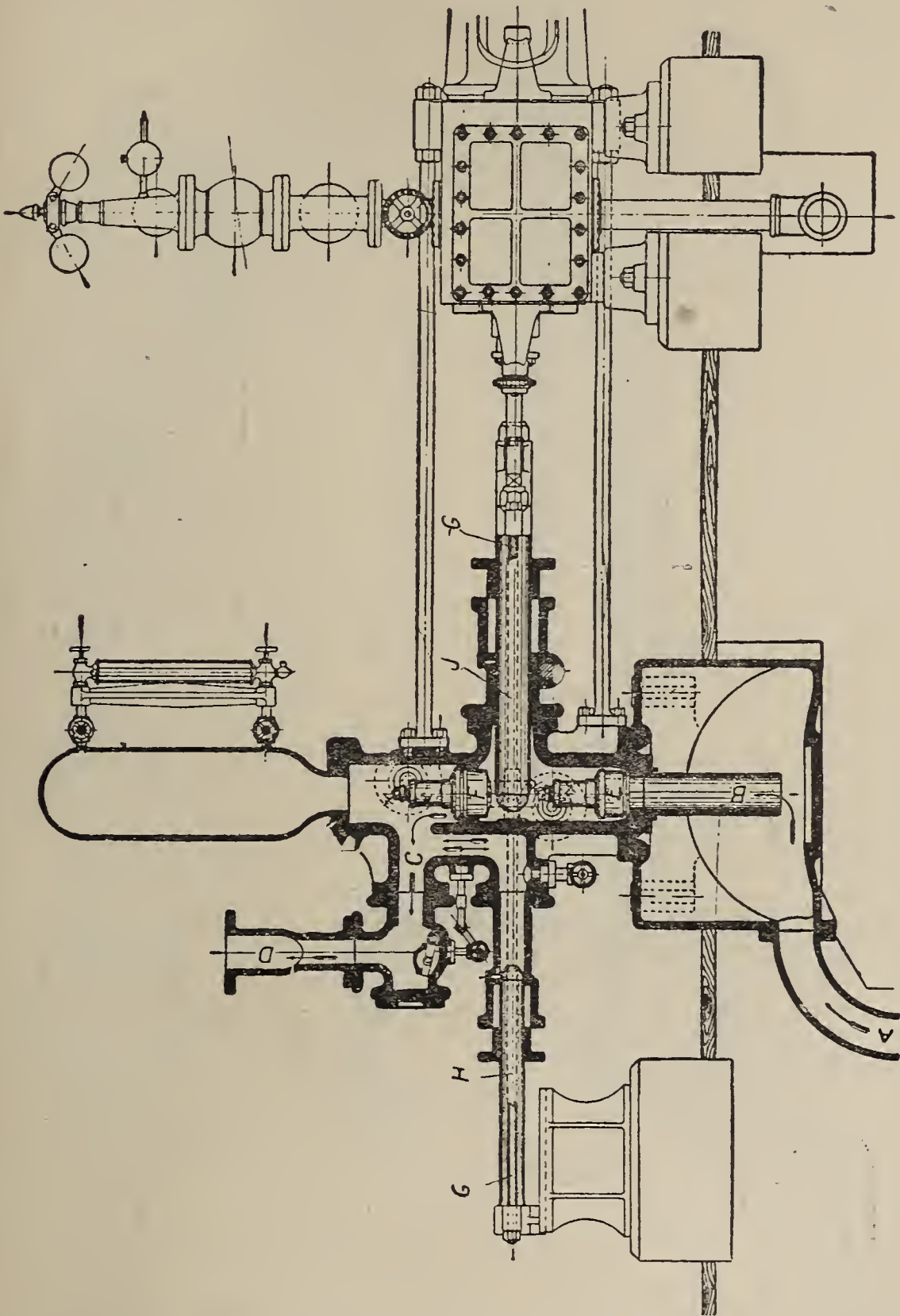


FIG. 554.—Riedler differential pump.

its displacement of water through the suction valve E, and on its return stroke, the suction valve having been mechanically closed, it forces a volume of water equal to its own bulk through the discharge valve F, half of this water passing out into the main pipe D, the

other half passing down and following the differential plunger H. The discharge valve F being now closed, the main and differential

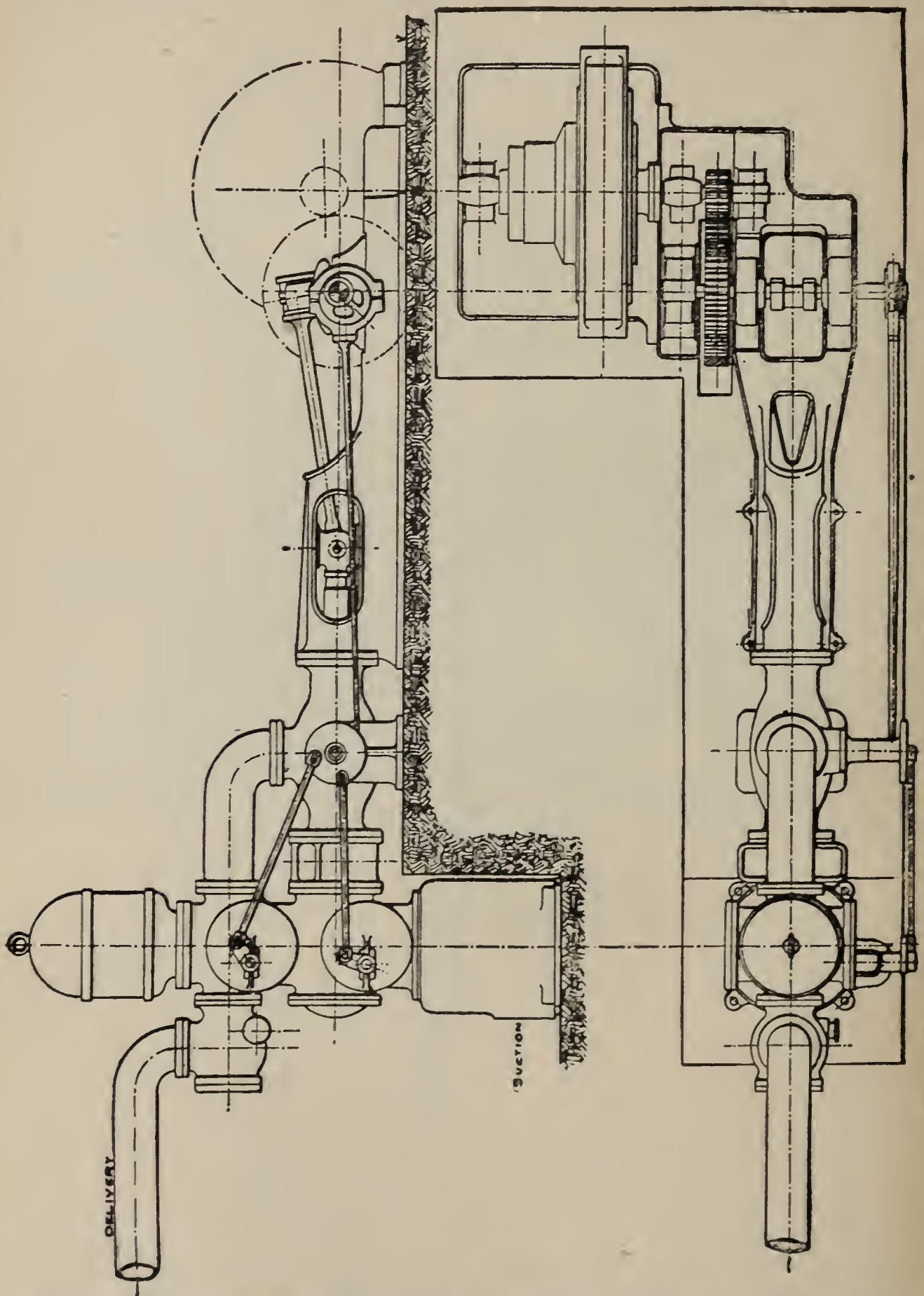


FIG. 555. — Plan and elevation of Riedler differential electrically driven geared pump.

plungers, which are connected by means of side rods, again move towards the engine, the main plunger drawing the water through

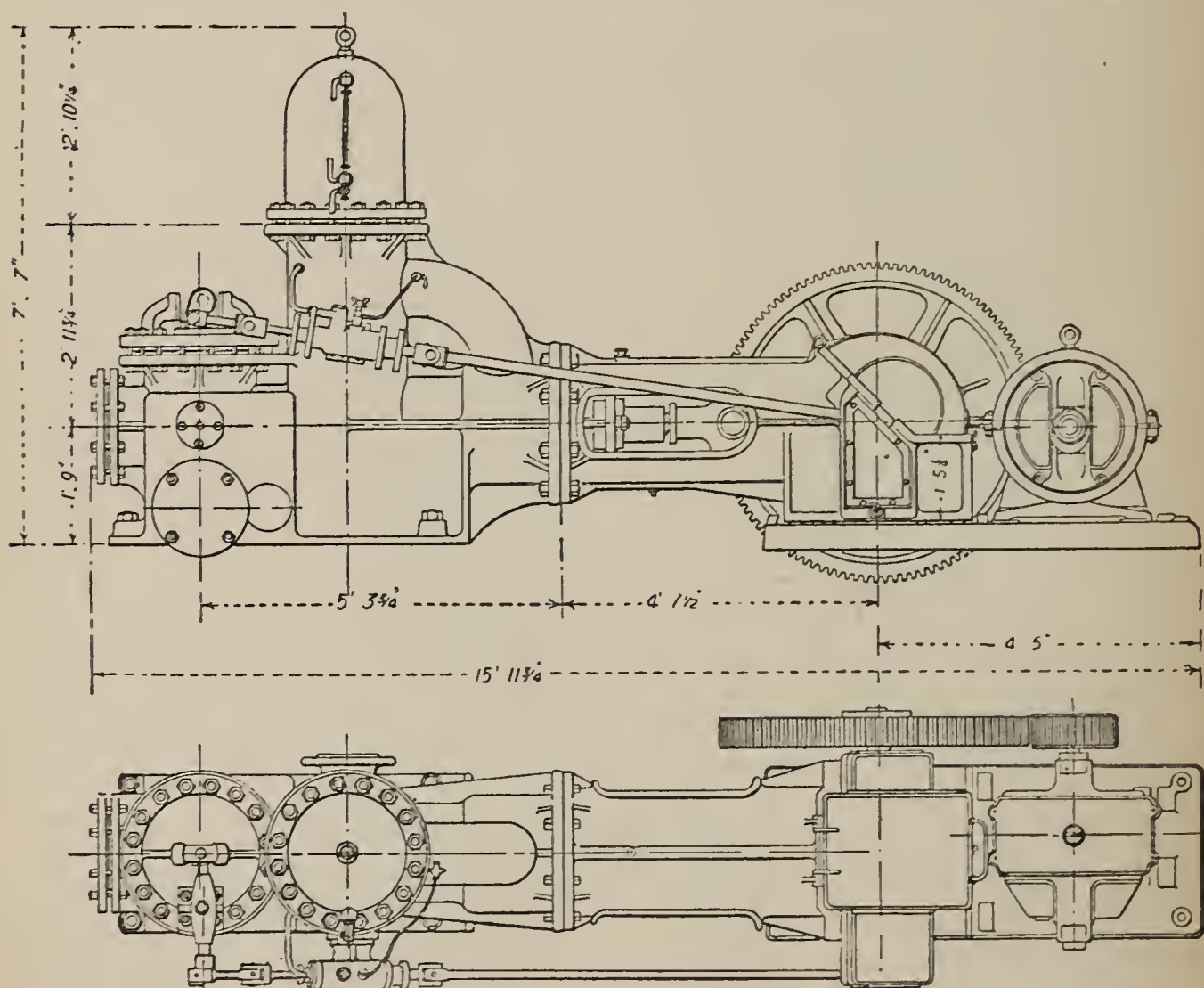
the suction valve as before (E), the differential plunger thus displacing a body of water equal in bulk to its displacement, and forcing it through the discharge pipe C into the main D.

The cross-sectional areas of the main and differential plungers are generally made in the proportion of about two to one, so as to equalise the work done, as is the case in a double-acting pump. The rods G are the side rods connecting the cross-heads of the main and differential plungers, these rods being always in tension. In front of C, and connecting to the main pipe D, is a clack valve (shown open). The valve is for the purpose of preventing the water in the pipe from running out when it is desired to remove the valves or examine the interior parts of the pump. This valve is kept open when the pump is working, but can be shut when required by means of levers on the outside of the clack chamber.

The main feature in this pump is that the valves are aided in their movements by mechanical means, and so nearly perfect action is secured. The valves are constructed so as to open freely without any mechanical aid, but a little before the time when they should close entirely, when the velocity of the water is considerably reduced so that a partial closing will offer no appreciable obstruction, a lever or rod operated by valve gear from the crank-shaft moves forward and closes the valve; the arm then recedes and removes the pressure from the valves before the time for opening arrives. The valve gear is constructed in various ways, and may be operated by levers, cams, or eccentrics. Fig. 554 shows the main outlines of the pump. These pumps are driven direct, and although they cannot be driven at such high speeds as some other types of steam pumps, they can be worked very successfully for very high lifts up to 2000 ft., and pump 600 to 800 gallons of water per minute. A Riedler pump has recently been installed as a duplicate to the large pumping engine on the surface at the Aitken Pit, Kelty, Fife, to deal with 1500 gallons per minute against a head of over 1200 ft.

Oddie-Barclay Differential Pump.—This is another pump of the differential ram type introduced a few years ago by Messrs Andrew Barclay, Sons, & Co., Ltd., Kilmarnock, well-known mining plant-makers. In the Riedler pump just described both the suction and delivery valves are operated mechanically, but in the Oddie-Barclay only the suction valve is so operated. The ram is of two diameter dimensions and is enclosed in suitable casing. It receives a reciprocating motion from the connecting rod and crank operated by the main crank by belt or wheel gearing. The valves are constructed of two concentric annular rings resting on suitable seats. On the outside of each valve is a guard, and between the valve and the guard there is a series of rubber buffers. The guard on the suction valve is controlled by means of the cross-rod and the rod situated parallel to the length of the pump driven by an eccentric from the

main shaft. On the suction stroke the guard on the suction valve moves inward and allows the valve to open, giving a wide passage to the water. The motion of this valve is simply a reduced form of that of the ram, so that as the ram accelerates its speed so does the valve, giving a greater area of opening as required. On the second half of the suction stroke the guard comes gradually back, shutting the valve slowly. The eccentric controlling the opening of the valve is set with a slight lead, so that although the motion of



FIGS. 556 and 557. — Plan and elevation of a single gear-driven Oddie-Barclay pump with an air-charger.

the valve is a reduced form of that of the ram, it is always a little in advance. The result is that the valve is always open and shut slightly before the commencement and finish of the stroke. On the delivery stroke the water is displaced through the delivery valve, half of the water flowing into the delivery column and half in behind the ram. This portion of the water which fills the space between the small ram and the casing is displaced on the return or suction stroke of the large ram. The form of valves gives a large area of opening with a comparatively small vertical rise of the valve rings, and prevents undue wear. An air-charging pump is also supplied,

except for low heads, thus ensuring an adequate supply of air being present in the air-vessel. The air-charger is worked from the eccentric rod from the main pump shaft, which also controls the

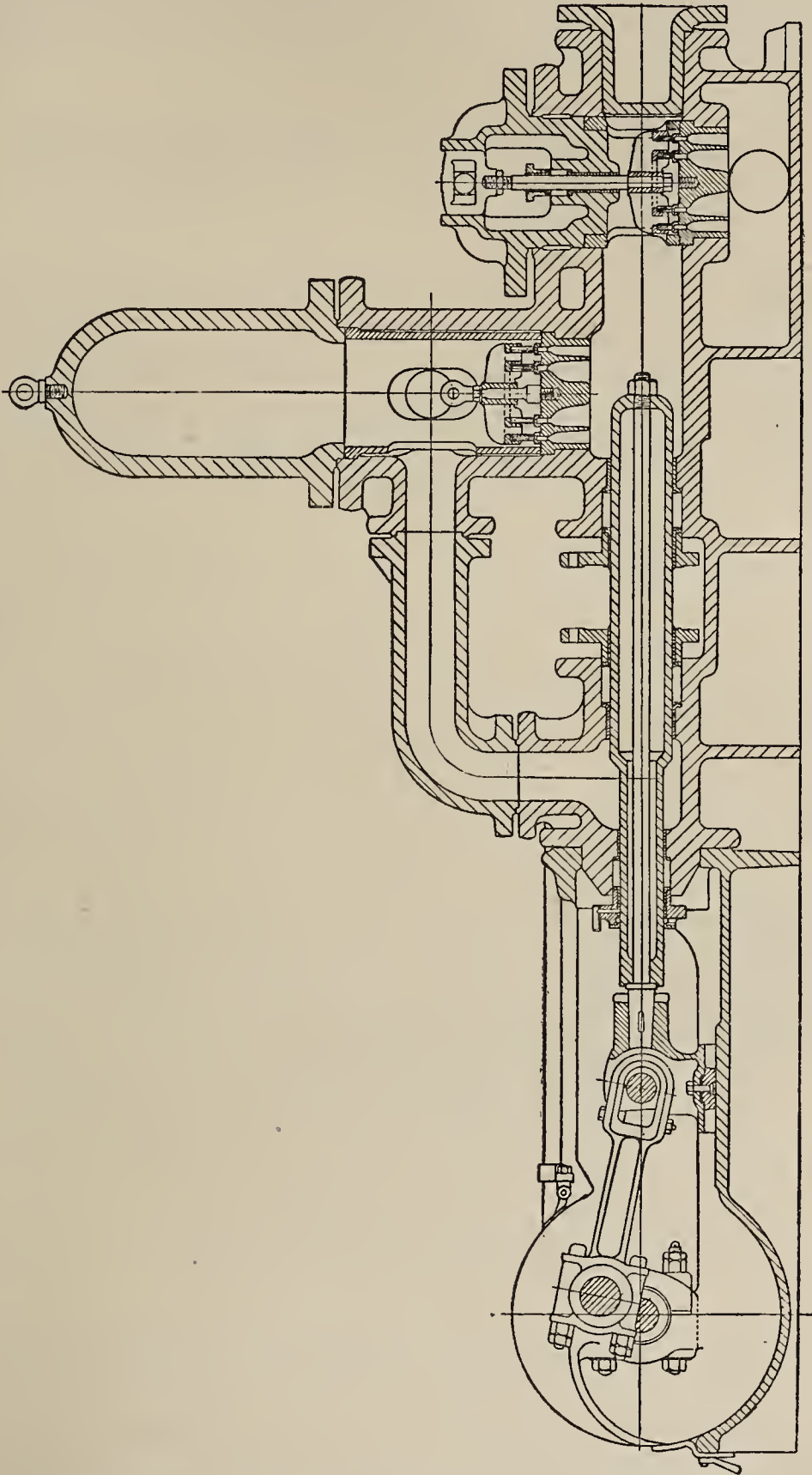


FIG. 558. —Sectional elevation of Oddie-Barclay pump.

valve. Lubrication is provided for all the working parts by a small force pump, also driven from the crank shafts. It is claimed for this pump that the pump-chamber is completely filled without eddies, and as the valve is firmly seated before the return stroke commences,

there is no danger of hammering or banging of the suction valve. The pump is made to be either steam or electrically driven with gear wheel or belt drive. A large number of these pumps have now been installed in collieries in various districts, and have given excellent results in raising large volumes of water through long vertical lifts.

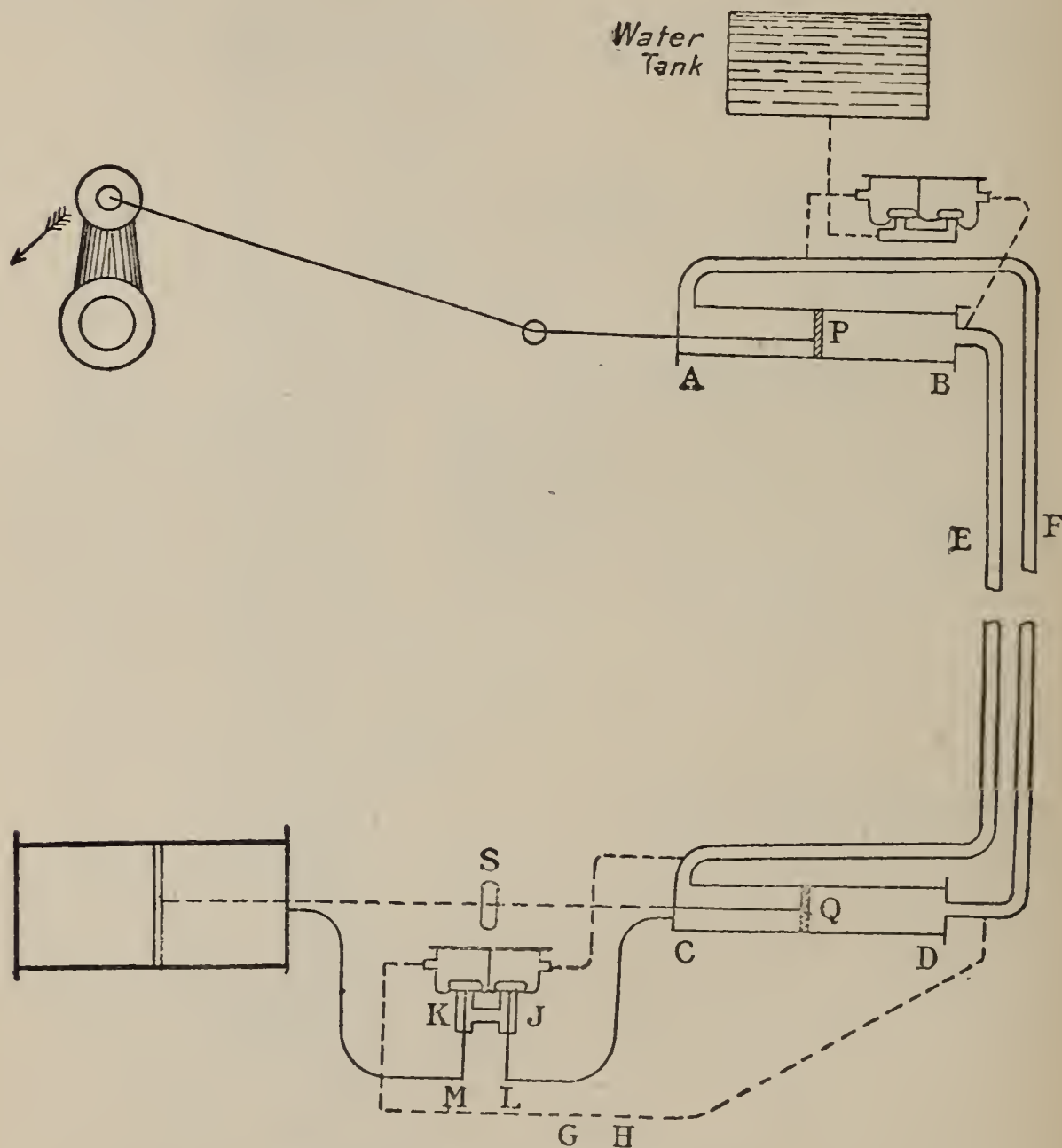


FIG. 559.—Moore's hydraulic pump.

Moore's Hydraulic Pump.—In this pump no rods are used, two columns of water being substituted for the ordinary solid rods connecting the steam engine to the pump. The action of the pump will be understood from fig. 559. On the surface is a cylinder A B, in which travels a piston P, driven by the tumbling crank of a steam engine. Underground there is another cylinder C D, exactly similar to the first. The piston Q, which travels in it, is connected to an ordinary double-acting pump. There are pipes F from the ends A

of the first cylinder A B to the end D of the second C D, and another, E, from the end B to the end C. The pipes and the cylinders are both kept full of water. When the surface piston makes its first stroke, the water is forced out of one end of the cylinder through the pipe to the corresponding end of the cylinder underground, and the piston is then driven back from C to D. When the stroke on the surface is reversed, the piston underground is forced in the opposite direction, and the motion is thus transferred from the engine on the surface to the pump underground, in the same way as would have been done had there been two rods instead of two columns of water. Should there be any leakage in one of these pipes, the plunger at the bottom would make a shorter stroke in the one direction than in the other, and would work towards the end, and unless there was some regulator, would knock the end of the pump off. This is obviated by having valves work by tappets, set at such a distance that, when the stroke is completed, they are opened, and the water allowed to pass from one pipe to the other, thus adjusting the stroke of the rams.

Curves based on the work done by one of these pumps 10 in. diameter, showed an efficiency of 66·26 per cent. The energy lost in the friction of the engine gearing and surface power rams was 10·24 per cent.; in transmitting the power through the pipes 14·36 per cent., and in the friction of the underground rams 9·12 per cent. These figures compare favourably with the efficiency of pumps operated by electricity or compressed air.

This hydraulic pump can work with a column of water of 150 fms. or less, but more satisfactory results are got when the column is between 40 to 80 fms. When it is too great, much trouble may ensue, and the pipe joints become very difficult to keep tight, while the pipes themselves are apt to burst, especially at bends. Again, at long distances (say over a mile underground) these pumps do not work satisfactorily, and in such cases it is better to employ either electricity or compressed air as a motive force.

Kaselowsky Pump.—With this pump a combination of hydraulic pressure and compressed air is used. The system includes at the surface: (1) a steam engine M (fig. 560) operating the water-compression pumps P and the air-compressor C, the latter furnishing the air for the pressure regulator R. Underground the plant consists of: (a) an hydraulic motor K, of a special form, operating the lifting pumps G; (b) two pressure regulators, R^1 and R^2 , for the water pipes.

In the shaft: (a) The pipe 1, conducting the water to the pumps; (b) the pipe 2, carrying the water discharged or exhausted from the pump; (c) the pipe 3, carrying compressed air to the regulators; (d) the pipe 4, for the water lifted or discharged from the mine. The water operating the pumps passes from the pump P, through the accumulator A, which regulates the pressure through the pipe 1 to

the motor K at the bottom of the shaft. After performing the work it is returned to the surface through the pipe 2 on which is the regulator R^1 , and is discharged into the reservoir S, from which the pump P^1 is supplied. The regulator, as shown, is formed of two cylinders, C_1 and C_2 , in which are two pistons, P_1 and P_2 , of such diameters that the ratio between them is 1:10. The water acting on P , with a pressure, say, of 200 atmospheres (3000 lbs. per sq. in.) is counterbalanced by a compressed air pressure at 25 atmos-

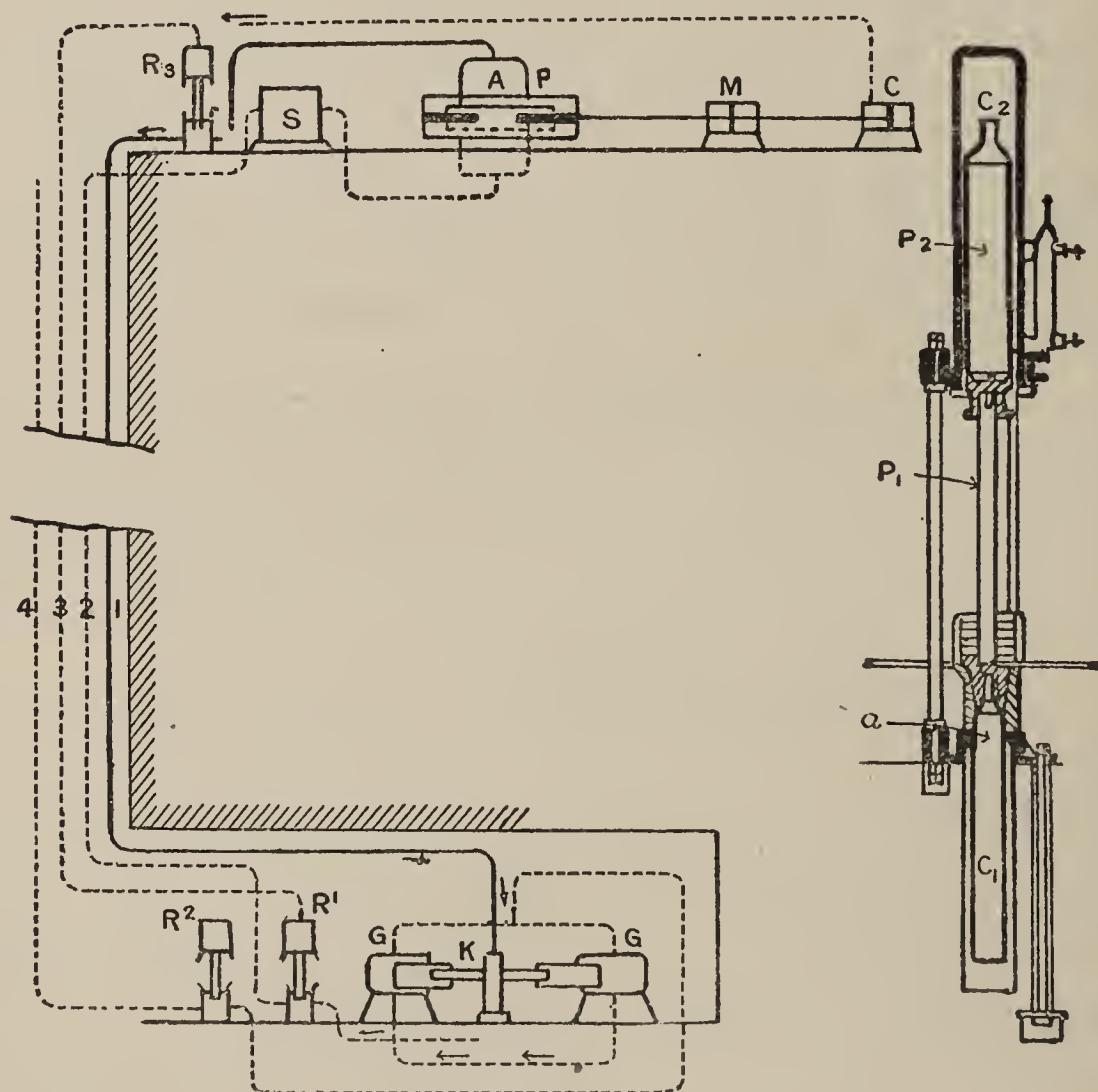


FIG. 560. — Kaselowsky pump.

pheres (375 lbs. per sq. in.) or P_2 . The piston is a hollow cylinder of phosphor bronze. The water piston P_1 has a small cavity a , which permits the escape of water in case the pressure rises too high and causes the piston P_2 to rise too high. Several pumping plants on this system have been installed at collieries in Germany, where they have given a useful effect of 69 to 75 per cent. The principal advantages of this method are the small size of the different parts, both fixed and moving, and the ability of the pump to work when submerged, which is impossible with steam. On the other hand, it is rather an expensive system of pumping.

Pulsometer Pump.—The Pulsometer is very largely used for pumping water under various conditions, particularly when the lift is small. It will work best, and give the greatest efficiency, when the height to which the water is to be raised is between 30 and 50 ft. It is very suitable for drainage of dip workings, or sinking shafts, or

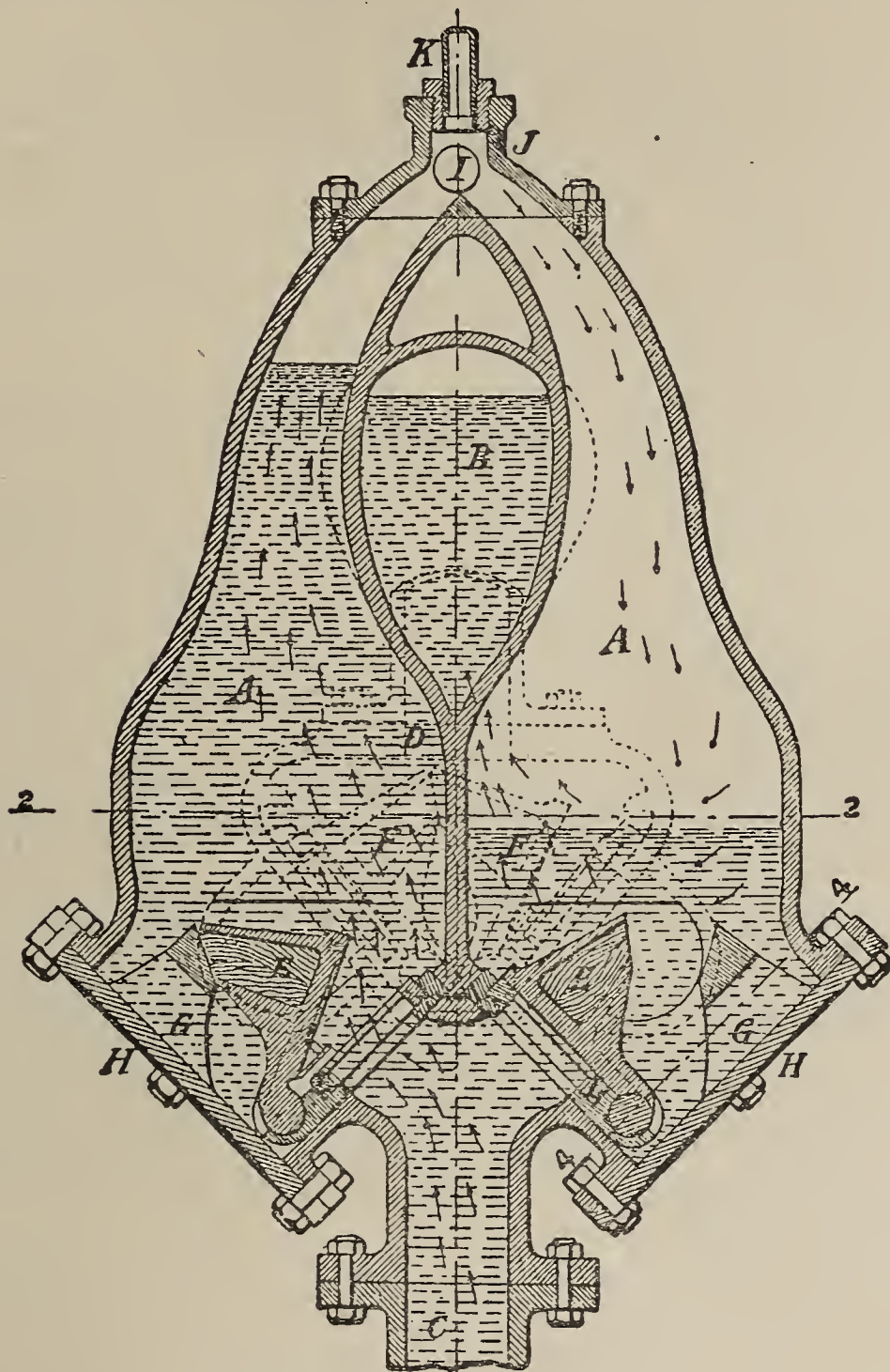


FIG. 561. — Pulsometer pump.

for raising water from settling ponds to coal-washing machines, as it can work fairly well with dirty or gritty water. The pulsometer is entirely different in construction from an ordinary steam pump, inasmuch as it has no steam cylinder, piston, piston-rod, or bucket. Its construction may be understood from the accompanying illustration in fig. 561.

The body consists of a casting shaped somewhat like a pear, and divided into two chambers A A joined side by side, and with tapering necks bent towards each other, surmounted by another casting called the neck J, accurately fitted and bolted to it, in which the two passages terminate in a common chamber, wherein is fitted the ball-valve I, which can oscillate between seats formed at the junction of the neck. Downwards, the chambers A A are connected with the suction passage C, wherein the inlet or suction valves E E are arranged. A discharge chamber, common to both chambers, and leading to the discharge pipe, is also provided, and this also contains one or two valves F F, according to the purpose for which the pump is required. The air-chamber B communicates with the suction. The suction and discharge chambers are closed by covers H H, accurately fitted to the outlets. These can be readily removed when access to the valves is required.

Starting the Pulsometer.—To set it at work, the pump is filled with water, either by pouring water through the plug-hole in the chamber, or by drawing the discharge. Steam being admitted through the pipe K, by opening the stop-valve to a small extent, it passes down that side of the neck which is left open to it by the position of the ball, and presses upon the small surface of water in the chamber which is exposed to it, depressing it without agitation, and consequently with very little condensation, and driving it through the discharge opening and valve into the rising main.

The moment that the level of the water is low enough to uncover the horizontal orifice which leads to the discharge, the steam blows through with a certain amount of violence, and being brought into intimate contact with the water in the pipes leading to the discharge chamber, *instantaneous condensation takes place*, and a vacuum is in consequence so rapidly formed in the newly emptied chamber that the steam ball is pulled over into the seat opposite to that which it occupied during the emptying of the chamber, closing its upper orifice and preventing the further admission of steam, and making the vacuum complete until water rushes in, as it does immediately through the suction pipe, lifting the inlet valve E and rapidly filling the chamber A again.

The condition of things is now exactly in the same state in the second chamber as it was in the first, and similar effects are therefore obtained.

Small air-cocks are screwed into the cylinders and air-chambers, to prevent the too rapid filling of the chambers on low lifts. While the pulsometer is admittedly a handy and useful pump under various conditions, it has the drawback that it consumes a large amount of steam for the work done, compared with direct-acting steam pumps. For instance, a pulsometer raising 100 gallons of water per minute, to a height of only 26·25 ft., required 29·7 lbs. of coal per H.P. per hour, which is a very large consumption, considering the conditions.

A good deal of difficulty is also sometimes experienced with the steam ball valve getting worn flat in some places and sticking.

Centrifugal Pumps.—Centrifugal pumps have been long used as blowers for air in forges and furnaces, and are now largely used for raising water to moderate elevations. They are particularly well

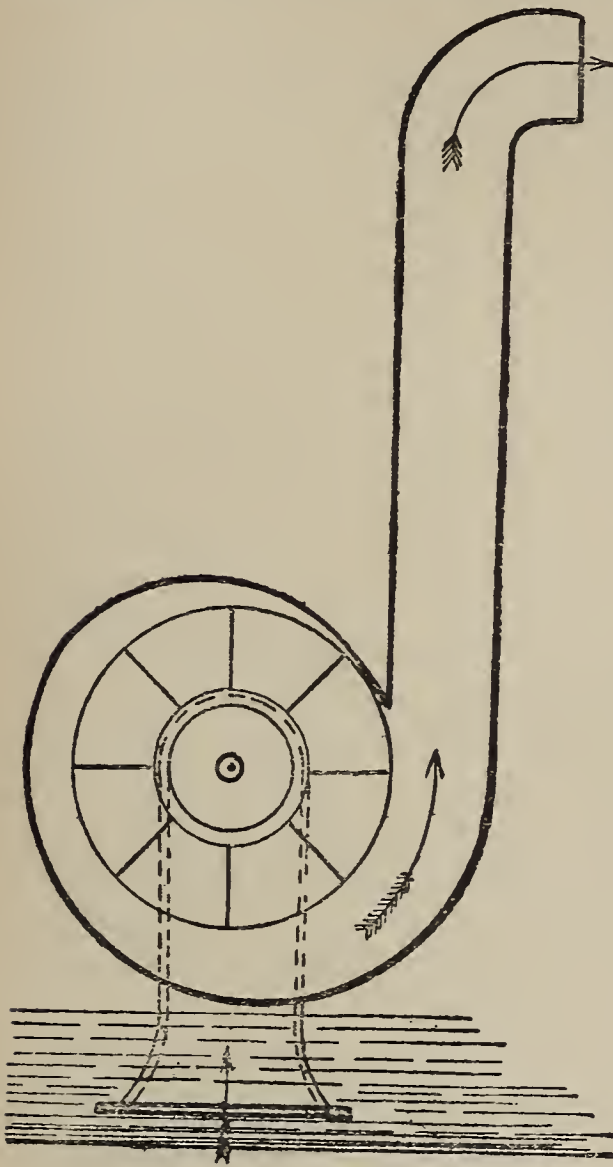


FIG. 562. —Centrifugal pump.

adapted for disposing of dirty gritty water, such as the discharge water from coal-washing machines, or where the water from machines has to be used over again, providing the height be not too great. They will work under such conditions much more efficiently than pulsometers. The construction of the pump will be understood from fig. 562. Inside a flat casing of approximately circular outline are the paddles or blades, which extend from near the centre outwards to the circumference, and are usually curved backwards. The water between the blades tends, in virtue of the centrifugal force, to move outwards, and is allowed to pass off through a large discharge orifice tangential to the circle described by the paddles. The height to which water can be raised, if there is no loss, by centrifugal pumps, may be expressed by the formula $h = \frac{v^2}{2g}$;

in practice, however, h is only equal to about $\frac{3}{4} \frac{v^2}{2g}$. The velocity of the blades is usually taken at $N = 10 \sqrt{h}$.

Centrifugal pumps, to work well, must have a very short suction pipe, or, what is better, have the water flowing into them, or be submerged altogether in the water to be raised.

If not submerged, they ought to be primed with water before being started, otherwise, in driving the air out of them, they simply act as a blower.

Turbine Pumps.—With the type of centrifugal pump just described only a very limited head could be dealt with, 50 to 80 ft., and even with this a low efficiency was generally the result. The problem was to get a centrifugal pump that would pump water

against a head of a greater height than 100 ft., say 500 to 1000 ft. This has now been accomplished by the introduction of turbine pumps, which are practically centrifugal pumps in series. Thus, if an ordinary single pump can deliver water against a head of 80 ft., the addition of another chamber will give a final delivery of 160 ft., while three chambers will enable the pump to discharge the same quantity of water against a head of 240 ft. ; so that while in the past a lift of 50 to 80 ft. was considered the limit for pumps of the rotary type, lifts of ten times those heights are now quite common. The range of duty of turbine centrifugal pumps is very wide, the lift being capable of almost indefinite increase by the simple addition of stages. The use of turbine pumps is making very rapid headway in collieries for raising large quantities of water against high heads. So rapid has been its progress that it bids fair to oust ram and other forms of pumps from colliery work altogether. In many instances it has taken the place of the ordinary types of ram pumps for raising water in shafts. As an instance of this, at Bardykes Colliery, Blantyre, the whole of the water is raised in the shaft by means of centrifugal pumps, instead of rod pumps formerly in use. The total depth is about 1400 ft., and the water is dealt with in three stages. At the Powell Duffryn Colliery, South Wales, there is a 900-H.P. electrically driven turbine pump in operation.

It was principally the electric motor that made the turbine possible. Both the turbine pump and electric motor are essentially high-speed machines, and hence they can be coupled up direct. As such pumps are usually driven at a high speed, 1200 to 3000 revolutions per minute, they do not require to be of large dimensions, and thus take up little space, and are easily handled, features which make them specially valuable for mining work.

The principle underlying the turbine pump is similar to that of the centrifugal, except that the velocity of the water (kinetic energy) is converted into pressure (potential energy) by means of a diffuser having guide plates. This construction is specially suitable for high-lift pumps, the necessary head being obtained by placing a number of impellers in series upon a common shaft. Thus all turbine pumps have the following features in common.* One or more impellers, mounted on a shaft, revolving in suitable chambers, each delivering the water through a diffusion vane into the succeeding impellers, and finally into an annular space from which the water passes into the delivery pipe.

In recent years the turbine pump has been increased in power and efficiency by the provision of diffusion vanes. The diffusion vane takes hold of the water after it leaves the impeller at a high velocity and guides it into suitable passages until the velocity is comparatively reduced ; the kinetic energy due to the velocity being thus converted into static energy. In the turbine pump there is no

* *Proceedings of Ass. Min. El. Engs.*, vol. ii. p. 250.

actual displacement of water as in a ram pump, as the action depends on the velocity imparted to the water, due to the revolving impeller, which velocity is ultimately changed into pressure head. There is thus no necessity to provide relief valves on a turbine pump, as the pressure can never rise above that due to the velocity set up by the speed of the impellers. Consequently, the pump can be started up with the delivery valve closed, which is an advantage. The water always passes through turbine pumps in one direction, and is acted on equally by the impellers at all parts, and, therefore, the flow of water is continuous, and there is no necessity to provide air-vessels as in a ram pump. Shocks, so common to the latter class of pump, are thus avoided, and as the flow of water is constant and uniform, the minimum size of delivery pipes can be used. We have already stated that the speed at which turbine pumps can be run varies from 1200 to 3000 ft. per minute, 1500 to 2000 ft. being a common speed, but there is a certain velocity at which a maximum efficiency will be obtained, and if the speed varies from this there will be a loss in efficiency. When water is flowing through passages at a high velocity, or a body, such as the impeller, is moving through the water at a high speed, the friction developed may attain considerable dimensions. As the friction in the latter case is found to vary as the cube of periphery velocity (v^3), a limit is reached when it would be impracticable to increase the velocity beyond it.

The fundamental principle underlying these pumps is that known as 'Bernoulli's Theorem,'* which states that "the energy in a liquid flowing through a pipe is constant if we neglect friction." In other words, the sum of the pressure energy, kinetic energy, and potential energy contained in each pound of water is the same at any point in its line of flow. If the difference in potential energy be negligible, the difference in pressure energy may be calculated as follows:—

$$\frac{p}{W} + \frac{V^2}{g} = \frac{p_1}{W} + \frac{V_1^2}{2g} \text{ or } p_1 - p = \frac{W}{2g}(V^2 - V_1^2),$$

where V = velocity of water in feet per second at a point A.

V_1 = " " " " " " " " B.

p = pressure in lbs. per square inch " " " " A.

p_1 = " " " " " " " " B.

W = weight of column of water 1 ft. high and 1 sq. in. in section.

g = gravity = 32.2.

If H = head in feet which the water is able to overcome owing to its increased pressure,

$$\text{then } H = \frac{p_1 - p}{W} = \frac{V^2 - V_1^2}{2g}.$$

For example, suppose that water in the pump at point A has a velocity of 120 ft. per second given to it, and at point B the velocity

* *Trans. Scot. Inst. Min. Students*, vol. ii. p. 17.

of the water is reduced to $2\frac{1}{2}$ ft. per second ; then the gain in head,

$$H = \frac{120^2 - 2.5^2}{2 \times 32.2} = 223.5 \text{ ft.}$$

This principle is applied, as already explained, by allowing the water to flow into the pump at a comparatively low velocity, and later a high velocity is given to it by the impellers. After leaving the impellers it passes through a passage so arranged that the velocity is reduced to almost the same velocity with which it entered the pump. Thus the velocity of the water (kinetic energy) is converted into pressure (potential energy), the water then being able to overcome the equivalent head in the manner shown.

To calculate the work* done by a pump of this type we must take into consideration the momentum given to the water by the impeller, and also the *moment* of momentum. Suppose the water enters the eye of the pump with only a radial velocity, and leaves the impeller with a velocity of whirl $= V_1$, then if W = weight of water flowing through the pump per second, the change of momentum which has taken place during its passage $= \frac{W}{g} V_1$; and the change in moment of momentum will be $= \frac{W}{g} V_1 \cdot r$. (where r is the radius of impeller). The work done per second by the twisting moment on the shaft $= \frac{W}{g} V_1 \cdot r \cdot w$. (where w = angular velocity per second). That is, work $= \frac{W}{g} V_1 - V$ (where V = velocity of periphery of impeller in feet per second = W.P.).

The useful work done by the pump is $W.H.$, where H = head in feet.

$$\therefore \text{Efficiency} = \frac{WH}{\frac{w}{g} V_1 \cdot V} = \frac{Hg}{V_1 \cdot V}.$$

The actual efficiency obtained from turbine pumps is about 70 per cent. when new and tested under favourable conditions, but later, when the pumps have been in actual operation for some time, a fair average is about 60 per cent.

Messrs Mather & Platt, of Manchester, have recently introduced a high-lift centrifugal pump. This machine (fig. 563) is known as the 'Patent High-Lift Turbine Pump,' and its main feature is that it consists of one or more sets of vanes, or impellers, each set running in its own chamber, but upon a common shaft, the delivery pressure of the water varying directly as the number of chambers used. In Mather & Platt's patent centrifugal pump the water enters the revolving wheel axially, traverses the curved internal passages

* *Trans. Scot. Inst. Min. Students*, vol. ii. p. 20.

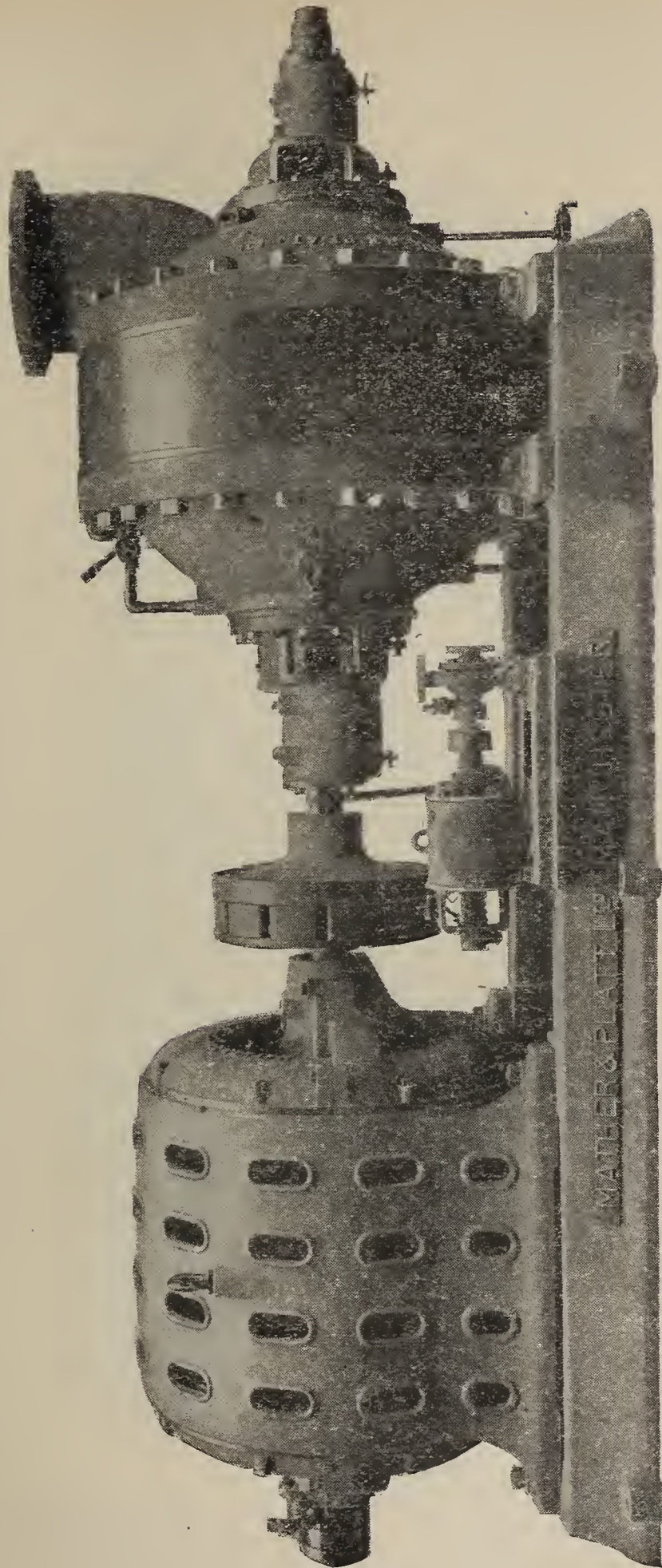


FIG. 563.—Mather & Platt high-lift turbine pump.

between the vanes, and is discharged tangentially at the periphery into a guide ring of special construction; this conveys it to the annular chamber in the body of the pump, where the velocity head

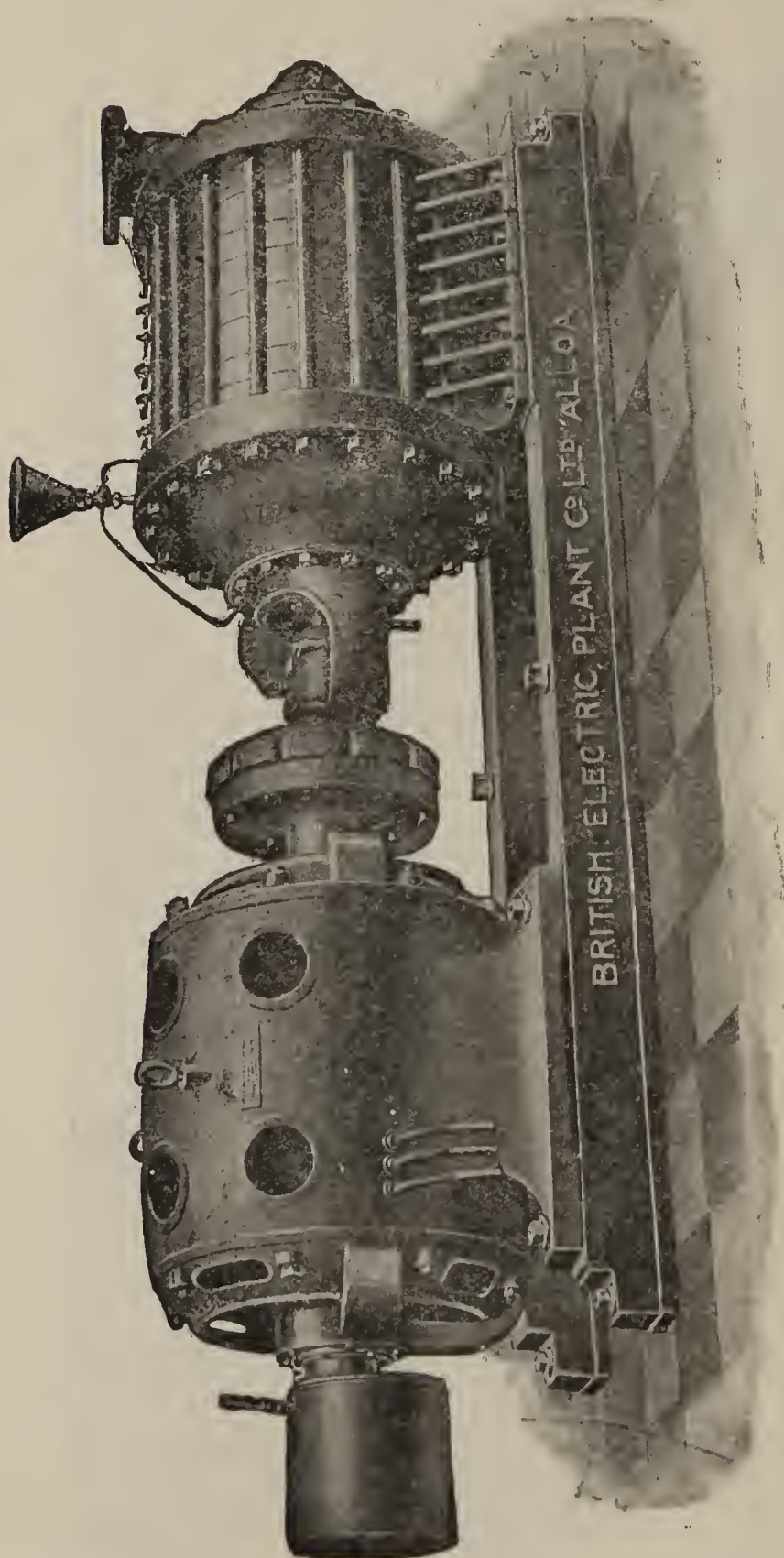


FIG. 564. — High-lift turbine pump, by the British Electric Plant Co., Ltd.

imparted to the water by the wheel is converted into pressure head. From this chamber the water is finally discharged into the delivery pipes, or, if the pump be a multiple one, into the second and subsequent chambers. A special feature of this pump is the provision of

the stationary guide ring mentioned above ; this is fixed concentric with the revolving vanes, and, owing to its design, enables the conversion of velocity into pressure head more perfectly than hitherto, thus increasing the possible height of lift and efficiency of the pump.

The British Electric Plant Co., Ltd., Alloa, are also well-known makers of high-lift turbine pumps.

The chief features in this firm's pumps are that there is no outer casing, and a self-balancing arrangement. In several other makes of turbine pumps a balance piston is fixed on to the end of the shaft, and consists of an annular disc, bearing against a turned face on the end chamber. The annular space surrounding this disc is connected with the suction side of the pump. Leakage water under pressure passes to the interior of the annular disc, causing the same to lift off its face, when the imprisoned water immediately escapes to the suction and the disc goes on to its face. In ordinary course of working, the disc takes up a permanent position, due to the out-of-balance position of the pump, just permitting sufficient pressure water to escape to overcome the out-of-balance. The disadvantage of the balance piston is that as the wear increases in the neck rings of the impellers, the leakage tends to increase. The out-of-balance thus becomes greater and the efficiency of the pump decreases. The British Electric Co. do not employ this device. In their pump the end thrust is destroyed by equalising the pressure on each side of the impeller by means of spiral passages connecting the chambers on the two sides of the impeller. This arrangement is simple and effective in operation, entirely achieving the desired result. The reduction of the parts is a desirable feature, and the elimination of the balance piston simplifies and cheapens the design.

The Pulsometer Engineering Co., Ltd., Reading, are now also makers of turbine pumps for collieries.

The casing of their pump is of cast iron, and consists of a number of sections or stages held together by long bolts passing through the end covers, which form, respectively, the suction and discharge chambers. The makers claim the advantage that this construction obviates the difficulty that exists when the impellers and diffusion rings have to be withdrawn axially through a long barrel. Alignment is secured by registering or nipling of the component parts, and tapped holes for forcing screws are provided, to ensure rapid dismemberment. The faces in contact with the bosses of the impeller are of bronze, and renewable. The diffusers are separate from the casing and fit into tooled recesses. The impellers are of bronze, and each impeller is mechanically balanced, independently, in order to eliminate vibration, and is secured to the shaft by keys. End thrust is eliminated by having the bosses so arranged as to balance the water pressures on each side, and also the axial thrust due to the change in direction of flow. To maintain the impellers in their correct position, relative diffusion rings are provided,

outside the pump, for doing this, an indicator being fitted for show-

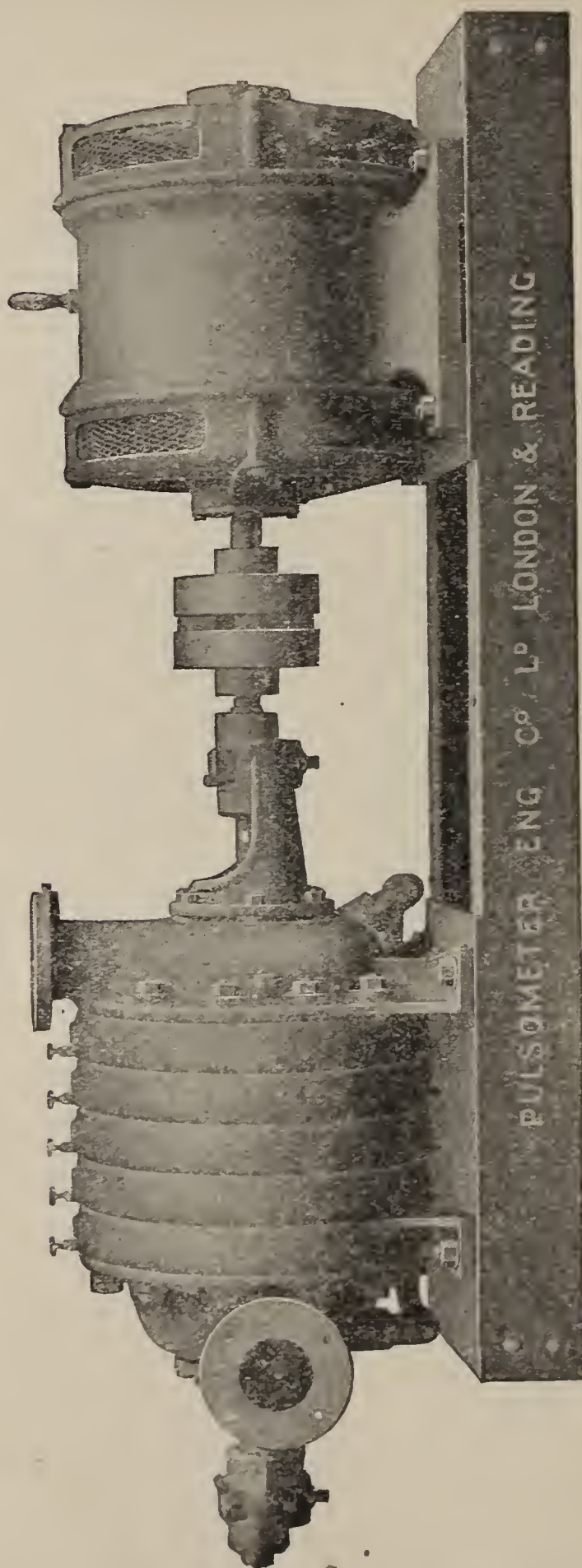


FIG. 565.—Pulsometer turbine pump.

ing the correct position. Any correction that may be required is effected by means of an adjustable bearing, which serves also for

taking any thrust caused by the driving gear The makers also

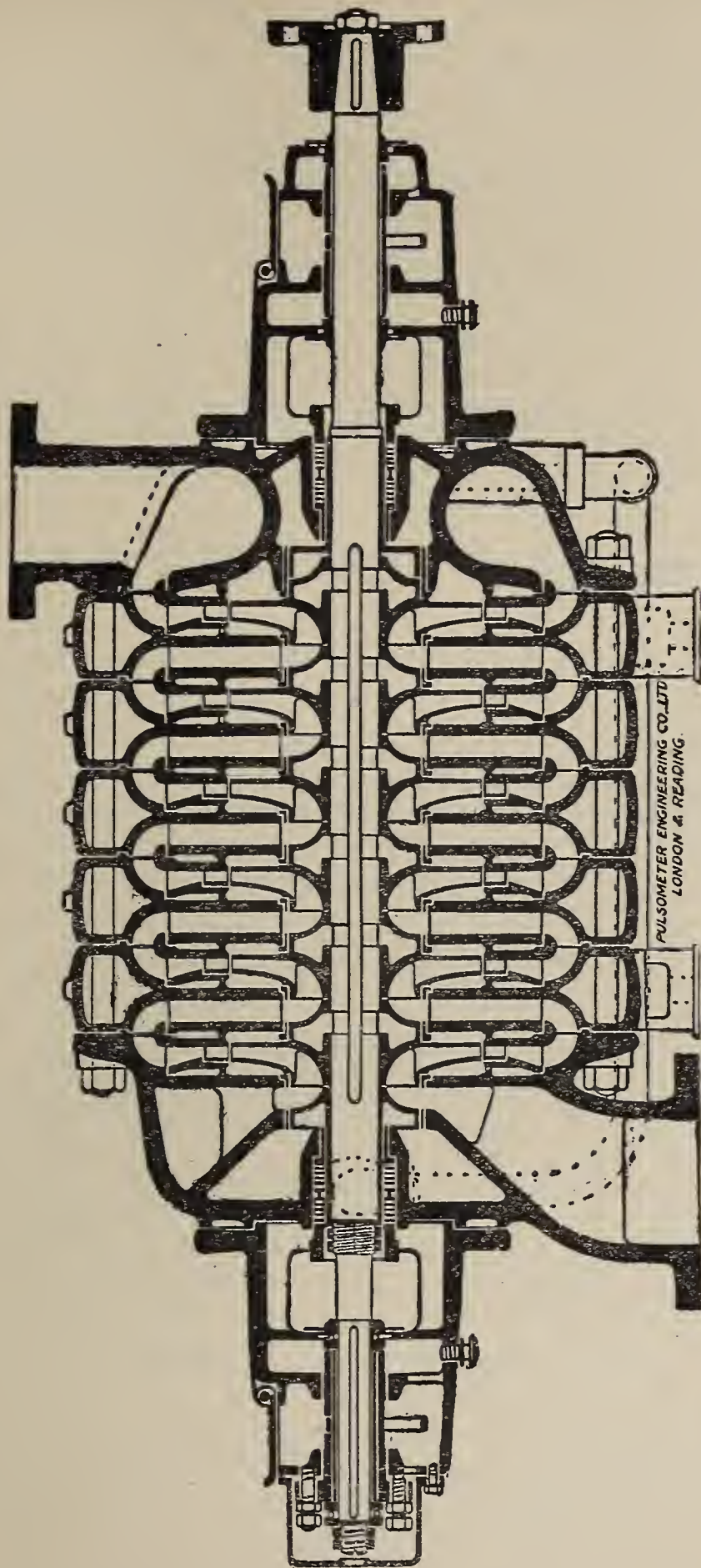


FIG. 566.—Section of Pulsometer turbine pump.

claim that in their high-lift turbine pump it is impossible to seriously overload the motor, even when the head is reduced to nil.

There are a large number of other makes of turbine pumps, made by home and foreign firms, but it is impossible to find space here to describe each separately. Besides, the construction of most of them are very similar, differing only in details.

There can be no doubt that pumps of the types described have advantages over the ordinary plunger pumps so largely used in mining work, and they are specially useful for dip workings. They can be easily moved into a new position with the extension of the workings, which in itself is a great advantage. Other advantages claimed for these centrifugal pumps are:—

They have few moving parts in contact, thus reducing wear and tear. As they can be designed to run at a high speed, they occupy little space; that is, very much smaller pump chambers are required compared with ram pumps. Thus for ram and turbine pumps of equal capacity the cost of making chambers in one instance was £900 for ram pump and £240 for turbine pump.

The following formulæ are used for calculations with centrifugal pumps:—

(I.) *To find the required peripheral speed of the impeller or wheel in feet per second for a given head.*

$$V = k\sqrt{H}$$

where V = peripheral speed in feet per second,

H = head water is to be delivered against in feet,

k = a coefficient = 8 for small pumps and 9.82 for large pumps;

$$\text{or } H = \frac{V}{64} \text{ in small pumps,}$$

$$\text{and } H = \frac{V}{90.25} \text{ in large pumps.}$$

(II.) *To find diameter of suction and delivery pipes in inches.*

$$d = .225\sqrt{g}$$

d = diameter of pipe in inches.

g = gallons delivered per minute.

(III.) *To find diameter of wheel for a given quantity of water and given head.*

$$D = .18\sqrt{\frac{g}{\sqrt{H}}}$$

where D = diameter of wheel in feet.

g = quantity of water in gallons per minute.

H = head in feet (per stage).

0.18 = coefficient.

Example.—Find diameter of impeller to raise 486 gallons per minute through a head (per stage) of 150 ft.

$$D = .18\sqrt{\frac{486}{\sqrt{150}}} = .18\sqrt{\frac{486}{12.24}} = 1.13 \text{ ft.,}$$

or $D = 2d$ to $2.5d$, where d is diameter of suction or delivery pipe as found in (II.).

(IV.) *To find revolutions of wheel per minute, when height of delivery and circumference of wheel are given.*

$$R = k \frac{\sqrt{H}}{D}$$

R = revolutions per minute.

H and D = same value as above.

k = 153 for small pumps and 187 for large pumps.

Example.—What speed would the pump require to be run at if the head (per stage) is 150 ft. and the diameter of impeller 1.13 ft. as above?

$$R = 153 \frac{\sqrt{150}}{1.13} = 153 \times \frac{12.24}{1.13} = 1657.27 \text{ revs. per minute.}$$

(V.) *To find effective horse-power required, given gallons per minute, for a given quantity of water to be delivered against a given head.*

E.H.P. = effective horse power.

G = gallons to be delivered per minute.

H = same value as above.

E = 50 to 80 per cent. (.5 to .8) according to head and type of pump.

$$\text{E.H.P.} = \frac{G \times 10 \times H}{E \times 33,000}$$

Sinking Pump.—During sinking operations an arrangement of pumps is required differing from that in use under ordinary circumstances. A common arrangement in sinking is a 'sliding suction' pump working through a 'packing gland.' It may be lowered as the sinking proceeds. A short joining piece is used between the sliding suction and the working barrel, this short piece being made of weaker metal than the other parts, so that in case of a side stroke from a shot it may give way, without injuring the more expensive parts. The method of operating is usually to fix all the pipes above the working barrel with collaring, and to allow the working barrel and sliding piece to be lowered as sinking proceeds. The sliding piece is generally made the length of one of the pipes, so that when it has been let out this distance, the column is 'cut' above the working barrel and another length added, when the pumping may proceed as before. The objection to this method is, that the pipe column requires to be cut at intervals, say 9 ft., as the sinking proceeds; but if the pumps are not large it works very satisfactorily. For light pumps a strong flexible hose may be used instead of the sliding suction. The pump rods are usually longer than the column of pipes, and short pieces are used as lengthening parts, or an arrangement with a gland fixed to the bell-crank, and the rods clamped to it, is used. This is more satisfactory, as the rods can then be put in the full length and lengthened as sinking proceeds.

The second method of employing a sinking pump is to lower the whole column, either with iron rods and screws, or by ropes worked from a steam winch at the surface, as the work proceeds. A combination of both of these methods is possibly the best and safest. Fig. 567 shows the arrangement of lowering a sinking set with ropes and

ground spears. The pipe column *a* is fixed rigidly to the suction piece *b*, and between the suction piece and the clack piece a short pipe *h* is inserted, having extra broad and strong flanges. Immediately below the flanges, two strong iron glands *c c* are fixed and connected to the 'ground spears' *d d*; at the top of these spears are two sheaves *e e*, connected to the spears with strapping plates. At

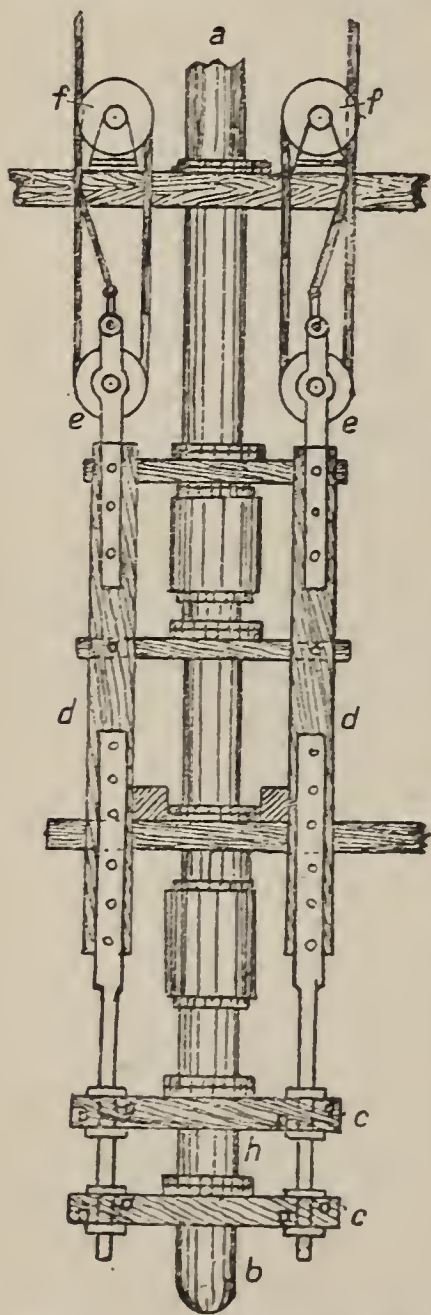


FIG. 567.—Sinking pump with lowering gear.

the surface two similar sheaves *f f* are fixed, round which the ropes work. They are operated by a steam winch at the surface, so that the whole lift can be lowered as sinking proceeds, and fresh pipes added at the top as required. The spears may be made of sections of pitch pine 4 in. to 8 in. square, according to the size of lift, or they may be of wrought iron 2 in. to 4 in. diameter. They ought to be well strengthened at frequent intervals with cross glands.

A third arrangement is to raise the water, while sinking proceeds, by means of a steam pump slung in the shaft, and connected to lowering screws of wrought iron which work through beams at the surface, with a large nut operated by spanners. As an additional precaution, a strong rope, operated by a steam winch at the surface, should also be connected to the pump, it being likewise useful for lowering the pump or other parts when pieces are being added to the screws. This method of pumping has much to commend it, as it does away with the necessity for using pump-rods and bell-cranks. The only difficulty is that a good deal of space is required if the quantity of water to be dealt with is large, as the engine has to be correspondingly large. Good strong tackle must be used, as the weight of pump and connections alone often amounts to between 12 and

18 tons. In this method the whole contrivance may be lowered as sinking proceeds, or the pipes may be fixed and the pump lowered alone, the pipes immediately above it being cut as required. This latter method is probably the safest and best in most cases. The usual method of arranging the pumps in the shaft is to make the sinking or bottom lift a bucket lift, until a point has been reached where it is proposed to put in a permanent lift. A plunger lift may then be put in, and the sinking can proceed as before, with the

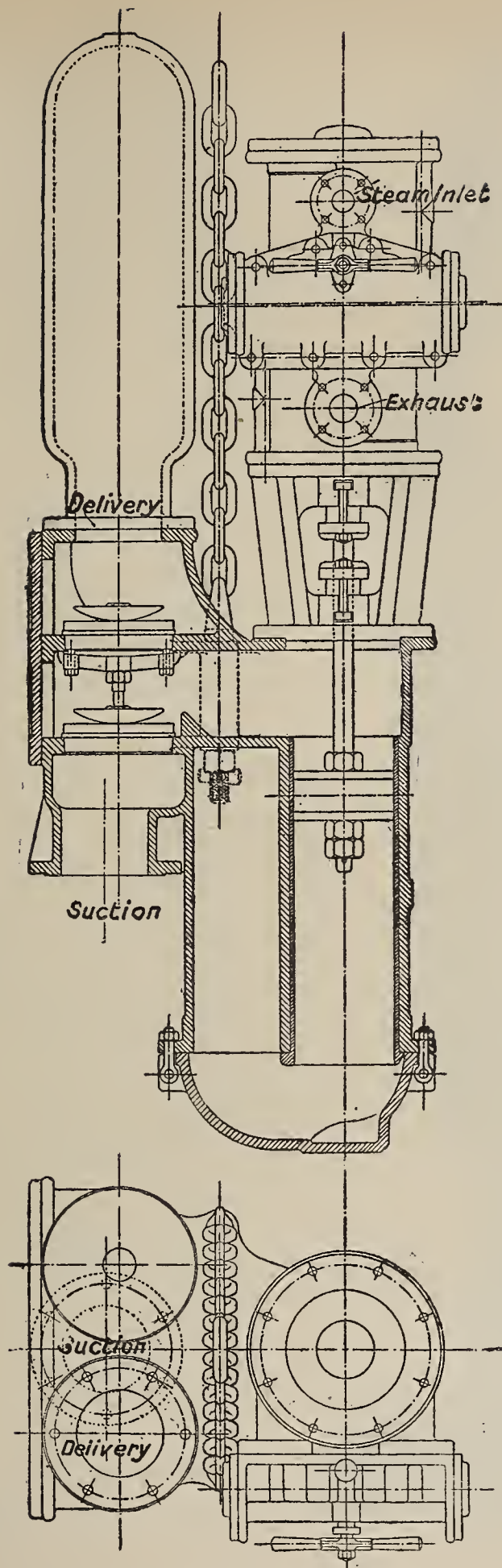


FIG. 568.—Evans' sinking vertical Cornish steam pump, with removable cast-iron liner.

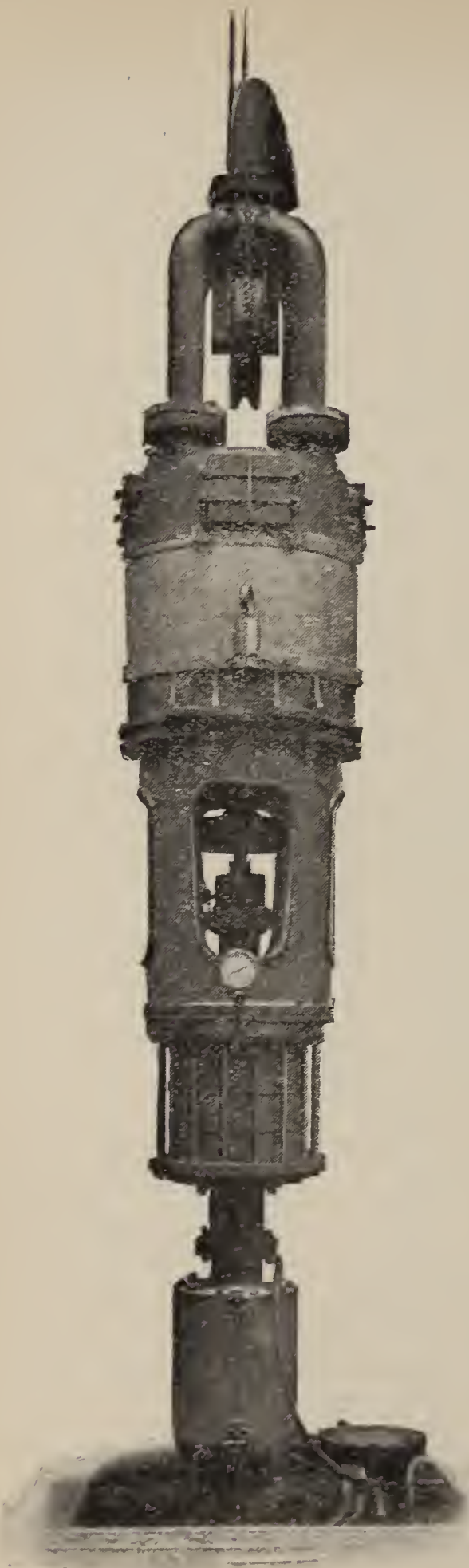


FIG. 569.—Mather & Platt's electric vertical sinking pump.

bucket set. In some instances plunger pumps are used until the lowest part of the sinking is reached, when a bucket lift is substituted with which to complete the sinking. The great advantage of the bucket lift is that, as has been seen, it requires very little space in the shaft, as the rods work inside the pipes.

The arrangement of sinking pumps adopted by Messrs Joseph Evans & Sons of Wolverhampton is a very suitable and handy method of dealing with water in sinking pits, and has been found to work satisfactorily.

The pump itself (see fig. 568) is what is known as a vertical 'Cornish' sinking pump, is double-acting, of the outside packed ram type fitted with differential ram, and is suitable for heads up to 300 ft. Pumps of this type were employed at the two shafts sunk by the Niddrie and Benhar Coal Co., Ltd., at Olive Bank, Musselburgh, near Edinburgh, where a large quantity of water was met with in the sinkings. The following were the sizes of pumps used at these pits:—two with 14-in. diameter steam cylinders, 9-in. water cylinder with 2 ft. stroke, each capable of delivering 275 gallons per minute at a speed of 100 ft. per minute; one 16-in. diameter steam cylinder, 9-in. water cylinder, and delivering 275 gallons per minute; two 24-in. diameter steam cylinders, 12-in. water cylinders by 2 ft. stroke, each capable of delivering 490 gallons per minute at a speed of 100 ft. per minute. Each pump was required to

deliver the water to a height of 300 ft., with a steam pressure of 100 lbs. per square inch at the boilers. The pumps were each fitted with barrels with removable liners, so that in the case of the barrels becoming worn or scored by dirty or gritty water, they could be withdrawn and rebored without taking the pumps out of the shaft. These pumps worked with great smoothness, and were admirably suited for the work they had to do, for in this case it would have been almost impossible to have used pumps operated by the ordinary method of pump-rods and bell-cranks, owing to the uncertain nature of the surface and the subsidence which took place.

Electric Sinking Pumps.—Sinking pumps operated by electricity are now used to some extent. Fig. 569 shows a vertical type of this class of pump, by Messrs Mather & Platt, Ltd., for a capacity of 500 gallons per minute and a maximum head of 300 ft. The motor is of the totally enclosed type, four-pole, and wound for 500 volts at 1150 revolutions per minute.

The connection between pump and motor is by flexible coupling; ball bearing at the top of motor case to take the weight of armature and spindle; and ball bearing just above the pump to take weight of impellers and spindle.

Special arrangements are provided to allow of the pump going 'on blast,' i.e. of continuing to run while lifted clear of the water. The distance piece between pump and motor forms a fourfold passage way for discharge of water from the pump; also to minimise the space occupied in the shaft. This fourfold water discharge pipe is carried up between the motor poles, whereby not only is much room saved, but the pipes are protected from damage in sinking and blasting operations, and the motor is kept cool.

The fourfold discharge pipe is combined in one junction pipe, which also carries the sheave round which the wire rope is carried to raise or lower the pump in the shaft.

It occupies a small amount of room, is of moderate weight, and is generally compact and handy.

Special precautions are taken to ensure no damage occurring to any portion of the pump from blasting operations.

Air-Vessels.—Every pump in which the plunger or bucket moves at a greater speed than 40 ft. per minute, and in which the area of the pipes is not larger than that of the plunger or bucket, will work better and more smoothly with an air-vessel than without one. The primary object of an air-vessel is to reduce the shocks that are liable to occur inside the pumps and pipes, especially with plunger pumps, and thereby equalise the pressure necessary to force the water up the delivery pipes. When an air-vessel is connected with the pumps, a greater velocity can be obtained, either with plunger or bucket, with the same degree of safety, thereby increasing the capacity of the pumps.

The benefits to be derived from air-vessels on pumps are much

questioned by many engineers, some averring that they are of no service, or that the pump does better without them. If the air-vessel be badly placed, or badly attended to, in not being charged properly, its presence will doubtless be a hindrance to the efficient working of the pump. The nearer the air-vessel is to the casing or barrel the better. If only one is used, it should be placed immediately above the discharge clack on the delivery side. In slow-moving pumps, air-vessels are not actually required. To get the maximum benefit from air-vessels they must be well looked after and kept constantly charged with air. In one of the best forms of air-vessel for a vertical column of pipes, the rising column is enlarged and an inside pipe is brought down near to the bottom of the chamber. Immediately below this pipe is fixed a cup, which tends to divert all the particles of air flowing with the water into the chamber.

To charge the air-chamber, the tank is connected to the rising column by a small pipe with a tap to it. At the other end of the tank are two small pipes for ingress and egress. The tank is first filled with water by opening the tap and keeping the other pipes closed. When it is filled, the tap is closed and the pipes opened, when the water rushes out at, and air enters by, the pipe and rises from the tank into the air-chamber.

An important point in connection with air-vessels is to secure their being of sufficient area. This should be four to six times the area of the working barrel, according to the speed of pump.

The capacity of air-chamber necessary will depend upon the type of pump used, single-acting pumps requiring much larger chambers than double-acting ones. Chambers made of cast iron should be well tested for tightness under full pressure; they ought also to be provided with pressure-gauge glasses to show the water level, or, if the pressures are high, a series of 'try' cocks should be fixed, for the purpose of ascertaining the position of the air in the chamber.

'Duty' of Pumping Engines.—The duty or efficiency of a pumping engine is measured by the number of foot-pounds of work performed per cwt. of coal consumed. It varies greatly with the type of engine and the circumstances under which it works.

The duty of pumping engines was first recorded in Cornwall by Watt in connection with engines which he erected at some of the mines there. Naturally, in such a district, where the price of fuel is high, there would be a desire to get the largest amount of work possible for a given coal consumption. The highest efficiency ever obtained was from a Cornish pumping engine which gave 146 million foot-pounds per cwt. of coal burned. After allowing for friction, this corresponds to a consumption of 1.21 lbs. of coal per hour per indicated horse-power, which is very high efficiency

indeed, as, with ordinary colliery pumping engines, it is not unusual to find a consumption of 10 to 15 lbs. of coal per horse-power per hour, and in some types of engines a great deal more. The 'duty,' as defined above, includes the efficiency of the boilers and engine, and depends a good deal on the quality of the fuel burned.

A fair comparison of pumping engines may be made, and the efficiency ascertained by testing the consumption of coal of average quality in raising the steam required, and may be calculated from the formula :

$$U = \frac{(H \times d^2) \times 2.045 \times N \times L}{W},$$

where H is the height of lift of pump in fathoms; d the diameter of pumps in inches; L the length of stroke; N the number of strokes in one month; W the weight of water in lbs.; U the units of work per lb. of steam (duty).

The test to ascertain the efficiency is generally made when the ordinary work of the mine is suspended.

The table gives the result of the 'duty' performed by various pumping engines from actual tests made by Mr Dugald Baird on the foregoing basis.*

Class of Engine.	Position of Engine.	Diameter of Cylinder in in.	Length of Stroke.	Depth of Lift in Yds.	Duty
a. Bull	Surface	100	12 ft.	300	50,265,312
b. Davey differential	Underground	36	4 "	300	41,360,000
c. Davey differential-compound	Surface	{ 33-in. H.P.C. } { 52 " L.P.C. }	{ 9 " } { 12 " }	200	69,705,890
d. Duplex differential	Underground	22	2 "	300	41,360,000
e. Compound Bull	"	{ 28 " H.P.C. } { 48 " L.P.C. }	12 "	168	35,827,200
f. Compound Davey differential	..	{ 34 " H.P.C. } { 64 " L.P.C. }	7 " 2½ in.	..	100,800,000

a, b and d, Leven Colliery; c and f, Davey's; e, Wellsgreen.

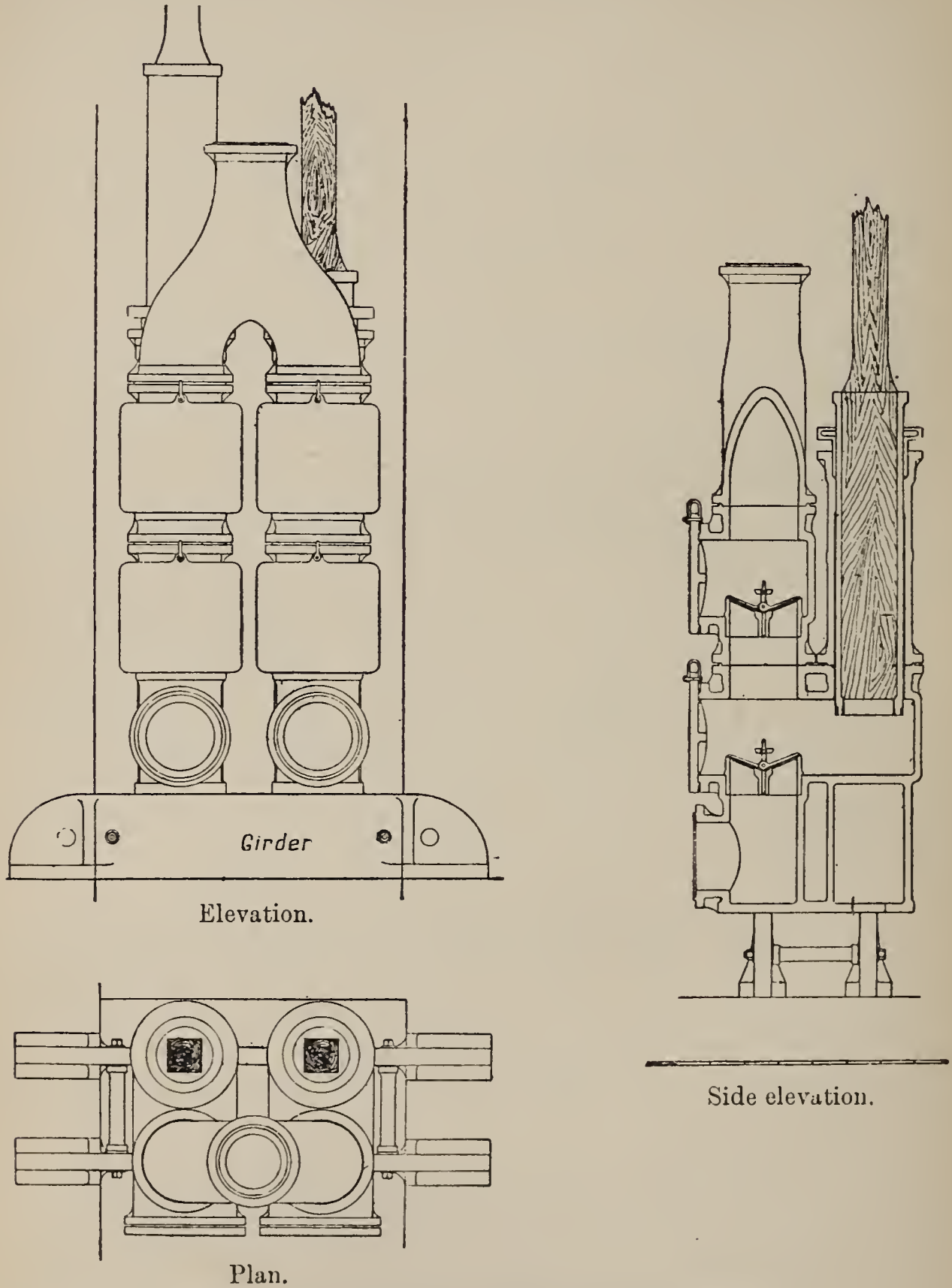
Mr Henry Davey states that the greatest efficiency of double-acting rotative pumping engines does not exceed 30 to 40 million foot-lbs. per cwt. of coal burned.

Arrangement of Pumps.—As already stated, it is usual to make the bottom lift in the shaft a bucket pump, and the other lifts forcing or plunger sets, which arrangement is found to work satisfactorily in most cases.

Many, however, prefer to have all the lifts as forcing sets, especially if there is no likelihood of the lower lift being flooded by a sudden inflow. Where pumps are operated with rods in the shaft, it is almost an invariable custom at coal mines to work the rods with a pair of bell-cranks, unless they are worked by a direct-acting engine, such as a Bull engine. By using double bell-cranks the power is more evenly distributed, and where a number of lifts have to be operated, fewer offsets are required than if the pumps

* *Trans. F. Inst. Min. Eng.*, vol. xi. p. 100.

were all worked off a single bell-crank. Figs. 571, 572 show front and side elevations of a good arrangement for working two or



FIGS. 570, 571, and 572.—Double plunger pump.

more lifts by a pair of bell-cranks. The features which commend this arrangement are that the pump is a double-acting one, and that there are two distinct sets of rods carried down from the surface,

the lower lift in each case being worked by a cross-head on the rods to the lift above.

When the sets are large, *i.e.* above 20 in. diameter, it is often a difficult matter to fix on a suitable arrangement to occupy as little space as possible. Figs. 570, 571, 572 show a plan, elevation, and side elevation, respectively, of a very compact arrangement for a double plunger set of 24 in. diameter, with one central delivery column common to both pumps. In this disposition of the pumps as little space as possible is taken up, and it is in every way convenient. The foundations for the pumps are generally strong beams of timber built into the shaft, or wrought-iron or steel girders, the latter being much to be preferred, as they give the maximum of strength with a minimum of space, and are less liable to decay than timber supports.

Cost of Pumping.—It would be interesting to have a statement of the actual cost of raising water for the many various types of pumps in use in mines, but so far as the writer is aware no complete figures of such have ever been published. Were they available, it might be a better guide to the mining engineer in selecting a type of pump than most of the other bases laid down. In the South Staffordshire district, where central pumping stations have been erected, and where the cost is distributed over the various collieries benefited on an output basis, a careful record has been kept of the actual cost of operating two large pumping plants, of which the following are the particulars:—

Bradley Engine.—H.P. cylinder 52 in. diameter; L.P. cylinder 90 in. diameter; each having a stroke of 10 ft. Two plungers are worked in the shaft, each 27 in. diameter by 10 ft. stroke; average piston speed 61 ft. per minute. The water pumped is equal to 2,462,800 gallons per day, and the following is the annual cost:—

	Per Annum.	Per Continuous Pump H.P. per Annum.
Coal,*	£1422	£7 2 11
Labour, ...	873	4 7 10
Stores, including oil, packing, etc., . . .	99	0 10 0
Repairs to engine and pumps, . . .	110	0 11 1
Repairs to boilers, ...	47	0 14 10
Total, ...	<u>£2551</u>	<u>£12 16 8</u>

Moat Engine.—H.P. cylinder 44 in. diameter; L.P. cylinder 76 in. diameter; stroke 10 ft. Works two plungers 19 in. diameter and below a pair of bucket pumps 19½ in. diameter; stroke in pumps 10 ft. Total lift 620 ft.; average piston speed 115 ft. per minute. Average quantity of water pumped per day 2,025,100 gallons.

* Small coal used of an average value of 4s. 11·25d. per ton.

Average Cost per Annum.		Per Continuous Pump	
Per Annum.		H. P. per Annum.	
Coal, ...	£1410	£5	6 10
Labour, ...	918	3	9 6
Stores, including oil, packing, etc., .	98	0	7 5
Repairs to engine and pump, ...	150	0	11 5
Repairs to boilers, ...	23	0	1 9
Total, ...	<u>£2599</u>	<u>£9</u>	<u>16 11</u>

At Short Mountain Colliery, Pa., U.S.A., 207,034,324 gallons of water were pumped in ninety-two days, approximately $2\frac{1}{4}$ gallons per day, from a depth of 1152 ft. The average quantity of coal burned was equal to 0·035 ton per 1000 gallons raised, equal to a cost of 2·78d. per 1000 gallons. At the Leigh Coal Co's. pumping plant in the same district the cost was approximately 1d. per 1000 gallons from a depth of 560 ft. Both of these costs seem to be for the cost of fuel only.

A few worked-out examples on pumping, such as are often set at examinations, are given below, in the hope that they may prove useful.

Question.—How many strokes per minute can be made by the piston of a pump whose area is 2 sq. ft., length of stroke 5 ft., and the height to which the water is raised 60 fms., driven by an engine of 80 horse-power?

$$\text{H.P.} \times 33000 = A \times L \times 62\cdot5 \times d \times x.$$

When H. P. = horse power, A = area of pump in sq. ft., L = length of stroke in feet, 62·5 = number of lbs. in 1 cub. ft. of water, d = height in feet water is raised, x = the number of strokes required per minute.

$$\therefore 80 \times 33000 = 2 \times 5 \times 62\cdot5 \times 360 \times x$$

$$\text{after cancelling } 880 = 75x \quad \therefore x = \frac{880}{75} = 11\cdot86.$$

Question.—At what rate will it be necessary to work a pump 12 in. diameter with $4\frac{1}{2}$ ft. stroke, to deal with 200 gallons of water per minute in a shaft 200 yds. deep, and what is the approximate horse-power required?

Using the same letters as above, and G = gallons of water per minute, and D = diameter of pump in inches, 6·25 = number of gallons in 1 cub. ft.

Allowing 10 per cent. of loss for slip, $G = 200 + 20 = 220$.

$$\therefore G = \frac{6\cdot25 \times D^2 \times \cdot7854 \times L \times x}{114}, \text{ or calculation can be simplified, as } \frac{6\cdot25 \times \cdot7854}{144}$$

$$= \cdot034 \quad \therefore G = D^2 \times \cdot034 \times L \times x$$

$$220 = 12^2 \times \cdot034 \times 4\cdot5 \times x$$

$$220 = 22\cdot03 x$$

$$\therefore x = \frac{220}{22\cdot03} = 9\cdot16 \text{ strokes per minute.}$$

As 1 gallon of water is equal to 10 lbs. $\therefore$ H.P. $\times$ 33000 = $220 \times 10 \times 200 \times 3$
 or H.P. = $\frac{220 \times 10 \times 200 \times 3}{33000} = 40$

Question.—A hydraulic pump having an 8-in. diameter plunger is wrought by means of a head of water brought from the surface in pipes 2 in. diameter. Find the total pressure on plunger and weight of water in pipes, if the depth of the shaft is 360 ft.

Pressure in lbs. per sq. in. due to head of water = $\frac{d \times 12 \times 62.5}{1728}$, or as $\frac{12 \times 62.5}{1728}$
 = .434 $\therefore$ pressure in lbs. per sq. in. due to head of water = $d \times .434 = 360 \times .434$
 = 156.24 lbs.

Total pressure on plunger = area of plunger in sq. in. $\times$ pressure in lbs. per sq. in.
 = $8^2 \times .7854 \times 156.24 = 7852.62$ lbs.

Weight of water in pipes $\left\{ \begin{array}{l} = 156.24 \times 2 \times .7854 \\ = 490.51 \text{ lbs.} \end{array} \right.$

Question.—What number of gallons and cubic feet of water can be pumped per hour from a pit 600 feet deep, by an engine of 200 H.P., assuming the efficiency of the engine to be .6?

Let x = weight of water in lbs. raised per hour.
 Then $x \times d = \text{H.P.} \times 33000 \times 60 \times .6$

$$x \times 600 = 200 \times 33000 \times 60 \times .6$$

$$\text{or } x = \frac{200 \times 33000 \times 60 \times .6}{600}$$

$$= 396000 \text{ lbs.}$$

$$1 \text{ gallon of water} = 10 \text{ lbs. } \therefore \text{gallons per hour} = \frac{396000}{10} = 39600$$

$$1 \text{ cub. ft. of water} = 62.5 \text{ lbs. } \therefore \text{cub. ft. per hour} = \frac{396000}{62.5} = 6336$$

Question.—Find the quantity of water delivered by a double-acting plunger pump, if the plunger is 7 in. in diameter, length of stroke $4\frac{1}{2}$ ft., and working at 20 strokes per minute. Also find the horse-power if the shaft be 90 fms. deep.

Using the same notation as in Question 2, and also let N = number of strokes of plunger per minute,

$$\begin{aligned} \text{Then } G &= D^2 \times .034 \times L \times 2 \times N \\ &= 7^2 \times .034 \times 4.5 \times 2 \times 20 \\ &= 299.88 \text{ gallons per minute.} \end{aligned}$$

$$\text{H.P.} \times 33000 = G \times 10 \times d$$

$$\text{H.P.} \times 33000 = 299.88 \times 10 \times 90 \times 6.$$

$$\text{After cancelling H.P.} = \frac{299.88 \times 9}{55} = 49.07.$$

Question.—Give the principal sizes of a direct-acting pumping engine fitted with a fly-wheel which you would erect to raise 400 gallons of water per minute from a depth of 80 fms.; allowing $\frac{1}{3}$ for stoppages, $\frac{2}{3}$ for efficiency of engine, and 10 per cent. for slip of pumps.

As $\frac{1}{3}$ has to be allowed for stoppages, and 10 per cent. for slip of pumps,

$\therefore$ Gallons to be raised per minute $= \frac{400 \times 24}{16} + 10 \text{ per cent.} = 660.$

We would therefore require to provide a pump capable of raising 660 gallons per minute. Assume the speed of the pump to be 100 ft. per minute.

$G = D^2 \times .034 \times \text{speed}$
or $D = \sqrt{\frac{\text{gallons per minute}}{.034 \times \text{speed}}} = \sqrt{\frac{660}{.034 \times 100}} = 13.9, \text{ or } 14 \text{ in. approximately as}$
the diameter of pump required.

Then to calculate size of engine required to raise 600 gallons per minute, we may equate the work thus:—

Work done by engine = work done in shaft ;
or $D^2 \times .7854 \times P \times \text{speed} \times E = \text{weight of water in lbs. per minute} \times \text{height to be raised in feet.}$
Where $E = \text{efficiency of engine} = \frac{2}{3} \text{ or } .66.$

If we assume the effective steam pressure to be 50 lbs. per sq. in., and the speed of engine to be the same as the pump, 100 ft. per minute,

Then $D^2 \times .7854 \times 50 \times 100 \times .66 = 600 \times 10 \times 80 \times 6,$
after cancelling $.08639 D^2 = 96$

and $D = \sqrt{\frac{96}{.08639}} = 33.3 \text{ in. diameter of cylinder.}$

The engine is to be direct-acting, so that it could have a 5 ft. stroke, and run at the rate of ten double strokes per minute ; the diameter of cylinder being 33 in., and the steam pressure 50 lbs. per square in. Suppose the above engine to be compound and working expansively, and the steam to be cut off at $\frac{1}{3}$ of the stroke ; what would the diameter of the low-pressure cylinder require to be ?

The size of the low-pressure cylinder may be found by the formula :

$a = \frac{A}{\sqrt{E}}$: Where $a = \text{area in sq. in. of high-pressure cylinder}$

- $A =$,, ,, low ,, ,,
- $E = \text{number of expansions of steam in cylinder} = \frac{P}{T}$
- $P = \text{mean effective steam pressure in lbs. per sq. in.}$
- $T = \text{terminal pressure of steam} = P \frac{l}{l_1}$
- $l = \text{length of stroke before steam is cut off}$
- $l_1 =$,, ,, after ,, ,,

Here $T = 50 \times \frac{20}{60} = 16.6$ lbs., and $E = \frac{50}{16.6} = 3$, and by formula $\alpha = \frac{A}{\sqrt{E}}$, or if

we let d = diameter of high pressure cylinder
and D = „ low „ „

$$\text{then } d^2 \times .7854 = \frac{D^2 \times .7854}{\sqrt{E}}$$

$$\therefore D^2 \times .7854 = d^2 \times .7854 \times \sqrt{E}$$

$$D = \sqrt{\frac{33^2 \times .7854 \times \sqrt{3}}{.7854}} = \sqrt{1883.97} = 43.4 \text{ in.}$$

If we allow the same efficiency for this cylinder as for the high-pressure one, then its diameter would require to be $43.4 + \frac{1}{3} = 57.8$ in. The ratio between the two cylinders is often taken as 1 : 1.6 or 1.5 when the number of expansions is less than 10.

CHAPTER XIII.

VENTILATION.

Gases Present in Mines.—The principal causes of impure air in mines are :—The exhalations of men and animals ; burning lamps or candles ; gases given off naturally from the strata and those resulting from blasting ; underground fires ; decaying timber in the workings ; absorption of oxygen by chemical agencies ; introduction of foreign substances.

In breathing, oxygen is withdrawn from the air and CO_2 is given off, together with a certain percentage of nitrogen ; a man working for eight hours will give off, on an average, over 5 cub. ft. of CO_2 . The quantity of air inhaled by a man is said to be .42 to .45 cub. ft. per minute, or 25.2 to 27 cub. ft. per hour when at rest ; but in a mine it will require much more than this to keep the air pure, possibly 100 to 120 cub. ft. per minute in a non-fiery mine, and in fiery mines, where much gas is given off, 300 to 400 cub. ft. per minute, while every horse in the mine will require three to six times as much air as a man. The quantity of air supplied to a colliery should bear some ratio to the amount of coal raised per shift, because the more actively the working proceeds the greater will be the amount of gas liberated. In non-fiery pits 80 to 100 cub. ft. of air per minute per ton of coal raised should suffice, and 100 to 200 cub. ft. per minute per ton of coal raised in fiery collieries. No hard-and-fast line can be drawn that will suit every case.

Burning lamps or candles also absorb the oxygen from the air and give off deleterious gases, principally CO_2 and small quantities of CO.

Fire-damp and choke-damp are given off more or less freely in nearly all mines, and are often the principal causes of impure air in the workings.

Timber in some mines, especially in return airways, decays very rapidly and pollutes the atmosphere. To prevent this as much as possible, the bark should be peeled off and the timber thoroughly seasoned before being used underground.

Chemical agencies, such as the action of water on iron pyrites or other ferruginous minerals, absorb considerable quantities of oxygen, and give off H_2S . The coal itself absorbs the oxygen of the air to

such an extent as sometimes to ignite spontaneously, and dangerous quantities of carbon monoxide gas are then given off.

The introduction of foreign substances into the air takes place on blasting by explosives, and fine coal-dust held in suspension, produced by breaking down the coal, or carried down into the workings with the intake air-current from the screens on the surface. The statement by manufacturers that explosives cause no fumes is wrong, as nearly all explosives give off CO and CO₂ in varying quantities, and also a considerable amount of solid residue. The smoke of gunpowder is largely composed of fine particles of carbonate and sulphide of potassium. Dynamite, when exploded, sends into the air, in a finely divided state, the 25 per cent. of infusorial earth which it contains. Boring shot-holes either in the rock or coal also sets in motion considerable quantities of fine dust which help to pollute the air.

From these causes, the atmosphere in the workings is very soon rendered impure and dangerous to breathe, unless means are adopted to clear them, and keep men and animals supplied with pure air. The only safe and practical method is to keep up a current of air of sufficient quantity and velocity as to carry off the deleterious gases as soon as formed.

As the symbols, specific gravities, and atomic or relative weights of the gases met with in mines are frequently referred to, it will be as well to commence by defining these terms.

The Specific Gravity or Relative Density of a substance is the weight of a given volume of that substance as compared with the weight of the same volume of a standard substance under the same conditions of temperature and pressure, or, briefly, the relative weights of equal volumes of gases in terms of a given unit are known as their specific gravities or relative densities.

$$\text{Relative density} = \frac{\text{Weight of gas}}{\text{Weight of same volume of standard gas at same temperature and pressure.}}$$

Air is nearly always the standard adopted when comparing the specific gravity of compound gases.

Example.—What is the specific gravity of unit volume of carbon dioxide compared with unit volume of air?

C	= 12 × 1 = 12	N ₂	= 14 × 4 = 56
O ₂	= 16 × 2 = 32	O	= 16 × 1 = 16
2 vols.		5 vols.	
44		72	
$\therefore \text{sp. gr. of CO}_2 = \frac{\frac{44}{2}}{\frac{72}{5}} = \frac{220}{144} = 1.52$			

When comparing the relative densities of elementary gases hydrogen is usually taken as the standard.

Molecular Weight.—The ratio that exists between the weight of a gaseous molecule and *half* the weight of a molecule of hydrogen is termed the *molecular weight* of that gas; hence the number which represents the molecular weight of a gas is double that of its density or specific gravity. If, therefore, it is desired to find the molecular weight of a given gas, it is only necessary to multiply its density by 14.43, which gives its density as compared with hydrogen, and to double the number so obtained. (Relative densities of air and hydrogen are as 1 : 0.0693.)

Thus, as already shown, the specific gravity of $\text{CO}_2 = 1.52$; $\therefore (1.52 \times 14.43) \times 2 = 43.87$, or, approximately, 44 = molecular weight of CO_2 .

The Atomic Weight is the lowest proportion by weight of an element which can combine *chemically* with one part, by weight, of hydrogen, or the atomic weight is the smallest weight of an element ever formed in two unit volumes of any gas. Thus an atom of oxygen is very nearly 16 times (exactly 15.96) that of an atom of hydrogen; therefore this number (16) is termed the atomic weight of oxygen. The atomic weight is also sometimes spoken of as the *combining weight*.

Symbols.—The chemist divides all substances into elements, compounds, and mixtures. Of the former there are between sixty and seventy, and for the sake of convenience and brevity in referring to them, the first letter only, or two distinctive letters of the (generally) English names, are used. Thus H is the symbol for hydrogen, O for oxygen, etc., etc.

Properties of Gases.—Gases have certain properties which are of importance in connection with the ventilation of mines, and which ought to be carefully considered. These properties may be considered under (1) relation of gases due to the effect of heat (temperature); (2) relation of gases due to pressure (barometric); (3) diffusion of gases.

(1) Nearly all substances expand when heated and contract when cooled, and gases follow this law. The law which governs the expansion or contraction of gases is known as Charles' law, which is as follows:—The volume which a gas occupies varies *directly* as the *absolute* temperature, or *all gases expand by nearly the same amount in being heated from one temperature to another*. (The absolute temperature = 273°C. , or 459°F. + the observed temperature.)

The increase in bulk when 1 volume of a gas is heated or raised in temperature from 0° to 1°C. , or from 32° to 33°F. , is termed the *coefficient of expansion*, and is represented by the fraction $\frac{1}{273}$

(0.00366) for $1^\circ \text{Centigrade}$, and $\frac{1}{459}$ (0.002118) for $1^\circ \text{Fahrenheit}$.

Taking the fraction $\frac{1}{459}$ for the coefficient—

$$\begin{array}{lcl}
 1 \text{ vol. of gas at } 32^{\circ} \text{ F. becomes } 1 + \frac{1}{459} \text{ vols. at } 33^{\circ} \text{ F.} \\
 1 \quad , , \quad , , \quad 32^{\circ} \text{ F.} \quad , , \quad 1 + \frac{2}{459} \quad , , \quad 34^{\circ} \text{ F.} \\
 1 \quad , , \quad , , \quad 32^{\circ} \text{ F.} \quad , , \quad 1 + \frac{459}{459} \quad , , \quad 491^{\circ} \text{ F. } (459 + 32) \\
 \text{or } 1 \quad , , \quad , , \quad t^{\circ} \text{ F.} \quad , , \quad 1 + t_1^{\circ} \quad , , \quad t_2^{\circ} \text{ F.}
 \end{array}$$

Charles' law is also expressed by the formula:—

$$v = V \frac{T_2}{T_1}, \text{ or } v = V \frac{459 + t_2}{459 + t_1},$$

where V = given vol. of gas at absolute temp. $T_1 = t_1 + 459$
 v = new $, , ,$ new $, , ,$ $T_2 = t_2 + 459$
 $t_1 + t_2$ = observed temperatures.

Example.—What would be the increase in volume of 2000 cub. ft. of a gas if the temperature is raised from 60° F. to 120° F. ?

$$v = 2000 \frac{459 + 120}{459 + 60} = 2000 \frac{579}{519} = 2231.21 \text{ cub. ft.}$$

$$\therefore \text{ the increase in volume} = 2231.21 - 2000 = 231.21 \text{ cub. ft.}$$

(2) The volume which any given quantity of a gas occupies is intimately related to the pressure under which it exists, and the variations in volume produced by changes of pressure are practically independent of the nature of the gas. The law expressing these variations is known as Boyle's law, which may be stated thus:—The volume of a given mass of gas, of which *the temperature is kept constant*, varies *inversely* as the pressure. If the original pressure is doubled, the volume is diminished to one-half; if it is trebled, the volume is reduced to one-third; if the pressure is halved, the volume is doubled, and so on.

This law is expressed by the formula, $v = V \frac{p_1}{p_2}$,

where V = given volume of gas at a pressure p_1
 v = new $, , ,$ new $, , ,$ p_2 .

Example.—What would be the alteration in 5000 volumes of a gas if the barometric pressure was reduced from 30.75 to 28.50 in. ?

$$v = 5000 \left(\frac{30.75}{28.50} \right) = 5394.73 \text{ vols., or an increase of } 5394.73 - 5000 = 394.73 \text{ vols.}$$

If the temperature and pressure are both altered, then Charles' and Boyle's laws may be combined, and expressed thus:—

$$v = V \frac{459 + t_2}{459 + t_1} \times \frac{p_1}{p_2}.$$

Example.—If a volume of fire-damp (CH_4) at a temperature of 45° F. and a barometric pressure of 29.5 in. measures 3000 cub. ft., what would be the alteration in volume if the temperature were raised to 85° F and the barometer to 31.5 in. ?

$$\begin{aligned}
 v &= 3000 \frac{459 + 85}{459 + 45} \times \frac{29.5}{31.5} = 3000 \frac{544}{504} \times \frac{29.5}{31.5} = 3000 \frac{16048}{15876} \\
 &= 3032.5 \text{ cub. ft. ; or an increase in volume of } 3032.5 - 3000 = 32.5 \text{ cub. ft.}
 \end{aligned}$$

(3) *Diffusion of Gases*.—The property of a gas to pass into or mix with another gas is called the diffusion of gases. This diffusion may be verified by inverting a jar of air over a jar of carbon dioxide (CO_2); it will be found after the lapse of a few minutes that some of the CO_2 will have passed into the upper jar containing the air, and that some of the air will have passed down and mixed with the CO_2 , notwithstanding that the latter is $1\frac{1}{2}$ times heavier than air. This process goes on until there is a uniform mixture of carbon dioxide and air in both jars, and the two gases never separate again according to their densities. This property is expressed by Graham's law of diffusion, and is stated thus:—The relative velocities (or rate) of diffusion of any two gases are *inversely* proportional as the *square root of their densities*.

For instance, the density of hydrogen is 1, and oxygen = 16, and therefore the rate of diffusion of these two gases would be in the proportion of $\sqrt{16} : \sqrt{1}$, or 4 : 1; *i.e.* hydrogen will diffuse or mix four times faster than oxygen. This may also be expressed by the formula, $v = \sqrt{\frac{1}{d}}$,

where v = vol. of the gas which diffuses in the same time as 1 vol. of air.
 d = density of the gas.

In dealing with mine gases it is convenient to take their specific gravities : air = 1.

This law of diffusion is important, as it explains why fire-damp becomes explosive by getting mixed with air, or why it is never or very rarely found in a pure state in mines, but is always mixed with varying proportions of other gases. The same fact is noted with carbon dioxide, which is always found mixed with a proportion of free nitrogen.

Gases are divided into three classes, viz., elementary or simple gases, compound gases, mechanical mixtures. An elementary or simple gas consists of one element only, *i.e.* of a substance which it is impossible to split up or divide, or a gas whose molecules are composed of *similar* atoms is said to be an elementary gas, *e.g.* H, O, Cl, etc.

Compound gases are composed of two or more elements chemically combined with each other, or a gas whose molecules are composed of *dissimilar* atoms is called a compound gas, *e.g.* CO_2 , CO, CH_4 , etc. This combination results in the production of a gas differing in its properties from either of the elements of which it is composed.

Mechanical mixture takes place when two or more substances or elements are brought together and no chemical action results. The atmosphere of a mine is a mechanical mixture of air and the various gases and emanations described later.

The elementary or simple substances of which the compound gases found in mines are composed, are—Hydrogen, H; Oxygen, O; Nitrogen, N; Carbon, C; Sulphur, S.

Hydrogen.—Symbol, H ; atomic weight, 1 ; density 0.0693 (air = 1). An inflammable gas ; possessing, when pure, neither colour, taste, nor smell ; and a non-supporter of combustion or life. The fact may here be noted that all inflammable gases are non-supporters of combustion in the ordinary sense. When 2 volumes of hydrogen are mixed with 1 volume of oxygen, a violently explosive mixture is formed. Hydrogen being the lightest substance known, it is usually taken as the standard of atomic weight, the weight of all other gases being expressed in terms of hydrogen as unity. 1000 cub. ft. of hydrogen at 14.7 lbs. (atmospheric) pressure per sq. in., and at a temperature of 32° F., weigh 5.606 lbs.

Oxygen.—Symbol, O ; atomic weight, 16. Oxygen occurs in the free state in the atmosphere, mechanically mixed with about four times its volume of the inert gas, nitrogen, which acts as a diluent to the highly active oxygen. Oxygen has neither colour, taste, nor smell ; does not burn in air, but is the great supporter of combustion and life. All forms of burning, breathing, decay, etc., are simply manifestations of the combination of various substances with oxygen. Since 1000 cub. ft. of hydrogen weigh 5.606 lbs., 1000 cub. ft. of oxygen = $5.606 \times 16 = 89.69$ lbs.

Nitrogen.—Symbol, N ; atomic weight, 14 ; non-inflammable gas ; no colour, taste, or smell, and does not support combustion. It forms $\frac{4}{5}$ ths by volume of the atmosphere, but is altogether a very inert gas, being very inactive in all its qualities under ordinary conditions. While not actively poisonous, it is incapable of supporting life. 1000 cub. ft. of nitrogen = $5.606 \times 14 = 78.48$ lbs.

Carbon.—Symbol, C ; atomic weight, 12. Carbon is a solid, and it is never found free in a gaseous form like hydrogen or oxygen. Charcoal, coke, graphite, and the diamond are all forms of carbon, the diamond being the purest. This element is often present in compound gases, and, from their properties, gaseous carbon is assumed to have no colour, taste, or smell, to be inflammable, but a non-supporter of combustion.

Sulphur.—Symbol, S ; atomic weight, 32. Sulphur is also a solid element at ordinary temperatures ; at higher temperatures it becomes a liquid with a clear amber colour, which on continuous heating becomes darker, and at a temperature of 840° F. it becomes a dense red vapour which is combustible, and a non-supporter of combustion and life, without smell itself, but with a strong pungent smell if allowed to combine with oxygen or with hydrogen. Its combination with hydrogen constitutes its chief claim to importance as regards mine gases.

Air.—The only naturally occurring mechanical mixture we have to deal with in mine gases is air, which cannot be correctly expressed by any formula.

Air is composed approximately of 4 volumes of nitrogen and 1 volume of oxygen ($N_4 + O$), or, approximately, 21 per cent. of oxygen

and 79 per cent. of nitrogen, 14·43 being its relative weight as compared with hydrogen. In addition to these two gases, air also contains several other constituents, such as carbonic acid gas, water vapour, and *argon*, a constituent discovered by Lord Rayleigh. The proportion of carbonic acid gas (CO₂) in the air is about $\frac{1}{2500}$ th part of the whole volume, or varies from 2 to 10 vols. in 10,000 vols. of air.

The average composition of normal air is :—

	Volumes per 1000.
Nitrogen, ...	779·0600
Oxygen, ...	206·5940
Aqueous vapour, ...	14·0000
Carbon dioxide, ...	·3360
Ammonia, ...	·0080
Ozone, ...	·0015
Nitric acid, ...	·0005
	1000·0000

In addition to these constituents, air also contains small quantities of the rarer gases—argon, neon, helium, krypton, and xenon.

The average amount of CO₂ present in the air is ·04 per cent. ; in ordinary mines, 0·78 per cent. Professor Gray, of the Royal Technical College, Glasgow, found that from the analysis of forty-eight samples of air, taken in the return airways of Scotch collieries, the average percentage of CO₂ was 0·75.

Moisture in the Air.—There is always a certain amount of water vapour present in the atmosphere, but the quantity is subject to great variation. The barometer gives indications as to the condition of the atmosphere in this respect. The amount of vapour or moisture which the air can take up depends on its temperature : the higher the temperature the more water can be held in suspension. There is, for any given temperature and pressure, a maximum amount of moisture which a given volume of air is capable of taking up, and at which it is ‘saturated.’ The following quantities of water correspond to ‘saturation’ for the temperatures given :—

Degrees F.	Weight of water in lbs.
1000 cubic feet of air at 32° contain . .	0·308 lb.
„ „ 50° „ . .	0·567 „
„ „ 68° „ . .	1·066 „
„ „ 86° „ . .	1·873 „

When air saturated with vapour is cooled, the moisture is condensed and falls in the form of rain or dew.

The relative weight of water vapour to air is as 9 to 14 $\frac{3}{4}$. Water vapour is therefore much lighter than air, and a column of moist air is much lighter than a column of dry air of the same height. When the barometer falls it indicates a decrease in local pressure, because the air is moist and there is a probability of rain, whilst when the

barometer is high it indicates that the air is dry and that dry weather will occur.*

The absolute amount of vapour in the air can be most accurately ascertained by means of tubes containing chloride of calcium carefully weighed before and after a known volume of air has been passed through them; the amount of moisture present in the intake and also in the return can be thus ascertained, and the difference between these two quantities will be the amount of moisture absorbed

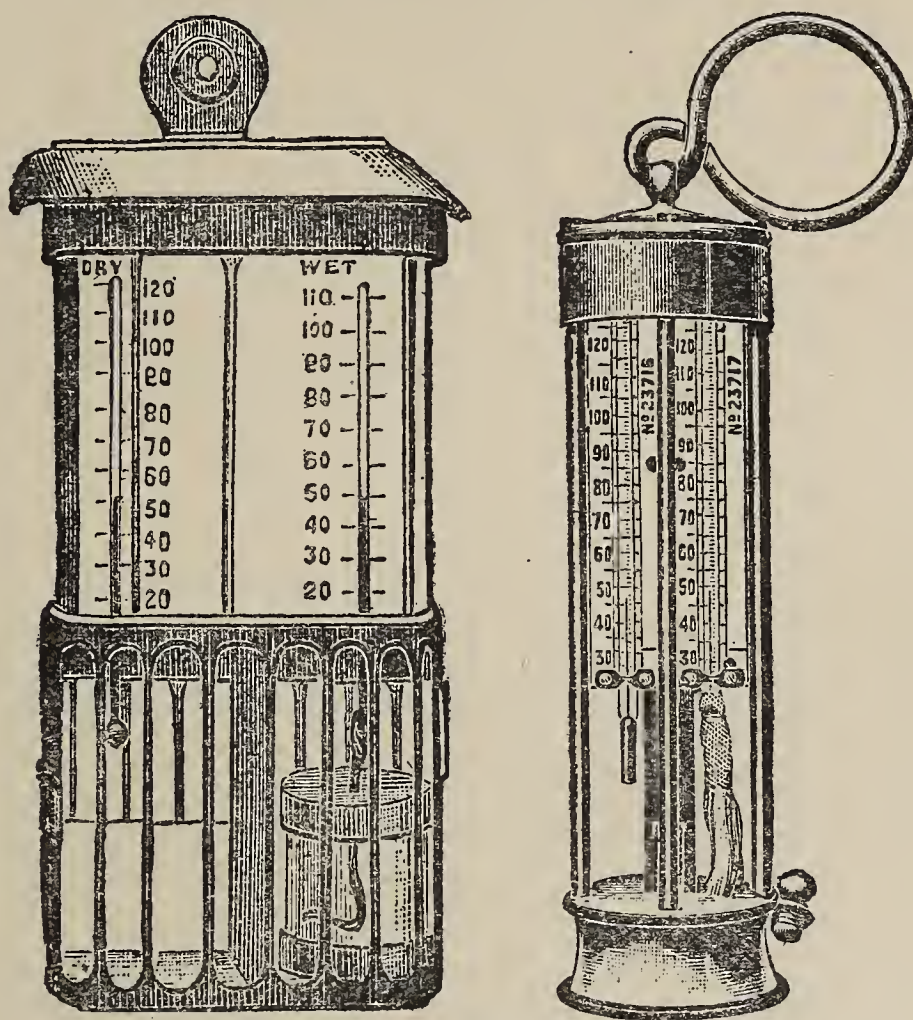


FIG. 573.—Mason's hygrometer.

from the underground workings. The most convenient method, however, for determining the amount of moisture in the air is by means of a hygrometer. The hygrometer consists of two thermometers placed side by side, known as the wet- and dry-bulb thermometers (fig. 573). One of these (the dry) thermometers has its bulb

* *Height of the Atmosphere.*—We are quite unable to tell to what height the atmosphere really extends, but we can readily estimate its height, from the observed pressure, if we assume it to have a uniform density.

The average pressure of the air at the sea-level is 14·7 lbs. per sq. in. at a temperature of 32° F. and 29·9 in. of mercury.

$14\cdot7 \times 144 = 2116\cdot8$ lbs. pressure per sq. ft., 1000 cub. ft. of air weighs 80·728 lbs. $\therefore 1$ cub. ft. = $\frac{80\cdot728}{1000} = \cdot08072$ lbs., and $\frac{2116\cdot8}{\cdot08072} = 26211$ ft. = height of air column if the atmosphere were of uniform density throughout.

exposed to the surrounding atmosphere, giving the temperature of the air, while the other (the wet) has a piece of muslin tied round the bulb, to which is attached a piece of absorbent wick. This wick is led into a small vessel containing rain or distilled water, an arrangement by which the muslin is kept constantly wet, the evaporation from the muslin and consequent cooling of the bulb being in proportion to the dryness of the air. The two thermometers will give different readings according to the amount of moisture in the air, the difference between the two being at a maximum when the air is

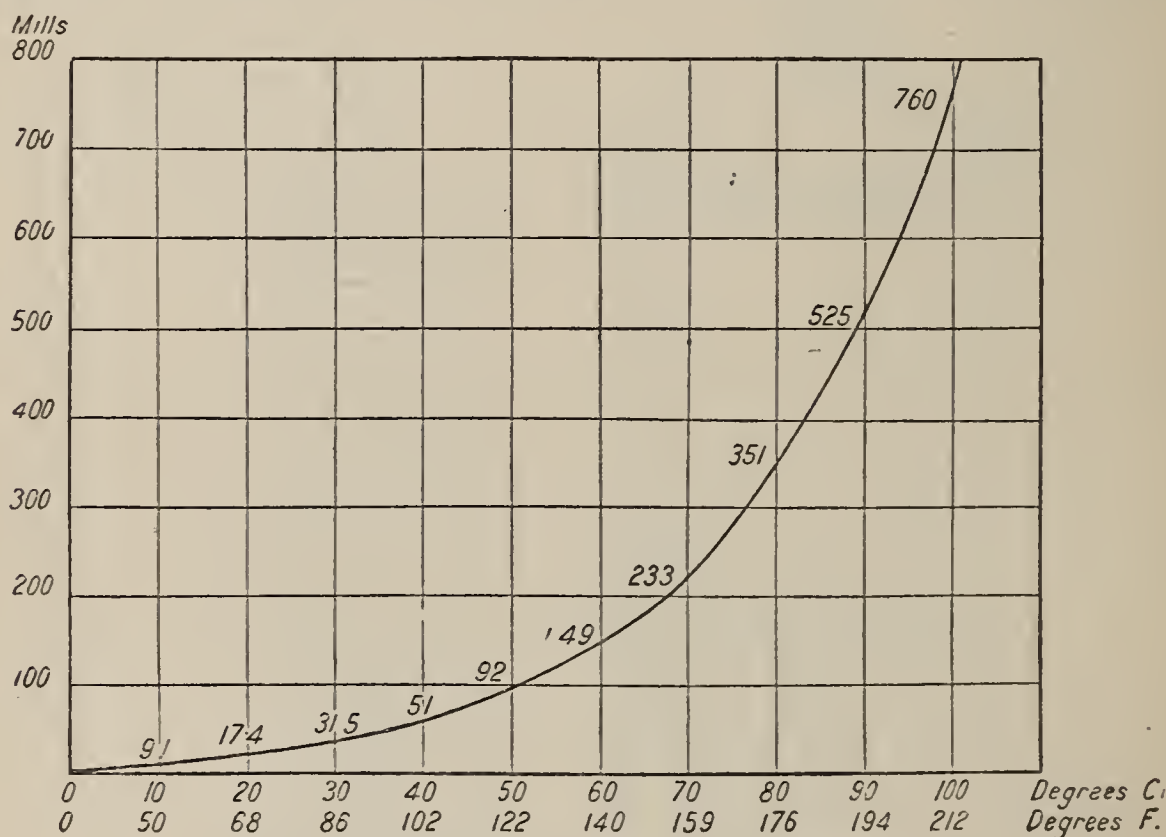


FIG. 574.—Curve of maximum vapour pressure.

If x = relative humidity of atmosphere per cent. actual vapour pressure = maximum vapour pressure,

f = maximum vapour pressure in mills at t° C.,

B = barometric pressure in mills,

then actual percentage of moisture present = $\frac{xf}{B}$.

x got from hygrometric.

f „ curve.

B „ barometer.

driest, and a minimum when the air is completely saturated with moisture. If the air is completely saturated, the readings of the wet and dry thermometers will be similar. When the air is charged to the limit of its capacity it is said to be saturated, and its 'relative humidity' is said to be 100 per cent. When its relative humidity is 75 per cent. it is said to hold 75 per cent. of the total amount of vapour that would be required to thoroughly saturate it. By the aid of tables the degree of saturation can be calculated.

Difference between the Readings of the Dry-Bulb and Wet-Bulb Thermometers.

Difference between the Readings of the Dry-Bulb and Wet-Bulb Thermometers.																						
Dry-Bulb Reading.	Dry-Bulb Reading.																					
	0°0	0°2	0°4	0°6	0°8	1°0	1°4	1°8	2°0	2°4	2°8	3°0	3°4	3°8	4°0	4°4	4°8	5°0	5°4	5°8	6°0	Fahr.
10°	100	92	85	78	72	67	57	48	44	38	32	30	25	20	18	14	11	10	9	8	7	10°
20	100	93	86	80	74	68	59	51	47	41	34	32	27	23	21	18	16	15	13	11	10	20
22	100	94	88	82	76	70	60	53	50	44	38	36	31	27	25	21	18	17	14	12	11	22
24	100	94	88	83	78	73	65	57	53	47	42	39	34	30	29	25	21	19	17	15	14	24
26	100	95	90	85	80	76	68	61	57	51	45	43	39	35	33	29	26	25	21	19	18	26
28	100	96	92	88	84	80	72	66	63	57	52	50	46	42	40	36	33	32	29	26	25	28
30	100	96	93	90	87	84	78	72	70	64	60	58	53	49	47	44	40	39	36	33	32	30
32	100	97	94	91	89	87	82	78	76	71	67	65	61	58	56	53	50	48	45	42	41	32
34	100	97	95	93	91	89	85	81	80	76	72	71	67	63	62	59	56	55	52	50	49	34
36	100	98	96	94	93	91	87	84	82	79	76	74	71	68	66	63	60	59	57	54	53	36
38	100	98	96	94	93	91	87	85	83	80	77	76	73	70	69	66	63	62	60	57	56	38
40	100	98	97	95	94	92	88	86	84	81	78	76	74	71	70	67	64	63	61	59	58	40
42	100	98	97	95	94	92	88	86	84	81	78	77	75	72	70	68	65	64	62	60	59	42
44	100	98	97	95	94	92	88	86	84	82	79	78	76	73	72	70	67	65	63	61	60	44
46	100	99	97	95	94	92	89	87	85	82	79	78	76	73	72	70	67	66	64	62	61	46
48	100	99	97	95	94	92	89	87	85	82	80	79	77	74	73	71	68	67	65	63	62	48
50	100	99	97	96	94	93	90	88	86	83	80	79	77	74	73	71	69	68	65	63	62	50
52	100	99	97	96	94	93	90	88	86	83	81	80	78	75	74	72	70	69	66	64	63	52
54	100	99	97	96	94	93	90	88	86	83	81	80	78	76	75	72	70	69	67	65	64	54
56	100	99	97	96	94	93	90	88	87	84	82	81	79	76	75	73	71	70	68	66	65	56
58	100	99	97	96	94	93	90	88	87	85	83	82	80	77	76	74	72	71	69	67	66	58
60	100	99	97	96	94	93	91	89	88	85	83	82	80	77	76	74	72	71	69	67	66	60
62	100	99	98	96	95	94	91	89	88	85	83	82	80	78	77	75	73	72	70	68	67	62
64	100	99	98	96	95	94	91	89	88	85	83	82	80	78	77	75	73	72	70	68	68	64
66	100	99	98	96	95	94	91	89	88	86	84	83	81	79	78	76	74	73	71	69	68	66
68	100	99	98	96	95	94	92	90	89	86	84	83	81	79	78	76	74	73	71	69	69	68
70	100	99	98	96	95	94	92	90	89	86	84	83	81	79	78	76	74	74	72	70	69	70
75	100	99	98	96	95	94	92	90	89	87	85	84	82	80	79	77	75	74	72	70	70	75
80	100	99	98	97	96	95	93	91	90	88	86	85	83	81	80	78	76	76	74	72	72	80

Read the dry- and wet-bulb temperatures carefully, and deduct one reading from the other.
Look in the column on the left or right for the nearest degree to the dry-bulb reading, carry the eye horizontally along until the column is reached corresponding to the difference between the dry- and wet-bulb thermometers, when the relative humidity will be found. Intermediate readings can be interpolated in the usual way.
Example.—Dry bulb 62·5, wet bulb 55·7; the difference is 6·8. Having found 62 in the column on the left or right, run the eye along the column until 6·8 is reached, when the relative humidity will be found, viz., 63 per cent.

Difference between the Readings of the Dry-Bulb and Wet-Bulb Thermometers.																				
Dry-Bulb Reading.	6°0	6°4	6°8	7°0	7°4	7°8	8°0	8°4	8°8	9°0	9°4	9°8	10°0	10°4	10°8	11°0	11°4	11°8	12°0	Dry-Bulb Reading.
Fahr.																				Fahr.
10°	7	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	10°
20	10	8	6	6	5	5	5	...	...	...	...	...	...	...	...	...	...	...	...	20
22	11	9	8	8	7	6	4	...	...	...	...	...	...	...	...	...	...	...	...	22
24	14	12	10	10	9	8	7	6	6	5	5	...	...	...	...	...	...	...	...	24
26	18	16	14	14	12	10	10	9	8	7	6	6	5	...	...	...	...	...	...	26
28	25	23	20	19	17	15	15	14	13	12	11	10	9	8	7	4	6	6	5	28
30	32	30	28	28	26	24	23	21	19	18	16	15	14	13	12	12	11	10	10	30
32	41	38	36	35	33	31	30	29	27	27	25	23	23	21	20	19	18	17	16	32
34	49	46	44	43	40	38	37	36	34	33	32	30	30	28	27	26	25	24	23	34
36	53	51	48	47	45	43	42	40	39	38	36	34	34	32	31	30	29	28	27	36
38	56	54	51	50	48	46	45	43	42	41	39	37	36	34	33	32	31	30	29	38
40	58	56	53	52	50	48	47	45	44	43	41	39	38	36	35	34	33	32	31	40
42	59	57	55	54	52	50	49	47	45	44	42	40	40	38	36	36	34	32	32	42
44	60	58	56	55	53	51	50	48	46	46	44	42	41	40	39	38	36	34	34	44
46	61	59	57	56	54	52	51	49	47	47	45	43	43	41	39	39	37	35	35	46
48	62	60	58	57	55	53	52	50	48	48	46	44	44	42	40	40	38	36	36	48
50	62	60	58	58	56	54	53	51	49	49	47	45	45	43	41	41	40	39	38	50
52	63	61	59	59	57	55	54	52	50	50	48	46	46	44	43	43	41	40	39	52
54	64	62	60	59	57	55	55	53	51	51	49	47	47	45	44	44	42	41	40	54
56	65	63	61	60	58	56	56	54	52	52	50	48	48	46	45	45	43	42	41	56
58	66	64	62	61	59	57	57	55	53	53	51	49	49	47	46	46	45	44	43	58
60	66	64	62	62	60	59	58	56	54	54	52	50	50	48	46	46	45	44	43	60
62	67	65	63	63	61	59	58	56	55	55	53	51	51	49	47	47	46	45	44	62
64	68	66	64	63	61	59	59	57	56	55	54	52	52	50	48	48	47	46	45	64
66	68	66	64	64	62	60	60	58	57	56	54	52	52	51	49	49	48	47	46	66
68	69	67	65	65	63	61	60	58	57	56	55	53	53	52	50	50	48	47	46	68
70	69	65	65	65	63	61	61	59	58	57	56	54	54	52	50	50	49	48	47	70
75	70	68	66	66	64	62	62	61	59	59	57	55	55	54	52	52	51	49	49	75
80	72	70	68	67	66	64	64	62	61	60	59	57	57	55	53	53	52	51	50	80

TABLE OF QUANTITY.

Temp. Fahr.	Weight in Grains	Temp. Fahr.	Weight in Grains.	Temp. Fahr.	Weight in Grains.
30	1.97	51	4.24	71	8.27
31	2.05	52	4.39	72	8.54
32	2.13	53	4.55	73	8.82
33	2.21	54	4.71	74	9.10
34	2.30	55	4.87	75	9.39
35	2.39	56	5.04	76	9.69
36	2.48	57	5.21	77	9.99
37	2.57	58	5.39	78	10.31
38	2.66	59	5.58	79	10.64
39	2.76	60	5.77	80	10.98
40	2.86	61	5.97	81	11.32
41	2.97	62	6.17	82	11.67
42	3.08	63	6.38	83	12.03
43	3.20	64	6.59	84	12.40
44	3.32	65	6.81	85	12.78
45	3.44	66	7.04	86	13.17
46	3.56	67	7.27	87	13.57
47	3.69	68	7.51	88	13.98
48	3.82	69	7.76	89	14.41
49	3.96	70	8.01	90	14.85
50	4.10				

∴ Intersection of 70° ordinate with
7° difference graph gives 64.16
= per cent. humidity.
∴ Saturation at 70°=8.01 grains
per cub. ft. (Table), and 64.16
per cent. of 8.01=5.14 grains
of moisture per cub. ft. at
given humidity.

Example :—
Dry bulb 70°
Wet bulb 63°
 —
Difference 7°

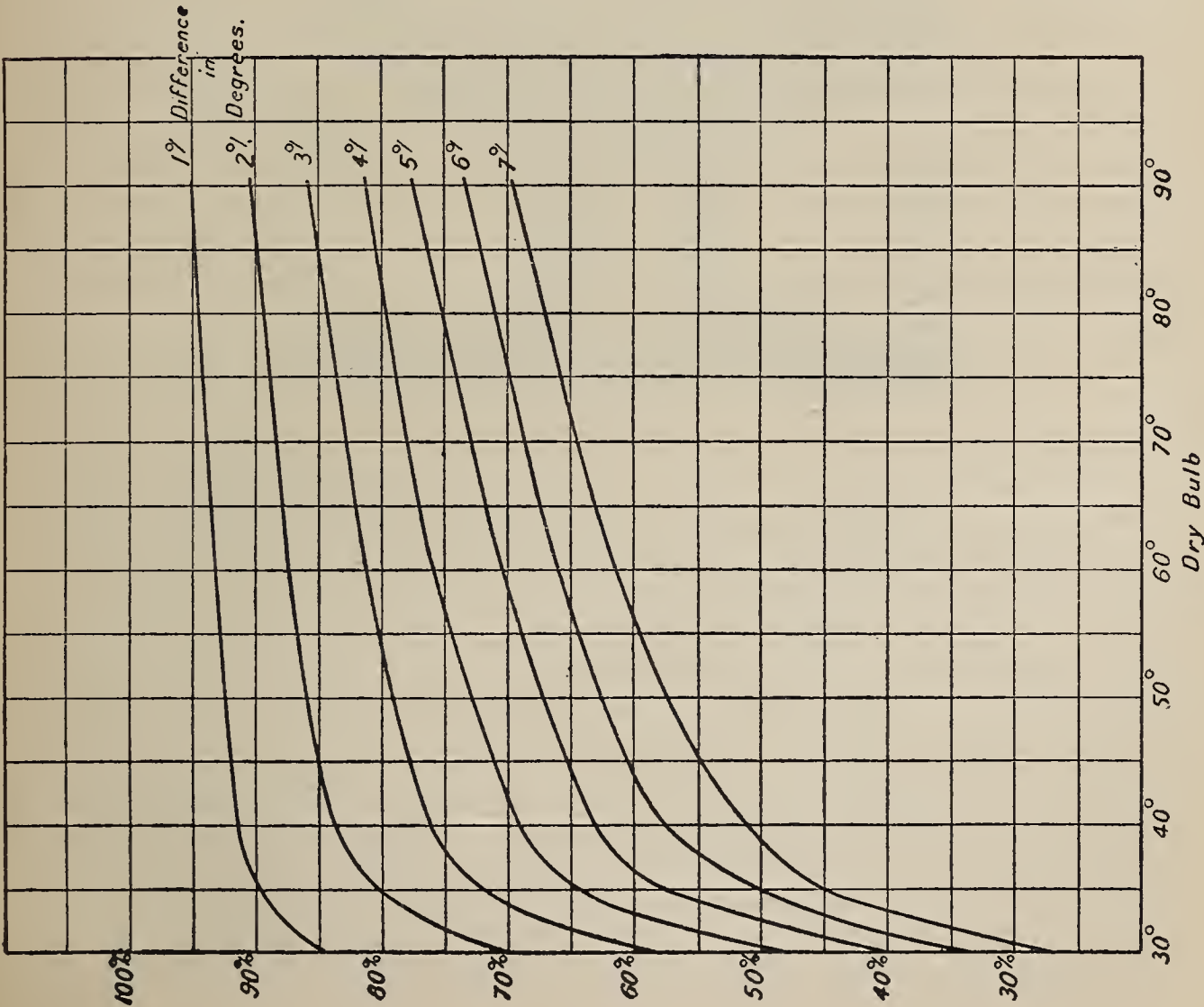


FIG. 575.—Diagram for humidity of air.

The amount of moisture in the air can also be readily ascertained by a diagram or graph, as shown in fig. 575.

The Hygrodeik.—In the methods just described the amount of saturation or relative humidity of the air requires to be ascertained by the aid of hygrometric tables or a specially drawn graph, but this instrument has been constructed with an ingenious scale to give direct readings. As will be seen from fig. 576, it is fitted with wet- and dry-bulb thermometers with a scale of relative humidity fixed between them.

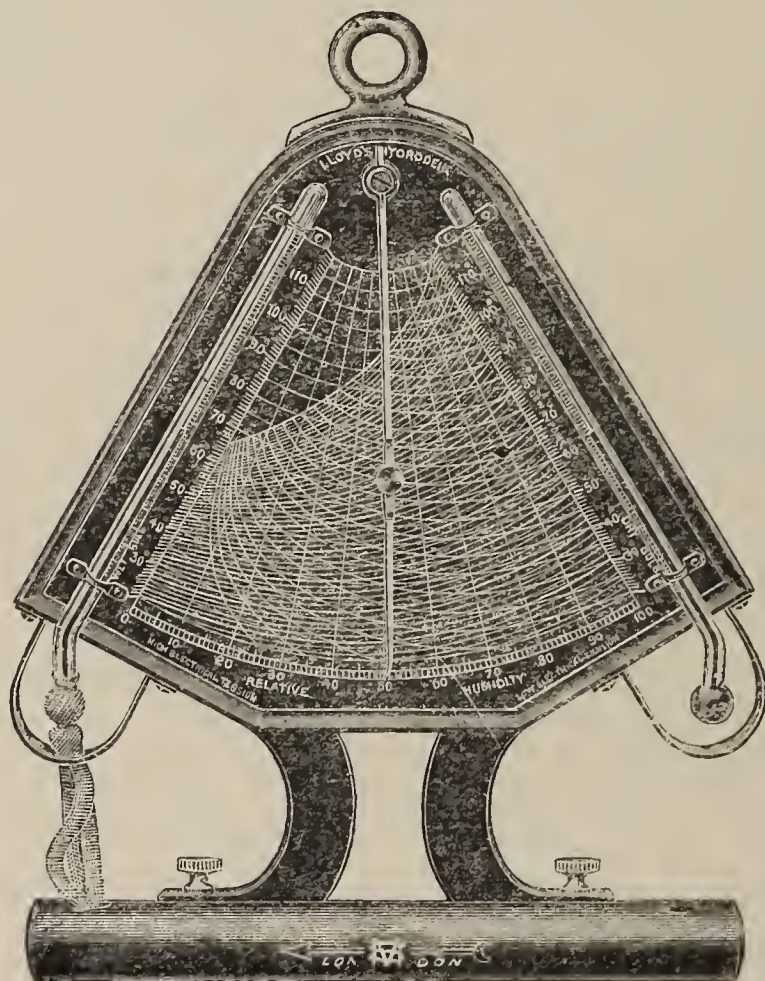


FIG. 576.

Directions for reading the Hygrodeik.—Observe the readings of the wet and dry bulbs, as indicated by the etched graduations on the tubes. Set the sliding pointer at the line on the wet-bulb side of the chart corresponding with the degree reading of the wet-bulb tube; swing the arm to the right, to the point of intersection with the red line curving from the dry-bulb side (the red line to observe is the one corresponding to the degree reading of the dry-bulb tube). At this intersection the index hand will point to the relative humidity on scale at bottom of chart. *E.g.*, should the temperature indicated by wet-bulb thermometer be 60° , and that of the dry-bulb be 70° , the index hand will indicate relative humidity 55 per cent. when the pointer rests on the intersecting lines of 60° and 70° .

To find Dew Point and Absolute Amount in Grains.—Observe the

intersection as above, and follow the heavy black curved line passing through it, which runs from the top downward to the right, to the point of contact with the dry-bulb scale; the degree (53) at this point on that scale is the dew point required; the figure at opposite end of this line will give the absolute amount of water in grains (4·5 grains) per cubic foot of air.

Electrostatic Stress.—Air of low relative humidity will act as an insulator, and thus allow a body to accumulate electricity while the air of high relative humidity prevents such accumulation; hence the term ‘high electric tension’ on the left of the line indicating a relative humidity of sixty, and ‘low electric tension’ on the right of that line.

Example.—A ventilating current of air of 150,000 cub. ft. per minute saturated with vapour passes down the down-cast shaft at a temperature of 32° F. When it leaves the up-cast its temperature is 75° F. and it is still saturated with vapour. Find how much water-vapour this quantity of air has absorbed from the underground workings.

By the formula $Q_2 = \frac{T + 459}{t + 459} \times Q_1$, the quantity or volume of air in the up-cast

$$\begin{aligned} \text{shaft} &= \frac{75 + 459}{32 + 459} \times 150000 \\ &= 1\cdot0875 \times 150000 \\ &= 163125 \text{ cub. ft. per minute} \end{aligned} \quad \left\{ \begin{array}{l} Q_1 = \text{quantity of air entering originally.} \\ Q_2 = \text{quantity leaving the shaft.} \\ T = \text{temperature of air in the up-cast.} \\ t = \text{,, ,, down-cast.} \end{array} \right.$$

From the tables already referred to it is found that

1 cub. ft. of air at 32° F. contains 2·20 grains of vapour
and 1 ,, ,, 75° F. ,, 9·41 ,, ,,

$$\therefore \text{vapour in down-cast volume} = \frac{150000 \times 2\cdot20}{7000} = 47\cdot10 \text{ lbs. } \{ 7000 \text{ grains} = 1 \text{ lb.} \}$$

$$\text{,, up-cast ,,} = \frac{163125 \times 9\cdot41}{7000} = 219\cdot05 \text{ ,,}$$

$\therefore$ vapour absorbed from mine workings = 219·05 – 47·10 = 161·95 lbs., or 16·195 gallons of water per minute.

The compound gases found in mines are four in number, viz., carbon dioxide (CO₂), carbon monoxide (CO), sulphuretted hydrogen (H₂S), and carburetted hydrogen or methane (CH₄).

Black-damp, Choke-damp, or Stythe, is a gas, or, more correctly, a mixture of gases, frequently met with in mines, especially in old workings or badly ventilated parts of the mine. In fact, in mostly all mines it is generally present to a greater or less extent in the return air-currents. Until quite recently black-damp was supposed to be composed of pure carbon dioxide or carbonic acid gas (CO₂), but the investigations and analyses of Dr John Haldane have shown that this is not the case. In a series of analyses of samples taken from the underground workings of several collieries it has been shown that the composition of black-damp was very regular, and consisted of 85 to 88 per cent. of nitrogen and 12 to 15 per cent. of

carbon dioxide. The following table, taken from Dr Haldane's published researches, will show more clearly the composition of this gas :—

TABLE SHOWING THE COMPOSITION AND SPECIFIC GRAVITY OF BLACK-DAMP.

Component Gases.		I.	II.	III.	IV.	V.	VI.
Composition of the sample	{ Oxygen, .	1.45	0.72	10.07	9.60	13.66	13.60
	{ Nitrogen, .	82.56	80.78	82.30	83.08	78.97	80.68
	{ Carbon dioxide,	10.64	11.03	7.63	7.32	4.49	4.82
	{ Marsh gas, .	5.35	7.47	0.00	0.00	2.88	0.90
		100.00	100.00	100.00	100.00	100.00	100.00
Calculated specific gravity of sample,		1.0106	1.0030	1.0274	1.0258	1.0029	1.0129
Calculated composition of the pure black-damp	{ Nitrogen, .	87.87	87.65	85.30	86.48	85.86	85.90
	{ Carbon dioxide,	12.13	12.35	14.70	13.52	14.14	14.10
		100.00	100.00	100.00	100.00	100.00	100.00
Calculated specific gravity of the pure black-damp		1.0390	1.0403	1.0534	1.0468	1.0502	1.0500

It will be seen that the specific gravity of black-damp is very much lower than was hitherto supposed to be the case. The specific gravity was usually taken at 1.52 (air=1), which is the density of pure carbon dioxide, whereas it is now shown to vary from 1.0390 to 1.0534, or only about 4 or 5 per cent. higher than air. Black-damp as actually met with in mines may, moreover, be sometimes lighter than air, in consequence of admixture with fire-damp.

Black-damp is produced by the decomposition or combustion of carbonaceous matter in a free supply of oxygen, and is formed in mines by the decay of organic matter, by the exhalations of men and animals, the gaseous products resulting from burning lamps and blasting operations—in fact, wherever combustion is going on. In some mines it is abundant, and is given off naturally from the strata like fire-damp.

According to Dr Haldane, the two principal theories which may be advanced to account for the presence of this gas in mines are—(a) that the formation of black-damp is due to the oxidation of coal or associated strata ; (b) that it is evolved from the coal or associated strata. He regards the first—the oxidation of coal—as being the principal and most likely cause of the origin of black-damp in coal mines. It is well known that many kinds of coal when exposed to air undergo a slow process of oxidation ; in fact, the oxidation may proceed so rapidly as to give rise to a considerable increase in the temperature and finally to spontaneous combustion, causing what are known as gob fires. He maintains that black-damp is nothing

but the residual gas left by this slow process of oxidation of the coal, at ordinary temperatures. Black-damp does not issue at high pressure from freshly-cut coal in the same way as fire-damp; and coal which has had ample time to drain off all its other gases, may still continue for months and years to produce black-damp. While oxidation of the coal may be the chief cause of the production of black-damp, it cannot be the only cause, for the gas is frequently met with in metalliferous mines and in fire-clay, limestone, ironstone, and other measures where no coal seams are present.

Black-damp has neither colour, taste, nor smell, except when present in large quantities, when a slightly acid taste is experienced. This would, however, be no guide to its detection in mines. It does not burn, nor does it support combustion or respiration.

The proportion of black-damp in air required to extinguish the flame of a lamp varies according to the percentage of oxygen present. Professor Clowes states that it requires the presence of 15 per cent. of black-damp in the air to extinguish a flame, while Dr Haldane gives the percentage required for the extinction of a flame at 15 to 19 per cent., according to the kind of light used. The following table gives the result of some of Dr Haldane's experiments on extinctive percentages of black-damp:—

Component Gases.	Upright Candle Extinguished.	Oil Lamp (bon- neted Clanny) Extinguished.	Hydrogen Flame Extinguished.
Oxygen, ...	17·64	16·43	11·41
Nitrogen, ...	80·15	79·25	79·53
Carbon dioxide, ...	2·21	2·49	5·37
Marsh gas, ...	0·00	1·83	3·69
	100·00	100·00	100·00
Percentage of black-damp,	15·60	19·56	41·72

Black-damp is always difficult to deal with, especially in the case of dip workings, as it may settle near the floor, and a current of air passing over it may fail to remove it. Its presence should always be suspected in such workings (especially if old and unventilated), and at the bottom of wells and sumps.

Tests for Black-damp.—This gas can be readily detected by the flame of a lamp, for if it is present the flame becomes dull, or black and smoky, and with an increasing percentage of the gas the flame will be difficult to keep alight, ultimately being extinguished when the percentage of black-damp has reached 15 to 19, according to the composition of the gas and the quantity of water vapour in the atmosphere. It should also be remembered that with a diminution

of oxygen the light given by a flame also diminishes. Dr Haldane states that, "roughly speaking, the light of a candle or lamp diminishes by 30 per cent., with a fall of 1 per cent. in the oxygen percentage, and the flame will no longer burn when the oxygen has fallen from the normal (20·93 per cent.) to about 17·5 per cent." He further states that "when the air is vitiated with black-damp of about the average composition met with in collieries, an upright tallow candle goes out at about 17·6 per cent. of oxygen and 2·2 per cent. of CO_2 . An open colza lamp went out at 17·4 per cent. of oxygen, and a bonneted Clanny lamp at 17·2 per cent. of oxygen." From this, then, it would seem that the flame of a lamp is extinguished when the percentage of oxygen falls approximately to $17\frac{1}{4}$ to $17\frac{3}{4}$ and $2\frac{1}{4}$ to $2\frac{3}{4}$ per cent. of CO_2 , representing 15 to 19 per cent. of black-damp.

The Coal Mines Act, 1911, enacts that "a place shall not be deemed to be in a fit state for working or passing therein if the air contains either less than 19 per cent. of oxygen or more than $1\frac{1}{4}$ per cent. of carbon dioxide." In view of this enactment, it is necessary that the manager or under-manager should be able to make a test to approximately ascertain the percentage of CO_2 in the air in any place where the presence of black-damp is suspected, and especially in workings where black-damp is frequently met with.

Haldane Tube.—This apparatus, which was invented by Dr Haldane, consists of a strong piece of glass-tube and some thin wax tapers. When a lighted taper is held inside the tube, an upward draught is produced by the heat, and this draught varies in strength, according as the taper is held high up or low down in the tube. The draught tends, naturally, to blow the flame out; and, according to the percentage of oxygen in the air, the flame is extinguished at a point lower down or higher up in the tube. The size of tube chosen as being convenient for the test is one 7 in. long by 0·75 in. internal diameter. The main graduations are in percentages of black-damp, from 0 up to $10\frac{1}{2}$; and there are subsidiary corresponding graduations in percentages of oxygen, from the normal of 20·9 down to 18·8. Below about this percentage the taper will no longer burn in the tube; and when held upright, it will no longer burn outside the tube with less than about 18·2 per cent. of oxygen, or 13 per cent. of black-damp, but when held in a horizontal position, it will still burn until the percentage of oxygen has fallen to about 17·2, or 18 per cent. of black-damp. There is thus a wide range within which the proportion of black-damp can be estimated by the taper and tube. As the flame becomes very small when it is just on the point of extinction, its position with respect to the graduations on the tube can be determined quite easily. The lighted taper should be first pushed up to a point where it burns easily, and then gradually lowered. With a little practice it is easy to find with considerable exactness the point at which it just extinguishes. If

there is much black-damp in the air, it may be necessary to push the taper through the tube before lighting it, and then to lower it cautiously into the tube. The tapers used are $\frac{1}{10}$ th of an inch in diameter; thicker tapers are much less convenient, and are apt to crack the glass. The tubes must, of course, be held vertical while the test is being made, to prevent the glass from cracking.

The tube of the size specified has been graduated by experiments in an air-tight chamber, the black-damp being produced either by respiration or by allowing gas to burn in the chamber. The experiments showed that the tube indicates the state of the air with very surprising sharpness and accuracy. There are, however, several other circumstances which affect the flame, besides the oxygen percentage as revealed by chemical analysis. The chief of these is the percentage of aqueous vapour in the air. An ordinary chemical analysis gives the percentage of oxygen in absolutely dry air—which is practically never met with outside a laboratory. The actual percentage of oxygen in ordinary air is, of course, less than 20.93, because the air is diluted to some extent with aqueous vapour; and, on an average, ordinary outside air is mixed with about 1 per cent. of aqueous vapour, which reduces the real oxygen percentage by a hundredth, that is, from 20.93 to 20.72. In mine-air saturated with moisture at a temperature of 90° F., to take a very extreme case, there would be 4.7 per cent. of aqueous vapour in the air. It will be readily understood that the more moisture there is present in the air the lower will be the percentage of oxygen which is given by the tube-and-taper method. For this reason, the tube has been graduated with air saturated with moisture at about 64° F., and containing

about 2 per cent. of moisture. Air with about this percentage of moisture is ordinarily met with in mines, although the air in an intake usually contains less moisture. Except in warm and moist weather, the flame will therefore not go out at the zero-point of the graduations when the test is made in pure air. Both moisture and carbon dioxide have a somewhat greater effect in extinguishing flame or diminishing the light than a corresponding percentage of nitrogen has; and this fact also complicates matters a little from the academic chemical standpoint. If, however, the effect on a light be taken as the measure of impurity in the air, the tube-and-

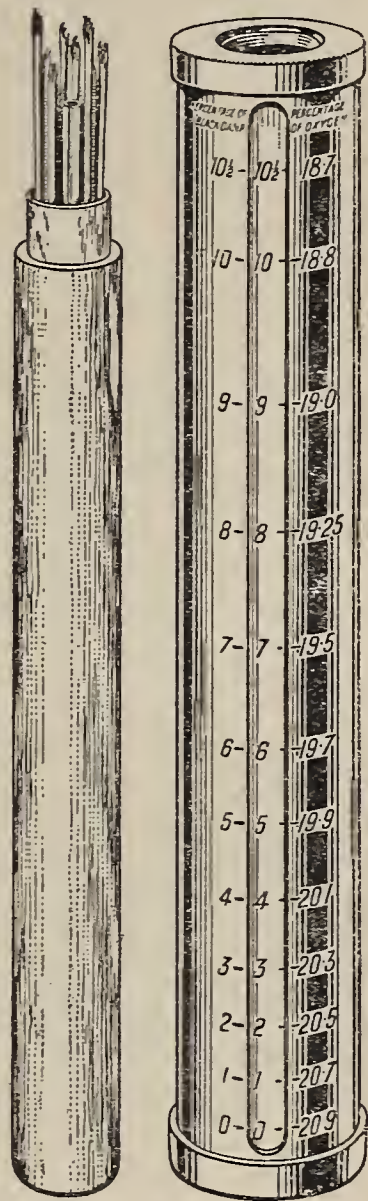


FIG. 577.

taper method gives more correct results than chemical analysis ; for, in so far as moisture and carbon dioxide affect the burning of a light, they are taken into account by the former method. Laboratory experiments indicate that, provided the amount of moisture is not much different from that in air saturated at 64° F., the indications may be relied on to within about 0·2 per cent. of oxygen, or 1 per cent. of black-damp. Roughly speaking, therefore, the extinction of the flame depends upon the true oxygen percentage in the air as it actually exists, instead of in dry air as indicated by a chemical analysis. The mark indicating 8 per cent. of black-damp corresponds roughly to the standard of purity specified in the Coal Mines Act.

Table I. contains the results of a series of experiments made in the closed chamber, with the object of finally testing the graduation of the tube. The vitiation of the air was produced by breathing in it, and by the combustion of the tapers used in the test ; but, while each test was being made, the writer breathed to the outside through a tube until the sample was taken for analysis. The air was also thoroughly mixed by a fan before each test, and the percentage of moisture, as calculated from the readings of wet- and dry-bulb thermometers waved about in the air, was kept at about 2.

The taper was extinguished, held vertically, when the oxygen fell to 18·2 per cent. These experiments show that under favourable conditions pretty accurate results can be obtained. The accuracy shown is, in fact, greater than that ordinarily reached by a Hempel gas-analysis apparatus. Similar accuracy cannot, however, be expected in tests made underground, unless troublesome corrections are introduced for the varying percentages of moisture in the air and the varying proportion of carbon dioxide in the black-damp, and unless great care is taken as to the exact point at which the flame is just extinguished.

TABLE I.—RESULTS OF EXPERIMENTS MADE IN THE CLOSED CHAMBER.

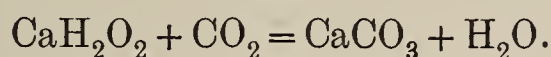
Percentage of Oxygen.		Percentage of Carbon Dioxide.	Percentage of Black-damp.	
Indicated by Tube.	Found by Analysis.		Indicated by Tube.	Found by Analysis.
20·90	20·90	0·06	0	0·1
20·45	20·33	0·49	2½	2·9
20·15	20·13	0·65	3¾	3·8
20·00	20·07	0·71	4½	4·1
19·65	19·63	1·07	6¼	6·2
19·25	19·32	1·33	8	7·7

TABLE II.—RESULTS OF EXPERIMENTS MADE UNDERGROUND.

Percentage of Oxygen.		Percentage of Carbon Dioxide.	Percentage of Black-damp.	
Indicated by Tube.	Found by Analysis.		Indicated by Tube.	Found by Analysis
20·9	20·78	0·16	0	0·7
19·7	19·73	1·02	6	5·9
18·9	19·28	1·51	9½	7·9
19·5	19·65	1·06	7	6·1
20·7	20·77	...	1	0·8
20·5	20·73	0·12	2	1·0

It will be seen that in every case but one the indication of the tube was correct to within about 0·2 per cent. of oxygen, or 1 per cent. of black-damp. In one case, however, the apparent error was 0·38 per cent. It appears from these experiments that the method is sufficiently delicate to make it useful in estimating the percentage of oxygen, or of black-damp, in the air of a mine. For ordinary practical purposes it is preferable to chemical analysis, as it gives the information at once, and the test can be repeated in as many places as may seem desirable. It does for black-damp what the cap-indications do for fire-damp; and with its help the proper distribution of air in a naked-light mine can readily be controlled. For instance, the return-airways may be tested at different points, and excessive leakage, or defective arrangement of regulators, can be detected at once.

Angus Smith's Method.—When carbon dioxide is passed into a solution of calcium hydroxide (lime water), a turbidity at once results, owing to the precipitation of insoluble calcium carbonate or chalk. Thus—



It is upon this reaction that Dr Angus Smith's method of detecting CO_2 is based. The apparatus required for this test consists of (1) one 8-oz. bottle with cork; (2) one 5-oz. bottle with cork; (3) one bottle containing lime water with excess of lime; (4) one pipette to measure up to $\frac{1}{2}$ oz.; (5) one piece of india-rubber tube 12 in. long; (6) four $\frac{1}{2}$ -oz. bottles with corks; (7) one $\frac{1}{4}$ -oz. bottle containing an alcoholic solution of phenolphthalein. To carry out the test, shake up the bottle containing the lime water, and then allow it to stand until the water becomes clear. With the aid of the pipette transfer $\frac{1}{2}$ oz. of the clear solution of lime to the 5-oz. bottle, and then fill up with boiled rain or distilled water. This will give a solution of $\frac{1}{10}$ th the strength of the original lime water. Add a drop or two of the

phenolphthalein, which will have the immediate effect of causing the lime water to assume a pink colour, which it will retain so long as there is any of the lime not neutralised. The dilute lime water is now of such strength that it will just neutralise the CO_2 in the 8-oz. bottle if filled with air containing $1\frac{1}{4}$ per cent. of the gas (the maximum percentage allowed by the Mines Act, 1911). The 8-oz. bottle is filled with air from the place to be tested by sucking out its contents by means of the india-rubber tube. When this has been done, add $\frac{1}{2}$ oz. of the coloured dilute lime water, cork the bottle, and shake it up. If the pink colour disappears, the air contains more than $1\frac{1}{4}$ per cent. of CO_2 . If the colour is retained, the air contains less than this amount. Should the colour disappear, the rate at which it loses its colour gives a rough indication of whether the percentage of CO_2 greatly exceeds $1\frac{1}{4}$ per cent. or not. A series of tests could be carried through with different proportions of the lime water. Thus, if $1\frac{1}{2}$ oz. of lime water made up to 5 oz. with an addition of distilled water will give a solution, $\frac{1}{2}$ oz. of which will just be neutralised by the CO_2 in the air contained in the 8-oz. bottle when $\frac{3}{4}$ per cent. of CO_2 is present. It is thus possible to arrange for a series of lime solutions varying in strength, being used to indicate the amount of CO_2 present, rising from $\frac{1}{4}$ per cent. until the maximum ($1\frac{1}{4}$ per cent.) allowed by the Mines Act is reached. If the test is carried out to ascertain if the air does not contain anything more than the above maximum, only one estimation need be made, as described in the first test. To ensure a fair degree of accuracy, care should be taken not to expose the solutions to the air more than is necessary. Care should also be taken, in sucking the air out of the sampling bottle, that none of the breath is passed back along the tube. The bottle should be well cleaned and dried after each test.

Lunge's Apparatus.—This apparatus, if used carefully, will give a fairly accurate test for the percentage of CO_2 present in air. It consists of a bottle 4 to 5 oz. capacity, fitted with an india-rubber cork, through which passes a short escape tube, and a longer one reaching to the bottom. A ball syringe of 60 cubic cm. capacity is connected by means of rubber tube to the longer of the two tubes. The test solution used is barium oxide (BaO) or sodium carbonate (Na_2CO_3) dissolved in water. Dissolve 5.3 grams of chemically pure sodium carbonate in about $\frac{1}{4}$ litre of distilled water, and dissolve in it 1 gram of phenolphthalein, heating slightly; cool to 18°C . and make up to 1 litre. If a BaO solution is used, dissolve 1.6 gram of BaO in about $\frac{1}{4}$ litre of distilled water, and making up the volume to 1 litre, there is thus 0.0016 gram in each 1 c.c. of the solution, or which is equal to 0.16 per cent. solution. This solution must be kept in a well-stoppered bottle. In making a test introduce into the apparatus 10 c.c. of the test solution by means of a 10 c.c. pipette, and add a drop or two of phenolphthalein to give it a pink colour. The ball syringe is then slowly but firmly pressed together, so that

the air to be tested streams through the solution (and the bottle shaken), and this is repeated until the pink colour is discharged. From the number of times it is necessary to press the ball, the CO_2 in the air can be calculated. If only a few pressures are necessary to discharge the colour, the air is very bad ; if from 9 to 10 pressures, moderately good ; if from 20 to 25, very good. If the proportion of CO_2 is large, the discharge of colour is sharp and complete. About 8 pressures of the ball gives the standard of $1\frac{1}{4}$ CO_2 allowed for in the Coal Mines Act.

The following table shows the number of pressures required for varying percentages of CO_2 in the air :—

Number of Pressures of India-rubber Ball.	Parts per 1000 of CO_2 in the Air.	Number of Pressures of India-rubber Ball.	Parts per 1000 of CO_2 in the Air.
2	3.0	16	0.71
3	2.5	17	0.69
4	2.1	18	0.66
5	1.8	19	0.64
6	1.55	20	0.62
7	1.35	22	0.58
8	1.15	24	0.54
9	1.00	26	0.51
10	0.90	28	0.49
11	0.87	30	0.48
12	0.83	35	0.42
13	0.80	40	0.38
14	0.77	48	0.30
15	0.74		

This table is based on the number of pressures of the ball required to neutralise 10 c.c. of a deci-normal solution of pure sodium carbonate in the proportions given above. If a barium oxide solution is used the results got will be somewhat different.

The amount of CO_2 in the air can also be got by calculation.

Example.—Using sodium carbonate solution.

Strength of Solution.

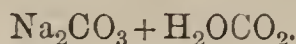
53 grams of Na_2CO_3 per litre of normal solution.

5.3	„	„	„	$\frac{\text{nor. sol.}}{10}$	„
.0053	„	„	c.c.	„	„
.0265	„	„	5 c.c.	„	„

Test. Solution to contain 5 c.c. of $\frac{\text{N}}{10}$ sol. in 50.

Reaction. $\text{Na}_2\text{CO}_3 + \text{H}_2\text{O} + \text{CO}_2 = 2\text{HNaCO}_3.$

The Na_2CO_3 used is pure and *dry*, therefore we only regard the reaction between a molecule of the salt in presence of H_2O and a molecule of CO_2 .



$$\therefore \left\{ (23 \times 2) + (12 + 3 \times 16) \right\} + 44$$

$$\begin{aligned} 46 + 60 + 44 &= 2\text{HNaCO}_2 \\ 106 \text{ grams} + 44 &= 150. \end{aligned}$$

$\therefore$ Weight of CO_2 required to react with Na_2CO_3 to form 2HNaCO_3 in presence of water

$$= \frac{44 \times .0265}{106} = .011 \text{ gram.}$$

$$\begin{aligned} \text{Now volume of this weight of } \text{CO}_2 \text{ at N.T.P.} &= .011 / .001975 \\ &= 5.571 \text{ c.c.} \end{aligned}$$

$$\begin{aligned} \therefore \text{CO}_2 &= \frac{5.571 \times 100}{NV} \text{ per cent.} \\ &= \frac{9.285}{N} \text{ per cent. for a 60 c.c. bulb at N.T.P.} \end{aligned}$$

If the number (N) of compressions of the bulb required to neutralise the solution be 10, then the percentage of CO_2 present in the air

$$= \frac{9.285}{10} = 0.928 \text{ per cent.}$$

Carbonic Oxide, Carbon Monoxide, or White-damp.—Symbol, CO; atomic volume, 14. This gas is also colourless and tasteless; but sometimes possesses a sweet and delicate odour, especially when present in large quantities. It is a combustible gas, burning in air with a characteristic blue flame and forming carbon dioxide, and is a non-supporter of combustion. When mixed with half its own volume of oxygen and ignited, it explodes violently. Fortunately, this gas is found only in exceptional circumstances, such as underground fires, etc. Carbonic oxide is formed by the combustion of carbon with a deficiency of oxygen, or, briefly, is the result of incomplete combustion. Small quantities of this gas are also given off on the explosion of gunpowder. It is usually a constituent of after-damp, and hence there is always great danger after an explosion of fire-damp. As Dr Haldane states: "The air displaced [by an explosion] by expansion is, to a large extent, replaced almost immediately by fresh air sucked in; but along the affected roadways a mixture is left which, although it contains enough oxygen to support life, and enough to support even lights, is extremely poisonous, owing to the presence of carbon monoxide." It is said also to be given off naturally in some metalliferous mines, and has been found in tunnels during driving operations. Although this gas is inflammable, it cannot be detected by the flame of a lamp until there is about 12 per cent. present in air, whereas very much smaller quantities are fatal to life. Carbon monoxide is an extremely poisonous gas; very small quantities present in the air rapidly give

rise to severe headache and giddiness, with palpitation of the heart, and if breathed for any length of time, insensibility and death quickly follow. It has been proved that as a very small percentage of this gas affects small warm-blooded animals more rapidly than man, mice or small birds should be utilised in the detection of this gas, the mouse or bird being carried in a small cage, or inside the gauze of a safety lamp. If on entering the foul atmosphere the animal becomes incapable of motion, it should be regarded as a sign of real danger. It should be remembered that while an atmosphere containing 1 per cent. of CO would be almost immediately fatal if breathed, a very much smaller percentage would be equally fatal if breathed for a sufficient length of time; 0·5 per cent. will saturate the blood as well as 1 per cent., but with the lower percentage it will take a much longer time for saturation to be completed. With about 0·06 per cent. of CO in the air, the blood of a man becomes 30 per cent. saturated after $1\frac{1}{2}$ hours. 0·1 per cent. will give 50 per cent. saturation; with 0·2 per cent. the saturation point is increased to 67 per cent., which would soon bring about unconsciousness and death. With 1 per cent. of CO in the air, saturation of the blood would take place in five or six minutes, and death would rapidly supervene.

The main thing for the student to remember is that the small proportion of CO necessary to produce a fatal result cannot be detected by the flame of a lamp, since the flame cap will not form until there is considerably over 1 per cent. present.

If a person is suffering from carbon monoxide poisoning, pure oxygen should be administered, stimulants given to act on the heart and stomach, and the victim wrapped in warm blankets, and hot water bottles applied. Bringing any one suffering from the effects of this gas suddenly into the fresh air may prove very dangerous, and even may prove fatal. Why this occurs has never been satisfactorily explained, but still it is a fact which has been noted in the case of several colliery accidents.

Detection of CO.—As already stated, this gas cannot be detected by the flame of an open or safety lamp until there is a far higher percentage present in the air than would be immediately fatal to life. The only reliable tests for detecting the presence of carbon monoxide in small quantities are: (1) the test by a small warm-blooded animal such as a mouse or bird; (2) the blood solution test.

It has been found by experiment, and also in actual practice, that a mouse or canary is very sensitive to small percentages of this gas, and hence this test is now frequently used after an explosion of fire-damp, or if an underground fire is suspected. The test by mice and birds for this latter purpose has been found extremely useful at several collieries in Fifeshire where the seams are liable to spontaneous combustion, and a supply of these animals is now kept regularly, either at the surface or underground, at these mines to be ready

for any emergency. The writer, however, does not agree with the practice of keeping either the mice or birds underground, as, even with extremely small percentages of CO, these animals will gradually get inured to its effects, and they will not give such a sensitive test as those brought from the fresh air on the surface.

Dr Haldane, who has experimented extensively with small animals for testing for CO, says that "if a mouse is used for testing, the animal should be in a cage large enough for it to creep about; otherwise, there will be difficulty in judging of the effect of carbon monoxide upon it. For the same reason, a bird should have a perch in its cage. Birds are more easily killed by carbon monoxide than mice, owing, probably, to the fact that they are apt to struggle if they lose their balance, and that any sudden exertion is dangerous. In one sense, therefore, a bird is a better indicator of carbon monoxide than a mouse. But if the animal dies it ceases to be of use as an indicator, just as a safety lamp becomes useless if it is extinguished. From this point of view the mouse is superior. A miner needs to know, not only whether he is in dangerous air, but whether he has got back to safe air." "A man who is moving about breathes much more air than a man at rest, but the rate at which he absorbs carbon monoxide does not increase in the same proportion. The carbon monoxide may, however, be absorbed about twice as fast during rest. Even, however, if this is the case, it will take seven or eight times as long for the man as the mouse to be affected. The safe rule, therefore, is to retire at once from air in which the animal shows any symptoms. It must also be remembered that with proportions of carbon monoxide which are not actually dangerous a man may be ultimately affected more or less, although a mouse or bird shows no very evident symptoms. The symptoms of the animal can only be relied upon to indicate more or less dangerous proportions of carbon monoxide."

The author knows of at least one case where this was fully borne out. A fire had broken out in the workings and some workmen were doing exploratory work, carrying a canary in a cage for testing the air as they proceeded; yet, notwithstanding this precaution, one or two of the men were overcome, and afterwards became unconscious, before the bird showed any signs of carbon monoxide poisoning. It should also be remembered that the carrying out of this test requires practice, and workmen or officials who may be called upon to make the test should be trained by observing, on the surface, the effects which varying proportions of CO will have on mice and birds. It might be worse than useless to send a workman into a mine (where CO was suspected) with a mouse or bird as a guide as to whether the air was safe to inhale or not. Such procedure would be on much the same parallel as sending a man with a safety lamp to test for fire-damp who had neither previously made the test nor had practical experience with fire-damp. As Dr Haldane says, the animal

test is in some ways a rough one, but he believes it to be a practical one if properly and intelligently used.

Test by Blood Solution—Make a dilute solution of normal blood, which may be obtained from a prick in the finger. Divide the solution into two portions, and place in test tubes of equal diameter. The amount of blood used should be such as give the solution a yellow tinge.

Pass ordinary coal gas through the solution in one of the test tubes till the hæmoglobin is saturated with CO, when its colour will change to pink. A preliminary experiment will easily decide whether the dilution is sufficient to show fully the difference in tint.

A third solution is made from the blood of the person who is suspected of having been breathing an atmosphere containing a small quantity of CO—or from the blood of a small animal, such as a mouse, which has been placed in the suspected atmosphere and killed before removal. In making up this third solution a test tube of similar diameter to the others should be used. This solution must be carefully diluted until its depth of tint is approximately the same as that of the other two solutions.

Compare now the tints of the blood in the three tubes by looking through them when held against a good light. It will be found that the normal solution is yellow, the saturated solution pink, while the third tube containing the portion under investigation will, if CO be present, be intermediate in colour.

From this it is possible to obtain a rough estimate of the percentage of CO, the blood being half saturated with .08 per cent., one-third saturated with .04 per cent., and one-fifth saturated with .02 per cent., while two-thirds saturation gives .16 per cent. The approximate condition of saturation is obtained from the colour of the solution under investigation.

The approximate percentage of CO may be calculated from the following formula. Let p = percentage present and s = the percentage of saturation by colour determination,

$$\text{then } p = \frac{s \times .07}{100 - s}.$$

The above is near the truth only when the percentage of oxygen in the air which has affected the blood is but little diminished. Should the air contain a much diminished oxygen percentage, then a correction must be applied. This is seldom necessary so far as mine air is concerned.

Sulphuretted Hydrogen.—Symbol, H_2S ; relative weight, 17. This is a combustible gas burning with a deep blue flame, producing sulphur dioxide (SO_2) and water; does not support combustion, but, if mixed in the proportion of 2 volumes of H_2S to 3 volumes of oxygen and ignited, it explodes violently. It has no colour or taste,

but a very strong odour of rotten eggs. Like carbonic oxide, this gas is never found in large quantities in mines. It is produced by the decomposition of pyrites by acid water, or it may be produced by the heating of coal prior to spontaneous combustion, giving rise to what is termed gob-stink. It is an exceedingly poisonous gas, a very small percentage causing sickness and giddiness. Its poisonous effects are more marked upon some animals than others; thus a bird was found to die in an atmosphere containing only $\frac{1}{1500}$ of H_2S , while it required $\frac{1}{200}$ to poison a hare. Cold-blooded animals are in no way affected by inhaling these proportions of the gas. Lehmann found from investigation that 0·07 per cent. caused death to various animals after an exposure of an hour or thereby, and it is thought that the same quantity would have the same result on a man. It is not usually a source of great danger in coal mines, as its presence, even in small quantities, is easily detected, owing to its strong smell; although Dr Haldane asserts that when this gas is present in large proportions the sense of smell disappears, or becomes unreliable. Sudden outbursts of this gas have been known to occur, however, in copper and salt mines, causing loss of life.

Carburetted Hydrogen, Methane, or Marsh Gas, is composed of 1 volume of carbon combined with 4 volumes of hydrogen, which, on uniting, decrease to 2 volumes of fire-damp, and is known amongst miners as 'fire-damp,' 'fire,' or 'gas.' Symbol, CH_4 ; relative weight, 8. Fire-damp is a gas with neither colour, taste, nor smell when pure. It is highly inflammable, and a non-supporter of combustion.

Carburetted hydrogen is found in petroleum, and is given off when the oil is taken out of the earth and the pressure removed. It is also found in marshes (hence the name marsh gas) as the result of the decay of vegetable matter. Vegetable matter is principally composed of carbon, hydrogen, and oxygen, and when it undergoes decomposition in the air, in a free supply of oxygen, the final products formed are carbon dioxide (CO_2) and water (H_2O). When the decomposition process takes place *without* access of oxygen, such as under water, carburetted hydrogen (CH_4), which is a reduction product, is formed. This explains why this gas is held in the coal and given off naturally when the mineral is worked. The association with carburetted hydrogen of free hydrogen, oxygen, nitrogen, and also heavy hydrocarbons, is said to be due to different stages of carbonisation of the vegetable matter, or it may have been produced by the simultaneous decomposition of animal matter.

Blowers.—As the formation of coal must have proceeded under a complete covering of layers of mud, sand, etc., it is evident that the gaseous products accompanying these changes must have collected and then filled, under considerable pressure, not only joints and fissures in the seam and surrounding rocks, but must have permeated the coal itself. The hissing and crackling noise observed at the face of

freshly-worked coal shows that the occluded gases are held under a certain pressure. Where the pressure is great the gas issues from the coal with a hissing noise, like that of steam escaping, a vent of this description being called a 'blower.' These blowers often continue for lengthened periods to give off fire-damp, showing that they must communicate with reservoirs of gas. The pressure has at times been measured, and in some places pressures varying from 460 to 900 lbs. per sq. in. have been recorded. Where gas at these enormous pressures is present, liability to sudden outbursts always exists. Such outbursts are very dangerous, both by their fouling the air currents, and dislodging material.

In the black vein seam of the South Wales coal-field large cavities filled with fire-damp are met with, and when the seam is being worked they sometimes burst out unexpectedly, forcing out large masses of coal and dust, and resulting at times in loss of life. This phenomenon has also been met with in collieries in Fifeshire and in other districts, great volumes of gas generally being given off, and usually resulting in the loss of life. Outbursts also occur from the floor of seams in such large volumes and with such suddenness as to foul the air to a dangerous degree and cause serious explosions. When faults are being approached, blowers of fire-damp are not infrequently liberated, being sometimes preceded by an outflow of water. Great care should therefore be taken when working in such circumstances.

In some shallow mines fire-damp is seldom met with, or found only in very small quantities, having probably escaped from such mines through the permeable strata to the surface. In some seams it is, on the other hand, given off in very large quantities, especially from those earliest worked, which usually drain off the gas from the other seams.

Generally, fire-damp is most abundant in seams of considerable depth, being given off naturally from the strata, and also freely from the coal face; it also issues from cracks in the roof and floor, large volumes being given off at times, sometimes heaving up the floor or causing falls of the roof.

Carburetted hydrogen is rarely found in a pure state. It is generally mixed with other gases, principally carbon dioxide and sulphuretted hydrogen. In eighty-five analyses of mine air made by Professor Thomas Gray, Royal Technical College, Glasgow, he found that all the samples contained CO_2 in proportions varying from 0.05 to over 1.25 per cent., four only containing more than $1\frac{1}{4}$ per cent., and all the rest under that quantity. The following is an analysis of the air from one seam :—

Carbon dioxide.	Methane.	Oxygen.	Nitrogen.
(CO_2)	(CH_4)	(O_2)	(N_2)
0.54 %	0.83 %	19.80 %	78.83 %

The small quantities of carbon dioxide found in blowers of fire-damp, or in the return air of mines, have apparently little influence on an explosive mixture of carburetted hydrogen and air, according to experiments which have been made with mixtures of fire-damp and air at Bochum, and by Kriescher and Winkler at Freiberg, where 5 per cent. of carbon dioxide (CO_2) was found to have no effect. This is probably also the case as regards other gases associated with fire-damp, such as nitrogen, such gases seeming to act in the same way as an excess of air, merely as a diluent, and tending to reduce the temperature of the explosion. Dr Haldane says that the presence of black-damp (including CO_2) in a mixture of fire-damp and air has little or no influence on the explosibility of the mixture, *provided that sufficient oxygen* is present for the *complete* combustion of the fire-damp. If, however, the oxygen is insufficient, the explosibility of the mixture is, of course, diminished by the black-damp. Fire-damp in old wastes frequently contains a considerable quantity of carbon dioxide, and if a lamp is put into it the flame may be extinguished, which might lead one to believe that only black-damp was present. This may prove deceptive, for if sufficient air be allowed to enter the waste or old roadway, it would make the gas explosive, proving the larger proportion of it was fire-damp. The writer has known cases where a workman put his open lamp into a mixture which at once extinguished the flame, and he (the workman) at once concluded that it was black-damp, but on more air being led in it soon rendered the mixture explosive. Great caution should therefore be taken in clearing gas out from old workings, and no one should be allowed on the return side until the gas is wholly cleared out, or if it is necessary for anyone to do so he should be provided with a safety lamp.

The experiments of Mallard and Le Chatelier, made on behalf of the French Fire-damp Commission, show that mixtures of CH_4 and air begin to be sharply explosive with 7 per cent. of marsh gas (1 vol. of CH_4 + 12 vols. of air), the maximum being reached with 10.8 per cent. (1 vol. of CH_4 + 8.3 vols. of air), and ceases to be explosive with 14.5 per cent. (1 : 5.9). For the inflammability of a mixture of CH_4 and air they fix the lowest limit with an amount of 5.8 per cent. of marsh gas, and the upper limit from 16 to 17 per cent.

When a mixture of fire-damp and air in certain proportions is brought in contact with a naked flame, combustion may result either in the quiet burning away or in an explosion of greater or less violence; that is, in the rapid translation of flame through the whole mixture.

In the course of the experiments it was found that whilst 2 vols. of CH_4 require for their complete combustion 4 vols. of oxygen, 2 vols. of hydrogen require 1 vol.; 2 vols. of ethane (C_2H_6) require 6 vols. of oxygen. Taking the average percentage of oxygen in the air as 20.7 vols., a mixture of air and each one of these gases, to

contain sufficient oxygen to burn each completely and form, therefore, the most explosive mixture, must contain the following amounts of each :—

	Per cent.
Pure hydrogen (H_2), ...	29·28 vols.
Marsh gas (CH_4), ...	9·38 „
Ethane (C_2H_6), ...	5·58 „

If, therefore, any of these gases be present with CH_4 , the percentage of such mixed gases required to form with air the most explosive mixture (explosive maximum) will be greater or less than 9·38, according as the gas is either hydrogen or one of the other hydrocarbons.

The following table shows the results obtained by the British Royal Commission on Accidents in Mines, of varying percentages of fire-damp on a naked light :—

CH_4 in air.	Effect.
2 per cent.	Produces slight elongation of the flame.
$2\frac{1}{2}$ „	A distinct elongation.
$4\frac{1}{4}$ „	Inflames and burns slowly.
6 „	„ explodes sharply.
9·38 „	„ „ with greatest violence and perfect combustion.
20 „	The flame just burns feebly.
25 „	Extinguishes the flame.

Mallard and Le Chatelier submitted the question of the temperature of ignition of carburetted hydrogen to a very thorough and complete investigation, and have found the temperature of ignition for mixtures of CH_4 and air to be $740^{\circ} C.$ ($1364^{\circ} F.$), and that it is practically constant whatever be the proportions of the constituents. But it was found that ignition does not take place immediately the gas, or even a portion of it, has been raised to this temperature ; the gas must be exposed *some seconds* to the above temperature before explosion takes place. Professors Wüllner and Lehmann, in their experiments on the same subject, found that mixtures of CH_4 and air in certain proportions were more easily ignited by some kinds of wire, when heated, than by others, copper-wire only causing ignition at the moment of its fusion (about $1100^{\circ} C.$), while platinum wire 0·50 mm. diameter ignited at $1480^{\circ} C.$ when the mixture was in the proportion of 1 of CH_4 to 14 of air. They also found that the most easily ignited mixtures are not the most explosive, viz., 1 : 9 and 1 : 10, but those containing gas and air in proportion of 1 : 14 (6·6 per cent. of gas). It was also noted that the higher the velocity at which the inflammable mixture was moving, the higher the temperature of ignition. Wüllner and Lehmann also made experiments with electric sparks, and found that open sparks, as produced by an electric current (using a dynamo) between copper wires 3 mm. diameter, ignited mixtures of CH_4 and air in the proportion of 1 : 9

with a current of 18 amperes and above, whilst with a current of 15 amperes or less, occasional sparks caused ignition. The ignition took place more easily with brass and iron wires than with copper, but the ignition is no doubt influenced by the heating of the wires, so that a current of 8 amperes may be dangerous. In using carbon points it was found much more difficult to ignite the mixtures than with metal wire; in fact, an arc of 10 amperes could be maintained in the most explosive mixture without danger. These results have been amply confirmed by Dr Thornton of Durham College, Newcastle, who carried out an extensive and most interesting series of experiments on the ignition of coal-gas and methane by electric sparks. The results of these experiments are given in an elaborate paper in the *Transactions of the Institution of Mining Engineers* (vol. xlv. p. 145), which should be carefully studied by all who are interested in the subject. Hitherto it was believed that all visible electric sparks would ignite mixtures of fire-damp and air, but Dr Thornton has proved beyond doubt that such is not the case. He found that with 11 per cent. of gas and air it was impossible to ignite the mixture at the highest current without burning the contacts badly. He also found that "the least current required for ignition in a circuit at a constant voltage must vary considerably with the richness of the gaseous mixture." He further states that "at 50 volts no ignition could be obtained at 4 or above 14 per cent. of methane in air, the limits generally given being 5 and 13 per cent." This is shown in the following table which he gives:—

VARIATION OF IGNITING CURRENT WITH PERCENTAGES OF METHANE
IN AIR (50 VOLTS. SMALL IRON POLES).

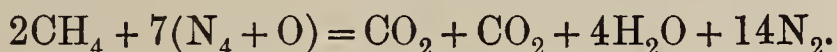
Percentage, . . .	5	6	7	8	10	12	13·5	14
Amperes, . . .	5	3·8	3·2	3·7	4·6	5·75	7·0	12

From the results of Dr Thornton's experiments it would appear to be demonstrated that sparking from electric current is not nearly so dangerous, in the presence of fire-damp, as seems to be imagined by a great many mining men, and there is no cause for the outcry which has been raised in some quarters against the employment of electricity in many of our mines.

Explosion of Fire-damp.—When fire-damp explodes with a mixture of air (9·38 per cent. CH_4 in pure air), the following reaction takes place:—



It is very seldom, however, that the exact proportions of fire-damp and air necessary for complete combustion are present when an explosion takes place, as in nearly every explosion the deadly after-damp has been found to contain, in addition to CO_2 and free nitrogen, varying percentages of carbon monoxide and free hydrogen. With an excess of CH_4 , *i.e.* over 9.38 per cent., the question of resulting products when an explosion takes place is a complicated one. There would be incomplete combustion and certainly CO formed, as well as some free hydrogen. Indeed, the reaction would be somewhat similar to that by which generator gas is got by incomplete combustion of coal. If there is an excess of CH_4 , *i.e.* not sufficient oxygen for complete combustion, we may get something like the following result:—



(This equation is for about 1 vol. of CH_4 to 8 vols. of air.) Even if a quantitative analysis could be made it would be different with different temperatures, etc. For instance, in the explosion at Micklefield Colliery, April 30, 1896, when sixty lives were lost, it was found on investigation that forty-six of these had been victims to carbon monoxide poisoning, the CO being present in the after-damp, and not to the force of the explosion at all. Dr Haldane gives it as his opinion that on an average about 70 per cent. of the lives lost in large colliery explosions are due to carbon monoxide poisoning.

Means of Detection.—Fire-damp is usually detected by means of a ‘Davy’ lamp, or by one of the other numerous safety lamps now used. To detect this gas the flame should be turned down as low as possible in the lamp, because, if there is a large flame, it is impossible to see the ‘blue cap’ which forms on the top of the flame if fire-damp is present. The cap is formed by the molecules of gas in the vicinity of the flame igniting and uniting with the oxygen of the air, and the size and distinctness of the cap will vary with the temperature and volume of the testing flame, and with the amount of fire-damp present in the air.

The ‘cap’ increases in length as the proportion of gas increases, until, when there is $4\frac{1}{2}$ to $4\frac{3}{4}$ per cent. present, the ‘blue cap’ fills the gauze of the lamp. With this percentage of gas the mixture would be slightly explosive, the violence of the explosion becoming greater as the percentage of gas in the air increases, until it reaches 9.38, which is the most explosive point.

Professor Thomas Gray, D.Sc., Ph.D., of the Royal Technical College, Glasgow, has made some very careful experiments with safety lamps in underground workings, to determine the heights of caps with varying percentages of fire-damp in mine air. The following table gives some of the results as deduced from these experiments:—

No. of Sample.	Carbon Dioxide (CO ₂).	Methane (CH ₄).	Oxygen (O ₂).	Nitrogen (N ₂).	Size of Cap on $\frac{1}{10}$ -in. Flame.	Maximum. Visible Cap on Larger Flame.
					in.	in.
1	0.26	1.13	20.31	78.30	$\frac{1}{8}$	...
2	0.28	1.46	20.11	78.15	$\frac{1}{8}$	...
3	0.27	1.73	20.18	77.82	$\frac{1}{4}$	...
4	0.29	2.01	20.08	77.62	$\frac{1}{4}$	...
5	0.26	3.28	19.83	76.63	$\frac{3}{8}$	$\frac{5}{8}$
6	0.24	3.41	19.75	76.60	$\frac{3}{8}$	$\frac{5}{8}$
7	0.26	3.44	19.78	76.52	0.5	...
8	0.21	3.48	19.73	76.58	0.6	...
9	0.21	3.64	19.74	76.41	0.7	...
10	0.21	3.71	19.81	76.27	0.7	...
11	0.28	4.13	19.96	75.63	$1\frac{1}{4}$	$1\frac{3}{4}$
12	0.20	4.20	19.64	75.96	$1\frac{1}{4}$	$1\frac{3}{4}$
13	0.21	4.22	19.66	75.91	$1\frac{1}{4}$	$1\frac{3}{4}$
14	0.23	4.84	19.62	75.31	Cap filled the gauze.	

The tests for these caps were made with a Marsaut safety lamp, burning colzaline (double refined naphtha), and the height of the testing flame was adjusted in each case as nearly as possible to $\frac{1}{10}$ of an inch. The heights of the caps were measured with the aid of a scale divided into eighths of an inch, fixed inside the lamp. The tests were made in the workings of a colliery, and to check the percentages a sample of the air was taken with each test, the samples being afterwards carefully analysed. These tests agree very closely with those made by Professor Cadman of Birmingham University on behalf of the Home Office, although he gives the heights of the caps a little larger. Professor Gray states that the caps given by tests 1 and 2 were faint, badly defined, and incomplete, and that some of the observers who accompanied him failed altogether to observe them. It must be remembered that tests made with an ordinary Davy lamp will be incapable of showing these heights of caps with the percentages of caps given; in fact, it is difficult to detect a cap at all with a Davy lamp when less than $2\frac{1}{2}$ per cent. of fire-damp is present in the air. Another factor which has to be taken into account is the kind of oil burned in the safety lamp, for it is well known that a flame got from burning light oils, such as colzaline, shows a larger cap and more readily than lamps burning heavy oils, such as is commonly used in an ordinary Davy lamp. It is also important to remember that light oils of the naphtha class will give what is known as a *fuel* cap on the flame in the absence of fire-damp, and the student should learn to determine the difference between a fuel cap and a cap due to the presence of fire-damp. The presence of other gases intermixed with fire-damp

will also influence to some extent the size and colour of the cap. Again, some men can see a cap much more readily than others, a good deal depending on the eyesight and whether the observer is quick or otherwise in the perception of colours, there being a great difference in this respect in men. Firemen should test themselves in this, and also in the observation of fire-damp caps, by comparing their powers of observation with the other firemen or officials of the colliery at which they are employed. To detect fire-damp when less than 2 per cent. is present in the air requires skill and practice. The flame in the lamp should be reduced as low as possible, or until, if possible, it is non-luminous, because if there is a large flame it is difficult to observe the blue cap which indicates the presence of fire-damp when small percentages are present. After the flame has been reduced, the lamp should be raised cautiously towards the roof, and if a small percentage of fire-damp is present, the flame will 'draw' or flicker somewhat; if 2 per cent. or more is present, the blue cap will immediately be formed and should be readily distinguished. When the gas has been detected the lamp should be lowered with the same caution taken in raising it, because if it is withdrawn rapidly or with a jerk from an explosive mixture the flame may pass through the gauze and ignite the gas. If carbon dioxide be present in any quantity along with the fire-damp, the cap assumes a slightly brownish or yellowish tint.

None but experienced men ought to be allowed to act as firemen and examine for gas in underground workings. The Coal Mines Act, 1911, makes it obligatory for firemen to obtain a certificate certifying their ability to make accurate tests for inflammable gas with a safety lamp, to measure the quantity of air passing, and also as to sufficiency of eyesight for making tests. In collieries where gas has not been detected for twelve months, the examination of the workings may be made with an open light, but this is a practice which ought not to be allowed, as gas may appear at any time. The practice of allowing a fireman to take down a naked light along with his safety lamp, on the understanding that the naked light has not to be carried beyond a certain point, is a practice that cannot be too strongly condemned, and no manager of a colliery ought to allow such a thing, as many accidents have resulted in the fireman making an improper use of the open lamp and carrying it where it ought not to be.

Gas Analysis.—It may be necessary at times to obtain a more accurate estimate of the percentage of fire-damp present in the mine air than that got from the flame tests of a safety lamp. For instance, under the Coal Mines Act, 1911, safety lamps must be used in any section when the return ventilating currents contain more than $\frac{1}{2}$ per cent. of fire-damp (sect. 32 (1-6)), and a place is to be considered dangerous if there is $2\frac{1}{2}$ per cent. or more of fire-damp present in the air, when safety lamps are being used, and for naked

lights $1\frac{1}{4}$ per cent. and upwards (sect. 67 (2)). To determine accurately whether these requirements are being complied with, it would be necessary to have samples of the air taken and analysed. The most convenient apparatus for making such analyses is the Haldane gas apparatus.

The apparatus consists of a gas burette A in which the air sample is measured. It is provided at the top with a three-way tap. The upper wide part is about 1 in. in diameter and has a capacity of 15 c.c. The graduation, which is to 0.01 c.c., extends down the narrow part from 15 to 21 c.c. It is therefore possible to read to 0.001 c.c. with this burette. J is a water jacket which surrounds the gas burette A and control tube N. The water can be mixed by blowing air through a glass tube which extends to the bottom of the jacket. One of the three-way connections of the tap L is used for taking in the sample of air into A through the curved tube C, and the other connects A to the absorption and combustion pipettes of the apparatus. E is the carbon dioxide (CO_2) absorption pipette filled with a 20 per cent. solution of caustic potash. The pipette is filled through the potash receiver S. E is filled with glass tubes to increase the absorbing surface. F is the oxygen-absorbing pipette filled with an alkaline solution of pyrogalllic acid, made by dissolving 10 grams of pyrogalllic acid in 100 c.c. of a fully saturated solution of caustic potash in a stoppered bottle. This solution is introduced into F through the tube K, which is afterwards closed with a piece of solid glass rod. G and H are each partly filled with some of the strong potash solution. This protects the pyrogalllic acid from the air, and also prevents it from becoming gradually diluted with water, as would be the case if G and H were charged with water. The pressure in the burette is adjusted by using the potash pipette as a pressure gauge, and bringing the potash to the mark M before every reading of the burette. N is the control tube, used to make the readings of the burette entirely independent of changes of temperature and pressure during the analysis. A three-way tap P makes it possible to equalise the pressure in N with that of the atmosphere. By means of the tube O, and the three-way tap P, the potash pipette E is brought into connection with the control tube N. At the beginning of an analysis the potash is brought to the mark R by first opening P to the atmosphere and then raising or lowering B. P is then turned so that E is in connection with N, and is not touched again during the analysis. The combustion pipette T, which is filled with mercury, contains a spiral of fine platinum wire which, when on circuit, is raised to a white heat, and burns any combustible gas present in the sample under test. On beginning an analysis, all the connecting tubes of the absorption and combustion pipettes must be full of nitrogen. The nitrogen left at the end of the previous experiment is always kept and used to wash out the connections.

Method of Analysis.—Open P to the atmosphere and control tube. Raise or lower S until the potash is at the mark R. Close P by slightly turning to the left. Bring potash to mark M by means of the levelling tube B, bring pyrogalllic acid solution to mark D by means of levelling tube B, and also bring the mercury to the mark V

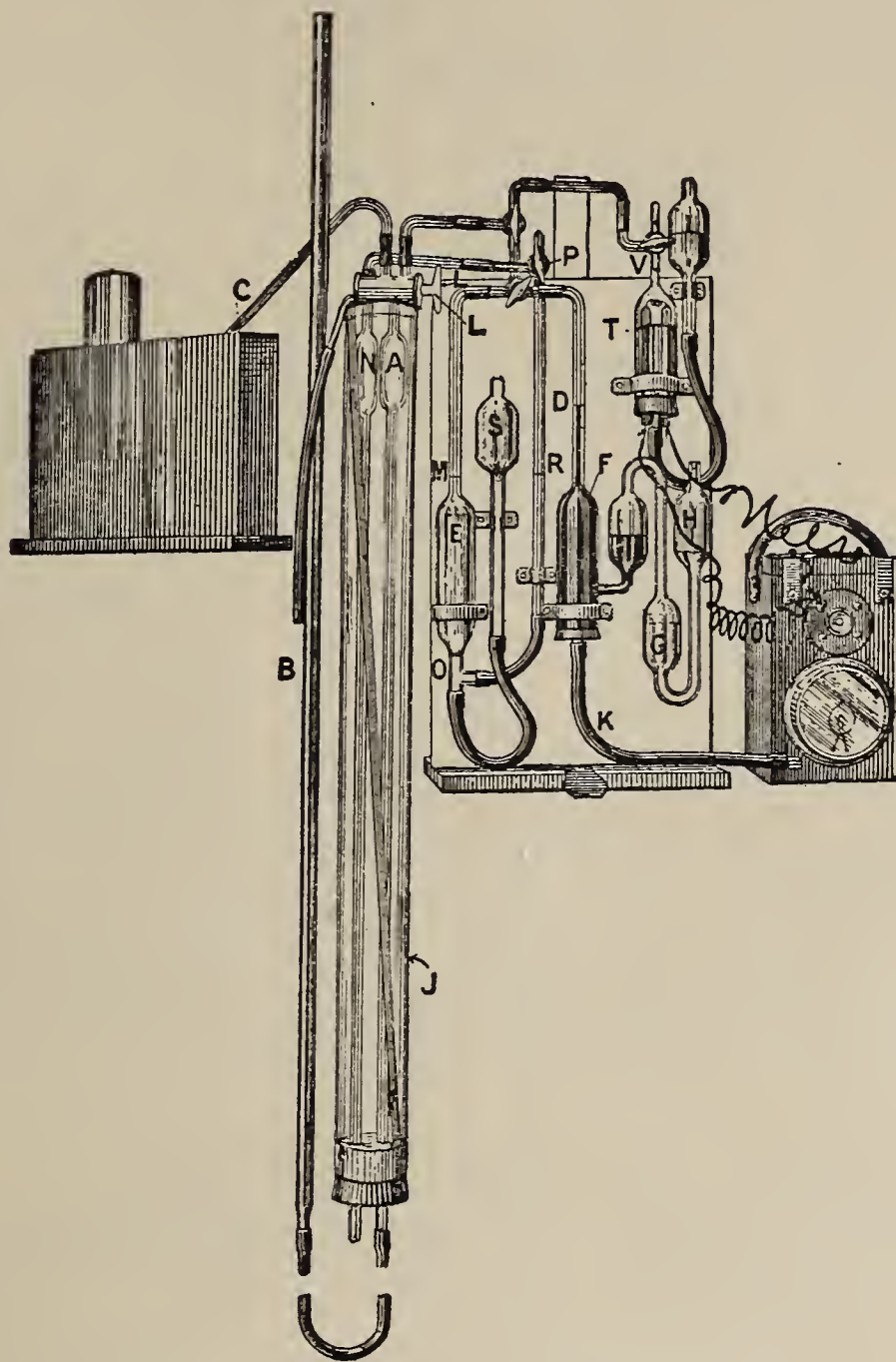


FIG. 578.

in the same way. Care should be taken to see that the taps are turned in the proper direction and that the liquids are exactly at their respective marks. Next turn P to the left to connect N with E; the potash should remain at the mark R, if not, it is brought exactly to R by means of S. The sample is then transferred to the burette by removing the stopper from the bottle under mercury contained in a small mortar. By placing the finger on the neck of the bottle

it can be then transferred to the mercury trough. The bent tube C is introduced into the neck of the bottle, and the burette filled with mercury by opening tap L to the atmosphere and raising B. About 20 c.c. of the sample is then taken into the burette by opening it to C and lowering B, then close tap of burette roughly against pressure by means of the levelling tube B. Connect burette to potash pipette, *i.e.* to pressure gauge, carefully adjust level of potash to mark M, and also to R. The volume taken is then read off accurately and the reading noted. The sample is passed into the potash pipette to and fro several times, and returned to the burette. Air is blown through the water jacket, and the gauges readjusted and the volume read. The diminution in this represents the carbon dioxide (CO_2) in the volume of the sample taken.

The sample is next passed to the combustion pipette, the spiral connected in circuit, part of the sample being passed back and forward for about half a minute. The current is then cut off, and the air returned to the burette by carefully bringing the mercury in the combustion pipette to the mark V, the pressure adjusted as before, and the diminution due to combustion read off. The carbon dioxide formed from the combustion is next absorbed in the potash pipette, the gauges being adjusted and the reading again taken. The sample is then passed into the pyrogalllic acid pipette F, and sent back and forward several times. Bringing the solution back to the mark D, the gauges are adjusted and read, the operation of absorption being repeated until a constant reading is obtained, indicating that the whole of the oxygen has been absorbed. Before each reading, air is blown through the water jacket, and the potash brought to the marks M and R, by means of the levelling tube B and the receiver S respectively. The experiment is repeated, and the following calculations made:—

Initial volume	= a .	$\text{CO}_2 = (a - b) 100 \div a$ per cent.
After 1st absorption	= b .	$\text{CH}_4 = \frac{1}{3}(b - d) 100 \div a$ per cent.
After combustion	= c .	$\text{O} = \left\{ \frac{2}{3}(b - d) + (d - e) \right\} 100 \div a$ per cent.
After 2nd absorption	= d .	
After 0 absorption	= e .	$\text{N} = e 100 \div a$ per cent.

Collection of Samples of Gas in the Mine.—The new Coal Mines Act confers very full powers on H.M. Inspectors of Mines to take samples of the air in any part of a mine, in order to determine by analysis whether the underground air-currents comply with the standards laid down in the Act as to the permissible percentage of oxygen and fire-damp allowed. In view of this, it is necessary that the manager and the officials under him should be able to take samples accurately for analysis. The most convenient method is to take the samples in ordinary two-ounce bottles made of clear glass and fitted with a proper stopper. A bottle of this size holds 70 c.c.,

which is sufficient air for three analyses, so that the determination can be repeated and compared if necessary.

Professor Thomas Gray, of the Royal Technical College, Glasgow, who has made a large number of analyses of the air from Scottish mines, gives the following instructions for taking the samples:—

(a) Remove the stopper from the sampling bottle, retaining the former in the hand and holding it by the projecting portion: avoid laying the stopper down, as gritty particles may adhere to the grease and prevent the stopper from fitting, thus leading to subsequent leakage.



FIG. 579.—Sampling bottle.

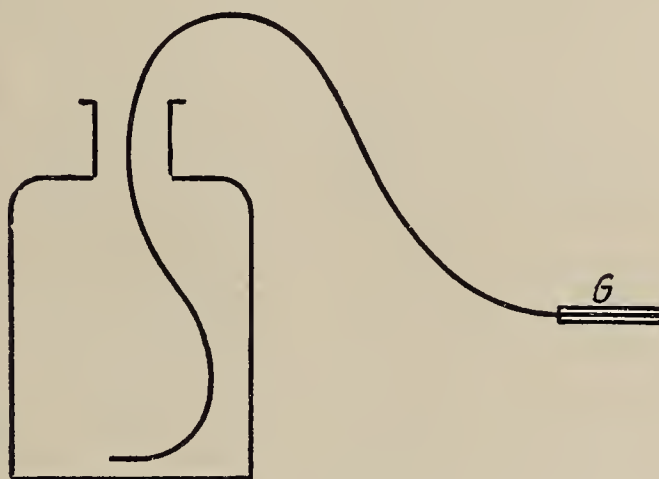


FIG. 580.

(b) Insert the rubber tube into the bottle as shown; stand facing the direction in which the air-current is passing; place the short glass tube G in the mouth and suck the air from the bottle, holding the latter as far away from the body as the rubber tube will permit. Remove G from the mouth *while still inhaling*, to avoid contamination of the sample with carbon dioxide from the breath, and turn *aside while exhaling*.

(c) Repeat the suction twice in the same way, to make sure that the original air in the bottle has been completely replaced by the mine air, taking *particular care* to prevent expired air from the lungs finding its way into the bottle.

(d) Insert the stopper firmly, press it down with a screwing motion, and fasten it down securely by the rubber band.

(e) Attach to each bottle a gummed label giving the following particulars:—

1. Name of mine.
2. Name of seam.
3. Depth of seam.
4. Date.
5. Size of airway.
6. Velocity of air-current in ft. per min.
7. Quantity of air in cub. ft. per min.
8. Place where sample was taken.

On the label, 5, 6, and 7 may be contracted to 5, area = ; 6, velocity = ; 7, quantity = ; and under 8 should be stated 'working face', 'main return', or return from a certain section.

Any additional information for which there is no room on the label should be noted on a separate slip of paper and tied on to the bottle.

The samples should be taken in the air-current at places where the greatest percentages of fire-damp or carbonic acid are expected.

To avoid the use of the mouth tube for sucking the air out of the bottles, and also avoid any risk of the breath passing back into the bottle, Professor D. Burns uses an india-rubber bulb force-pump. This apparatus consists of a rubber bulb and two short rubber tubes, one attached to each side of the bulb, the whole being fitted with a valve mechanism. "When the bulb is compressed its contents are forced out through the discharge tube, and when released it expands, drawing air in by the other tube, which forms an inlet. By inserting the discharge tube into a sample bottle it is easy to obtain a sample of air from any desired place. Care should be taken to pass four to five times as much air through the bottle as it will contain. This ensures that the contents of the bottle are completely displaced, and a proper sample secured."

Fire-damp may accumulate in large quantities in the open waste, where pillars are being taken out, and in rise workings or in holes in the roof, because, being very much lighter than air, it seeks the highest point in the workings. If the seam worked has a soft shale roof and a bed of hard rock, or fakes above, as in fig. 581, the soft shale falls at once, when a 'lift' has been taken off the pillar and the wood first drawn, but the rock does not break for some time afterwards. This causes an open space to be left between the fallen shale and the rock, in which fire-damp accumulates, if present, so that when the rock ultimately falls the space is filled up, and the gas is forced down the edge of the waste into the working places, and, if naked lights are used, may cause an explosion. The only method of dealing with this danger is to work with safety lamps.

If the Longwall system of working is adopted, there may be a space formed between the shale and the rock, which contains gas that escapes into the workings, owing to a subsidence of the rock.

To obviate this, it is common to rip one place up to the rock, the road of course being banked up; this road, which, if possible,

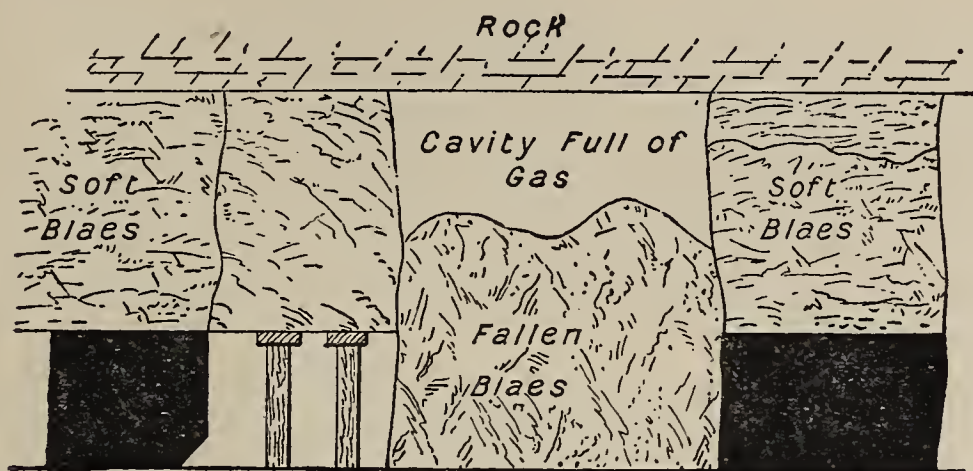


FIG. 581.—Gas-filled cavity.

should be the return airway, has now direct communication with any space that may exist, and consequently acts as a drain for the gas.

In sinking shafts, sudden outbursts of fire-damp have often occurred. When approaching a coal seam the strata are often displaced for a considerable distance from the seam, allowing the gas to issue into the shaft (fig. 582). This may take place more readily after a number of shots have been fired, and therefore great care should be taken in such cases to examine the shaft with a safety lamp before proceeding with the work. The danger may also be guarded against by putting a bore-hole to the coal-head 12 or 18 ft. in advance of the sinking.

Another danger lies in pumping water from old workings or disused shafts where gas may be confined, as when the water pressure is lowered to a certain point the pressure of gas may exceed it, and rush out, with dangerous consequences if any naked light be near. A case of this kind occurred at Kinniel Colliery, Bo'ness, some years ago, whereby two men lost their lives.

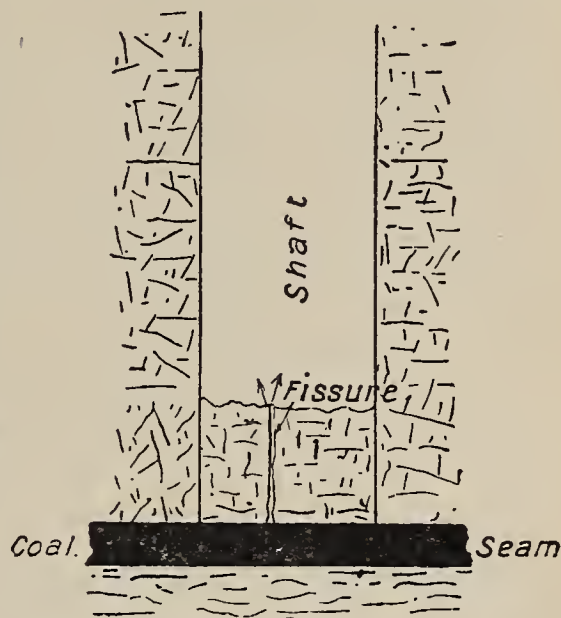


FIG. 582.—Gas vent, or 'blower.'

Systems of Ventilation.—The different methods of accomplishing ventilation are:—Natural ventilation, waterfalls, furnace or steam jets, reciprocating or displacement machines and fans.

We have seen that air possesses weight; 459 cub. ft. of air at

0° F., and 1 in. barometric pressure, weigh 1.3253 lbs. It is therefore forced along a level road or to the dip more easily than to the rise workings, and this is well known in practice, for with airways of equal area a greater proportion of air will find its way to dip workings.

Since air has weight it possesses the property of inertia, which is the resistance offered by a body to any force tending either to impart motion to it or to influence it when in motion.

*When two shafts are sunk and connected by a passage, and the density (weight) of air in the two shafts is equal, no current of air will circulate, no matter what their respective sizes may be.**

If, however, one column of air has a greater density than the other, then the heavier column will overbalance the lighter one and set a current in circulation.

To cause a current of air to circulate, some means must be adopted to alter the density of one of the air-columns. This may be done in three different ways, viz., by expansion in the up-cast; compression in the down-cast; exhausting the air from the up-cast.

Expansion of the air in the up-cast may be attained by natural ventilation, furnace, or steam jet.

Compression in the down-cast may be procured by means of a waterfall or by compressing fans.

Exhausting the air from the up-cast may be carried out by displacement machines or by exhaust fans, the latter being the commonest means employed at mines.

Natural Ventilation.—A mine communicating with the surface by two distinct shafts, having air circulating in them without artificial means, is said to be naturally ventilated. The ventilating current thus set up is caused by a difference in pressure between the two shafts, itself the result of a difference of temperature. Therefore, to have a current of air circulating *naturally*, there must be a difference in depth between the two shafts, to secure a difference in temperature and pressure.

At a short distance below the surface of the earth—50 to 60 ft.—a point is reached where the temperature is constant throughout the year, and as the air acquires nearly the same temperature as the strata, it also is nearly uniform throughout the year.

Descending from this point, the temperature increases on an average 1° F. for every additional 60 ft., and in the case of a mine with two shafts of different depths, it follows that there will be a tendency for a current of air to circulate between them. The air, in travelling round the workings, will also get heat imparted to it by the burning of lamps and the natural heat given off from the strata. Natural ventilation may therefore be due to a difference in depth of the two shafts, due to a difference in surface level, or a difference in depth due to the inclination of the seam.

* *Coal Mining*, by H. W. Hughes, 1899.

Suppose there are two shafts, one 20 fms. in depth and the other 100 fms., the difference in depth being due to difference of surface level. In the winter time, with the temperature at freezing point on the surface, at the bottom of the shallow shaft the temperature would be 52° F., and at the bottom of the deep shaft it would be 59° F.

Now there is a difference in the depth of the two shafts of 80 fms., which would represent a column of air of that height above the surface level at 32° F., opposing a like column of the same height in the deeper shaft at a very much higher temperature, and therefore very much lighter. Let us now examine the pressure exerted in both shafts; the shallow shaft would, of course, be the down-cast in winter, and the deep one the up-cast; the pressure in the down-cast would be—

480 ft. at 32° F. and 1 cub. ft. of air at 32° F. =	·0811 lb. ∴	480 × ·0811 =	38·92 lbs.
60 „ 50° F. „ 1 „ „ 50° F. =	·0780 lb. ∴	60 × ·0780 =	4·68 „
*120 „ 52° F. „ 1 „ „ 52° F. =	·0778 lb. ∴	120 × ·0778 =	9·33 „
Total for down-cast =			52·93 „

In the up-cast shaft the pressure would be

60 ft. at 32° F. and 1 cub. ft. of air at 32° F. =	·0780 lb. ∴	60 × ·0780 =	4·68 lbs.
540 „ 59° F. „ 1 „ „ 59° F. =	·0767 „ ∴	540 × ·0797 =	41·41 „
Total for up-cast =			46·09 „

The difference of pressure causing the current to circulate will be 52·83 – 46·09 = 6·84 lbs. in favour of the shallow shaft, which will cause it to be the down-cast in winter; while if we examined the case in the same manner in summer, it would be found that the deep shaft would then have become the down-cast.

The weight of a cubic foot of air at any temperature and height of barometer may be found by the formula $W = \frac{1·3253 \times B}{t + 459}$

Where W = weight of 1 cub. ft. of air at the given temperature.

B = height of barometer in inches.

t = temperature of air in degrees Fahrenheit.

459 = co-efficient of expansion.

1·3253 = a constant (459 cub. ft. of air at 0° F. and 1 in. bar. press. weighs 1·3253 lbs., and 459 cub. ft. of air at 0° F. and B inches of barometric press. = 1·3253 × B).

The more nearly the two shafts approach the same depth or surface level, the more nearly equal will be the pressure in both shafts, causing the current of air to circulate, and hence the greater likelihood of it stopping altogether at times. In mines where ventilation is effected by natural causes, it is no uncommon occurrence for the air

* 120 ft. at 52° F. is only approximately correct, as the temperature would be different at different points in the shaft.

in summer time to be travelling down one shaft in the morning, to come to a standstill in the middle of the day, and circulate in the opposite direction in the evening. In most mines, therefore, natural ventilation is totally inadequate, and should not be depended on where fire-damp or other dangerous gases are given off.

Waterfall Ventilation.—To set a current of air in circulation in a mine by means of a falling column of water is not a method that is very commonly resorted to, nor is it to be recommended, except in cases of emergency, such as the sudden stoppage of a fan. The author has known cases where the plan proved, however, of much service in ridding the workings of a body of choke-damp, but in this case it was only adopted as an auxiliary to the circulation arising from natural ventilation, and had there not been plenty of surplus pumping power to raise the water to the surface again, it could not have been used at all.

This method can only be used economically when there is an adit level connected to the shaft, and which will convey the water from the workings without the employment of machinery.

Furnace Ventilation.—A number of years ago nearly all collieries of any importance were ventilated by means of furnaces, but this system of ventilation has fallen into disrepute, and now few collieries, except in some districts in the north of England, are supplied with air in this manner; and the employment of furnaces for this purpose is within measurable distance of ceasing altogether, as it has little to recommend it.

The action of the furnace is very simple. The current of air, in passing over the fire, is heated, which causes it to expand, and its density being lessened, it can no longer resist the pressure of the denser and heavier column of cold air behind. Hence it is continuously driven forward into the up-cast shaft, and a current thereby established, while the higher the temperature of the heated air, the greater will be the quantity that will pass over the furnace.

The power of a furnace therefore depends on the amount of heat which it can communicate to the current of air; and as the efficiency of a furnace depends on what is known as the height of the motive column, it necessarily follows that the greater the depth of the up-cast shaft, the greater will be the quantity of air passing.*

The motive column is a head of air of the down-cast temperature and of such a height that it will equal the *difference* of the weight between the air in the down-cast and up-cast shafts; or simply, it is the difference in height between two air columns of different temperatures due to their different densities. The motive column can therefore be expressed in terms of its height in lineal feet and inches, in terms of ventilating pressure expressed in lbs. per square foot, and in

* In furnace ventilation the quantity passing varies as the square root of depth of up-cast shaft.

terms of ventilating pressure represented by inches of water gauge, each of these terms being convertible. The following formulæ will show how the motive column may be expressed in each of these terms:—

Let M=height of motive column in feet.	t_1 =absolute temperature of air in down-cast shaft.*
P=pressure in lbs. per sq. ft.	t_2 =absolute temperature of air in up-cast shaft.
h =motive column in inches of water gauge.	w_2 =weight of 1 cub. ft. of air at 32 Fahr. and 30 ins. barometric pressure (14·7 lbs. per sq. in.).
D=depth of up-cast in feet.	p =weight of 1 sq. ft. of water 1 in. deep (5·2 lbs.).
d =,, down-cast in feet.	

$$\text{Then } M = D \frac{t_2 - t_1}{t_2} \quad . \quad . \quad . \quad (1) \quad \text{or } M = \frac{P}{w} \quad . \quad . \quad . \quad (2)$$

$$P = wD \frac{t_2 - t_1}{t_2} \quad . \quad . \quad . \quad (3) \quad \text{or } P = M \times w \quad . \quad . \quad . \quad (4)$$

$$h = wD \frac{t_2 - t_1}{pt_2} \quad . \quad . \quad . \quad (5) \quad \text{or } h = \frac{M \times w}{p} \quad . \quad . \quad . \quad (6)$$

Example.—If two shafts are each 100 fms. deep, and the temperature of the air in the furnace shaft is at 160° F., while in the other or down-cast shaft it is at 60° F., what would be difference of pressure, and what height of water gauge would this represent?

Taking formula 3 we have the difference of pressure :—

$$\begin{aligned} P &= \cdot 080728 \times 600 \frac{(460 + 160) - (460 + 60)}{460 + 160} \\ &= 48\cdot43 \frac{620 - 520}{620} \\ &= 48\cdot43 \frac{100}{620} \\ &= 7\cdot8 \text{ lbs. per sq. ft.} \end{aligned}$$

and the height of water gauge

$$h = \frac{7\cdot8}{5\cdot2} = 1\cdot5 \text{ ins.}$$

These results are only approximately correct, as the weight of a cubic foot of air is taken at 32° Fahr. and 14·7 lbs. pressure per sq. ft.; to get it mathematically correct the weight of a cubic foot of the air would have to be calculated for each shaft and the average taken, but the difference is so small that this may be neglected.

Example.—If the temperature of the down-cast shaft is 50° Fahr., the depth of the up-cast shaft 300 yds., what would the temperature require to be to give a water gauge of 2 ins. ?

* Zero on the absolute scale may be taken as -492° Fahr. (460 + 32) below the melting-point of ice).

By formula 5 we have

$$h = wD \frac{t_2 - t_1}{p t_2},$$

and transposing we have

$$t_2 - t_1 = \frac{h p t_2}{w D}.$$

Substituting now the values we have

$$t_2 - (50 + 460) = \frac{2 \times 5.2 \times (460 + t_2)}{0.080728 \times 900}$$

$$t_2 - 510 = \frac{10.4(460 + t_2)}{72.65}$$

Transposing we have

$$(t_2 - 510) \times 72.65 = 4784 + 10.4 t_2$$

and

$$72.65 t_2 - 37051.5 = 4784 + 10.4 t_2$$

$$72.65 t_2 - 10.4 t_2 = 4784 + 37051.5$$

$$62.25 t_2 = 41835.5$$

$$\therefore t_2 = \frac{41835.5}{62.25}$$

or

$$= 672^\circ \text{ absolute}$$

$$672 - 460 = 212^\circ \text{ F.}$$

Taking the first example to find the motive column, we have

$$M = 600 \frac{160 - 60}{160 + 459} = 96.9 \text{ ft.}$$

and $p = M \times w = 96.6 \times 0.08072 = 7.82 \text{ lbs. per sq. ft., the same as before.}$

The water gauge may also be expressed in terms of the motive column, $h = \frac{M \times w}{5.2}$. It will be observed that the motive column,

as ascertained by the above formula, is at the *temperature of the down-cast shaft*, and at a pressure of 14.7 lbs. per sq. in. Now it is evident that if 'motive column' is to be used as an expression of ventilating power, the motive column of each mine must be reduced to some definite standard. This standard, as given by Mr Atkinson, is the temperature of 32° F. and 14.7 lbs. per sq. in. atmospheric pressure; therefore, to ascertain the motive column *under any conditions of temperature or pressure*, and reduced to above standard, the formula will become

$$S = \left\{ \frac{\left(\frac{1.3253 \times B}{459 + t_1} \right) - \left(\frac{1.3253 \times B}{459 + t_2} \right)}{0.080728} \right\} D \quad . \quad . \quad . \quad (7)$$

where S = standard motive column

B = height of barometer in inches

t_1 = temperature of down-cast shaft

t_2 = „ „ up-cast „

D = depth of up-cast in feet.

The horse power expended in furnace ventilation may be found by the formulæ

$$\text{H.P.} = \frac{Q \times P}{33000} \quad (8)$$

$$\text{H.P.} = \frac{Q \times M \times w}{33000} \quad (9)$$

or

$$\text{H. P.} = QD \left\{ \left(\frac{1.3253 \times B_1}{t_1} \right) - \left(\frac{1.3253 \times B_2}{t_2} \right) \right\} \quad (10)$$

When Q = total quantity of air glowing in cub. ft. per minute.

B_1 and B_2 = height of barometer in up-cast and down-cast shafts respectively.

As already stated, the efficiency of a furnace depends on the height of the motive column ; therefore, the greater the depth of the up-cast shaft, the greater will be the total quantity of air passing. The quantity of air given by a furnace also varies as the square root of the depth of up-cast shaft, and also approximately as the square root of the difference of temperature between the up-cast and the down-cast shafts. For instance, if the depth of an up-cast shaft is 25 fms., and suppose it increased in depth to 100 fms., then the increased efficiency would be in the ratio of

$$\begin{aligned} \sqrt{25} &: \sqrt{100} \\ 5 &: 10 \\ \text{or } 1 &: 2 \end{aligned}$$

Hence an up-cast shaft 100 fms. deep would give twice the quantity of air that one 25 fms. would give, all else being constant.

Take, again, the temperature ; suppose the mean temperature of the down-cast is 60° F. and the up-cast 85° F., and assume that the latter temperature has been increased to 160° F., then the quantity of air passing will be in the ratio of

$$\begin{aligned} \sqrt{85 - 60} &: \sqrt{160 - 60} \\ = \sqrt{25} &: \sqrt{100} \\ = 5 &: 10 \\ \text{or } 1 &: 2 \end{aligned}$$

Hence the quantity of air will be doubled.

Again, the efficiency of a furnace varies directly as the fuel consumption, therefore the horse-power for a given fuel consumption varies inversely as the depth, and may be shown as follows :—

Depth of shaft in ft.	Fuel consumed per horse-power per hour.
250 ft.	96 lbs.
500 „	48 „
1000 „	24 „
1500 „	16 „
2000 „	12 „
3000 „	8 „
4000 „	6 „

This table shows that, before a furnace can become as efficient as an engine working a fan, in regard to the fuel consumed, the shaft would require to be nearly 3000 ft. deep, as an engine for driving a fan, if in good condition, should not, on an average, consume more than 8 to 10 lbs. of coal per horse-power per hour.

The difference in cost and upkeep between a fan and furnace plant, both dealing with the same quantity of air, may be taken as under :—

(1) <i>Fans, etc.</i>				(2) <i>Cost of Furnace.</i>			
Approximate Price.				Approximate Price.			
Two double-flued Lancashire boilers 30 ft. × 7½ ft., ...	£750	0	0	Estimate for building and material, ...	£150	0	0
Guibal fan 40 ft. × 12 ft. (32 to 35 revs. per minute), with engine complete, ...	1100	0	0	Making 'dumb-drift,' etc., ...	100	0	0
Buildings, fore engine house, fan casing, etc., ...	500	0	0	Total, <u>£250</u>	<u>0</u>	<u>0</u>	
Sundries, ...	100	0	0	<i>Cost per annum (365 days).</i>			
Total, <u>£2450</u>	<u>0</u>	<u>0</u>		Interest on £250 at 5 per cent., ...	£12	10	0
<i>Cost per annum (365 days).</i>				Depreciation at 5 per cent., ...	12	10	0
Interest on £2450 at 5 per cent.	£122	10	0	Repairs, ...	5	0	0
Depreciation at 5 per cent., ...	122	10	0	Tripping, 8½ tons per 24 hours at 3s. per ton, . .	479	1	3
Repairs, ...	10	0	0	Attendants' wages, three men at 3s. 6d. each per day, ...	191	12	6
Stores, furnishings, etc., ...	10	0	0	Total, <u>£700</u>	<u>13</u>	<u>9</u>	
Dross for boilers (3¾ tons per 24 hours at 1s. 6d. per ton), ...	102	13	1½	<i>Cost of furnace per annum, ...</i>			
Enginemen's wages (two at 4s. per shift), ...	146	0	0	Cost of fan per annum, . .	513	13	1½
Total, <u>£513</u>	<u>13</u>	<u>1½</u>		Difference in favour of fan, ...	187	0	7½

If the item for enginemen's wages (£146) be omitted—and as at most collieries the winding or other enginemen attend the fan along with their other duties—then the difference in favour of the fan per year would be £333, so that the saving per annum would cover the entire cost of a fan installation in about seven years.

The shafts for which the plant in the above estimates was required were 125 fms. deep. In a more shallow mine the fan would compare to even greater advantage.

In the erection of furnaces the greatest care ought to be taken to minimise risk against fire; the side walls should be 14 in. to 18 in. thick opposite the bars, with plenty of room for a current of air

to circulate outside them, and if the furnace is built near to a seam of coal, it should be provided with a double arch, and space left between for a current of air to circulate through to keep the outside cool; the space between the outside arch and the strata being filled with sand and ashes, which are bad conductors of heat. Fig. 583 shows the construction of a ventilating furnace.

In mines where the return air contains fire-damp in dangerous quantities it must not be allowed to pass over the furnace, as it would ignite and cause an explosion, but should be led into the up-cast shaft by a separate way, termed a 'dumb' drift, at some distance from the furnace drift, or it may enter the shaft on the opposite side from the furnace, higher up the shaft, or by another seam, or from the workings of a lower seam, if the furnace be placed

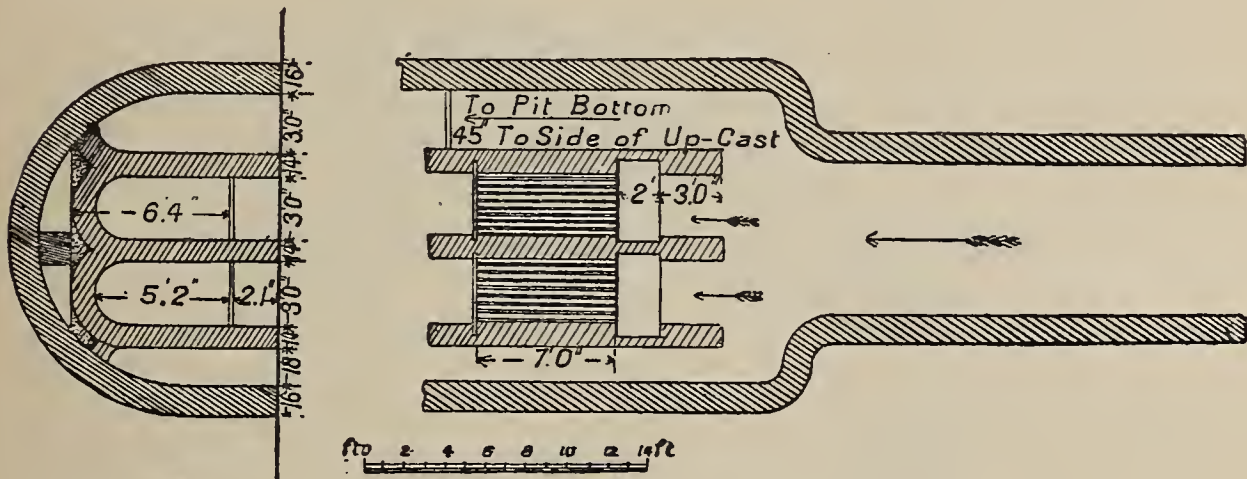


FIG. 583.—Ventilating furnace.

far enough back to prevent the flame of the fire from entering the shaft (figs. 584, 585).

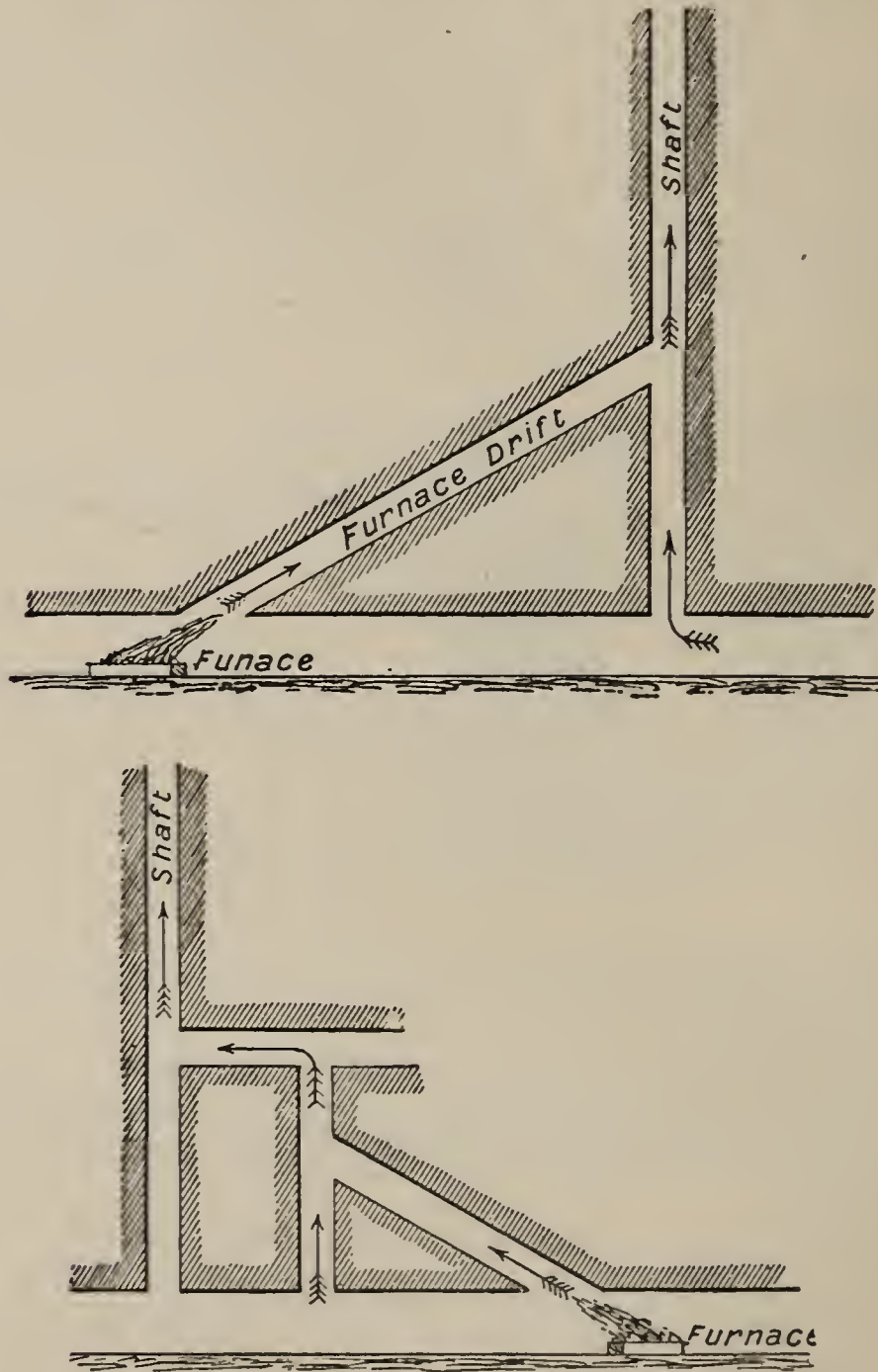
In such cases the furnace must be fed with pure air direct from the down-cast, or from some section of the workings sufficiently pure for the purpose; in either case the efficiency of the furnace would be greatly reduced.

When such is the case, the fire-bars are fitted with doors, so as to force as much air through them as possible, and in this way the air is raised to a higher temperature; if only a certain quantity of air is required to assist combustion, the space below the fire-bars may be fitted with doors, and the air brought from the down-cast in pipes, the supply being regulated so as only to let as much past as is necessary to secure complete combustion. If the fire burns feebly from the presence of black-damp (CO_2), a similar arrangement is necessary.

In cases where iron tubbing is used for lining the shaft, it ought to be protected by an inside lining of fire-brick from the destructive fumes and heated gases given off from the furnace. As fire-brick is a bad conductor of heat, less heat, and therefore less energy, will be

lost than if the air were in direct contact with the iron tubbing during its passage up the shaft.

Speed of Air in Up-cast.—The speed at which the air travels in the up-cast varies very much according to size of the shaft and the extent of the underground workings, and may range from 20 to 50 ft. per



FIGS. 584, 585.—Diagram of furnace ventilation.

second, according to the quantity of air passing, and the area of the shaft.

The use of furnaces underground for ventilation is objectionable and not to be recommended, especially in mines in which fire-damp is given off, the only thing that can be said in their favour being that they are very much cheaper as regards first cost than fans. They have, however, many disadvantages, among which may be noted :—

The possibility of an explosion from a sudden outburst of gas.

Shaft repairs can only be carried out with great inconvenience, if the up-cast is used as a winding pit.

The expense incurred for fuel and attendance is heavy compared with that required for fans.

Where iron tubbing, rope, or iron guides are used in shafts, great damage is done to them by the gases given off from the furnaces, and also by the expansion and contraction due to the variations of temperature.

The circulation of air by means of furnaces may become very unequal, depending often on the management of the man in charge.

Mechanical Ventilation.—Mechanical means of ventilation may be divided into two classes, viz. :—

Statical ventilators = Displacement machines or air pumps.

Dynamical ventilators = Exhausting or compressing fans.

Statical ventilators are now so little used for mine ventilation that they need not be described here.

Fans.—The principle on which all centrifugal fans act is known as centrifugal motion. In the case of a fan the centrifugal force drives the air from the centre, the particles being revolved in a circle by the fan until they are finally thrown off at a tangent, thus creating a partial vacuum at the centre, which causes a further supply of air to enter, and thus establishes a continuous current. In general, such ventilators are used to exhaust the air, and are usually placed at the top of the up-cast shaft, but they may also be used as forcing or compressing machines, in which case they would require to be placed at the top of the down-cast.

Of centrifugal ventilators there are now a large number in use, but those most in favour are the 'Guibal,' 'Waddle,' 'Schiele,' and 'Cappell' types, the others being all, more or less, modifications of one or other of these.

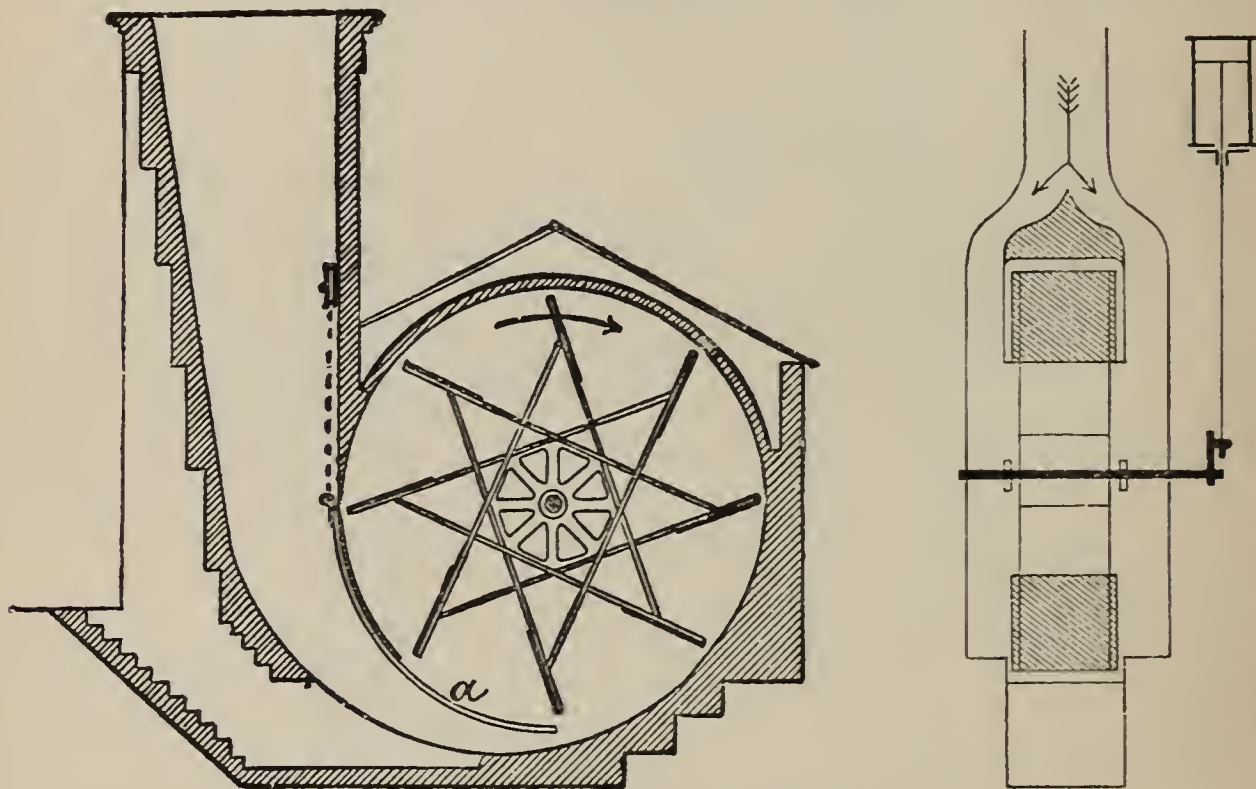
Guibal Fan.—This fan is usually constructed of large diameter, 20 to 45 ft., the width being $\frac{1}{4}$ to $\frac{1}{2}$ of the diameter, while the central opening is about $\frac{1}{3}$ of the diameter.

The Guibal fan works in a brick casing, exhausting the air into an *évasée* chimney, and is now usually fitted with a movable shutter *a*, to regulate the opening for the flow of air.

The fan itself is made of eight or ten straight blades, fitted on to a central casting (figs. 586, 587); in large fans the blades are made of wrought iron, while for small fans, 10 ft. or so in diameter, they may be of wood. The chief objection to this fan is its large size and consequent heavy cost, and also the large outlay for fan chamber and foundations. The great width, moreover, causes a very heavy weight to be distributed over a long length of driving shaft, which can only be supported at each end, and consequently requires very heavy shafting, while instances have occurred where the shafting has suddenly snapped, through the weight. The Guibal fan has, however, from experiments, proved to be a very efficient ventilating machine

under widely varying conditions. The speed of the fan varies with the size, a fan 30 ft. diameter being usually driven at from fifty to sixty revolutions per minute, and one of 40 to 45 ft. diameter at forty to fifty revolutions per minute.

Waddle Fan.—This type of fan is of the large open running description, and is the best of this class of ventilator. It requires no fixed casing or chimney, but delivers the air all round its circumference direct into the atmosphere, and therefore its width is reduced at the periphery, which causes it to be very narrow, in proportion to its diameter, compared with the Guibal fan. Its diameter varies from 20 to 50 ft., the blades *a b* being 12 in. to 18 in. wide at the circumference, and $2\frac{1}{2}$ to 4 ft. wide at the centre, and fixed into the



FIGS. 586 and 587.—Guibal fan.

casing *c c*, both casing and blades revolving in one piece (figs. 588, 589), which tends to reduce the vibration, which is so noticeable a feature in the Guibal fan. All the Waddle fans are now being made with a diverging outlet, *i.e.* the two rims projecting beyond the blades are inclined outwards. This tends to reduce the velocity of the air as it leaves the circumference, and also requires less power to drive it. Its speed is forty to sixty revolutions per minute. These fans are now also made in the small diameter high-speed type, varying in size from 4 to 8 ft. diameter, and running at 350 to 750 revolutions per minute, giving water-gauges of $1\frac{3}{4}$ to $2\frac{3}{4}$ in. for the larger size, and $2\frac{1}{2}$ to $5\frac{1}{2}$ in. for the smaller size. The useful effect varies from 60 to 80 per cent. Some of the advantages claimed for this fan are:—

Being direct driven, it causes little noise and vibration.

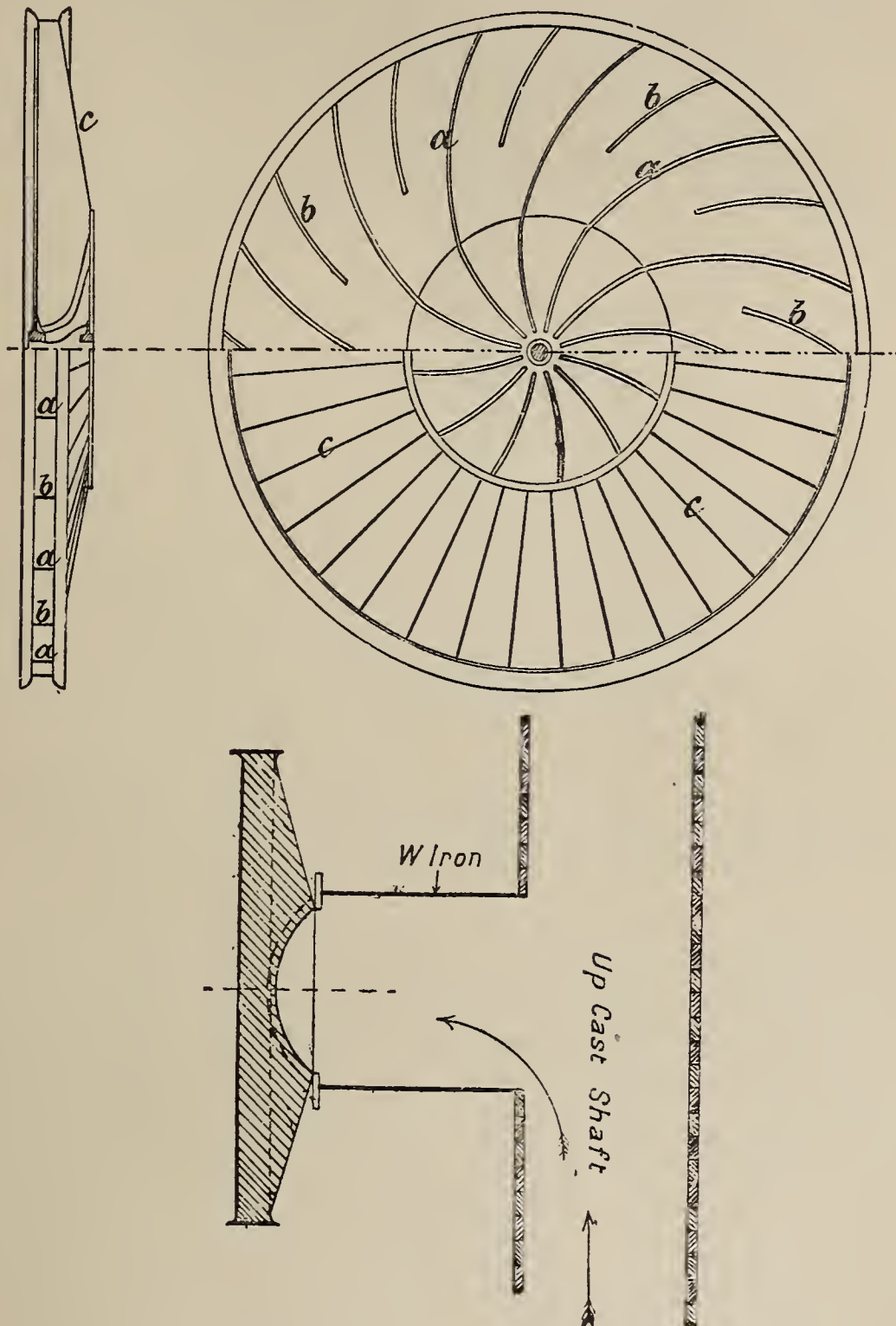
As it requires no enclosed casing or chimney, the first cost of masonry for foundations, etc., is small.

The small cost for repairs per annum.

A high percentage of the power employed is rendered effective.

The vibration and wear and tear are less than in the Guibal fan.

Schiele Fan.—This fan is somewhat like the Guibal in its construction. It has the same expanding chimney, but instead of the



FIGS. 588 and 589.—Waddle fan.

blades being straight, they are curved from the centre, and the casing is open, while the width of the blades is not the same throughout, but greatest in the middle, decreasing towards the centre and the tips (figs. 590, 591). The air is taken in at both sides of the fan, which may be driven either direct or by belting, the latter method

being the most common. In size the fan varies from 5 to 15 ft. in diameter, with a width at the circumference of 1 to 3 ft., and at the centre of 2 to 4 ft. The speed varies from 120 to 400 or 500 revolutions per minute, according to size.

Its chief advantages are :—

- (1) It is moderate in size, and occupies little space at the surface.
- (2) Freedom from vibration and small cost for upkeep.
- (3) The shaft bearings, not being in the return airway, are easier seen and more readily available for repairs.

Cappell Fan.—Like the Schiele, this fan is of the small quick-running type. It is constructed of two concentric cylinders, an

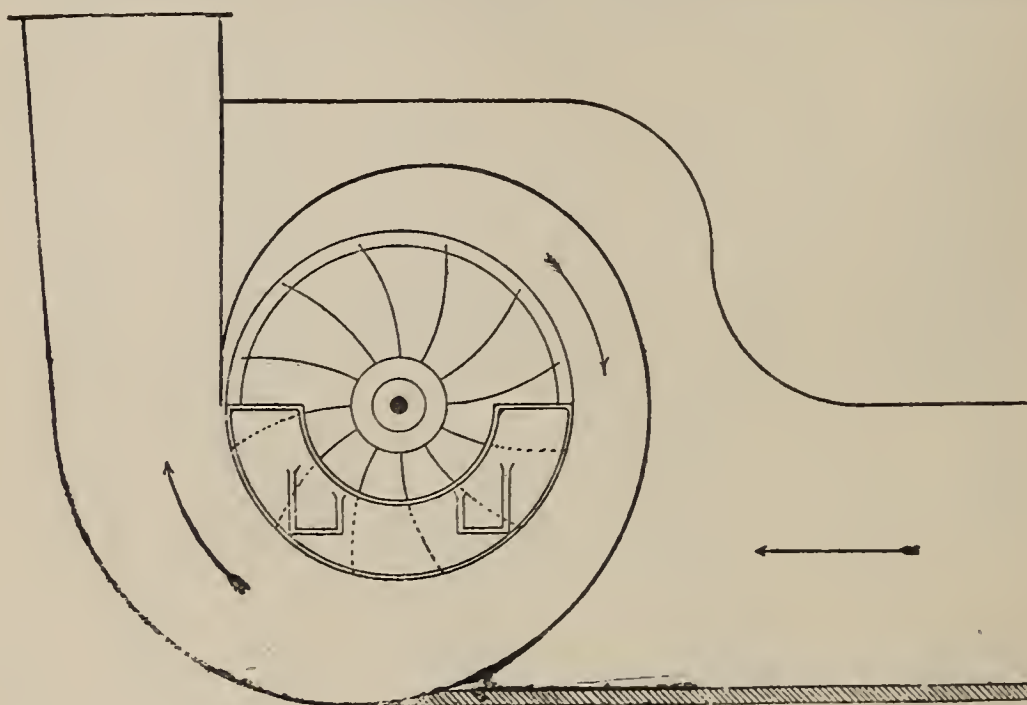


FIG. 590.—Schiele fan.

outer and inner one, each having six curved blades or vanes, the convex sides of which are in the direction of rotation (figs. 592, 593).

The cylinders (1) contain port-holes (2) between the two sets of blades, the air passing through these port-holes between the inner and outer chambers; the air is taken in at the centre and thrown off the inner blades into the outer chamber by centrifugal force at a high velocity. It then strikes against the outer blades, and gives back the greater part of the impulse received from the inner blades, which reduces its velocity when discharged. Velocity imparted to the outgoing air always, it should be noted, diminishes the efficiency of the fan.*

In size this fan varies from 8 to 15 ft. in diameter, with a width of 7 to 11½ ft., and it is worked at speeds of 180 to 300 revolutions per minute. The advantages claimed for this fan by the makers are :—

- (1) It can do a large amount of useful work in proportion to its size.
- (2) Smallness of fan reduces the capital outlay.
- (3) It can withdraw large quantities of air.

* *Ore and Stone Mining*, Sir C. Le Neve Foster, sixth edition, p. 532.

The latest type of this fan is shown in figs. 594, 595.

Walker Fan.—This fan (fig. 596) is constructed somewhat like the Guibal, but it also combines other types. It may be termed a medium-sized fan, as it is generally from 20 to 25 ft. in diameter. It is built exclusively of iron and steel. On the main shaft are fitted two strong cast-iron bosses, which extend lengthwise on each side towards the journals, thus distributing the weight of the fan over a

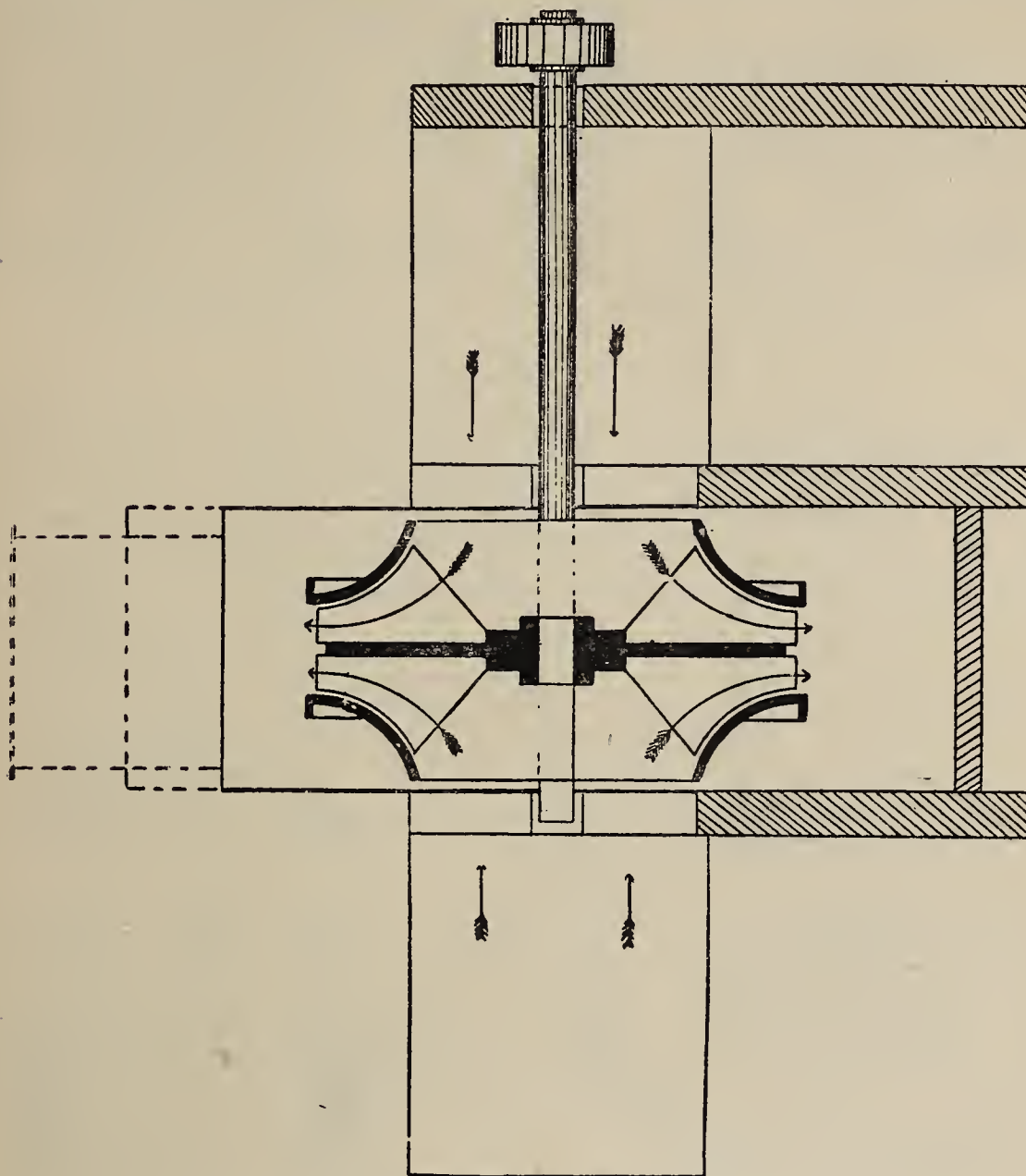
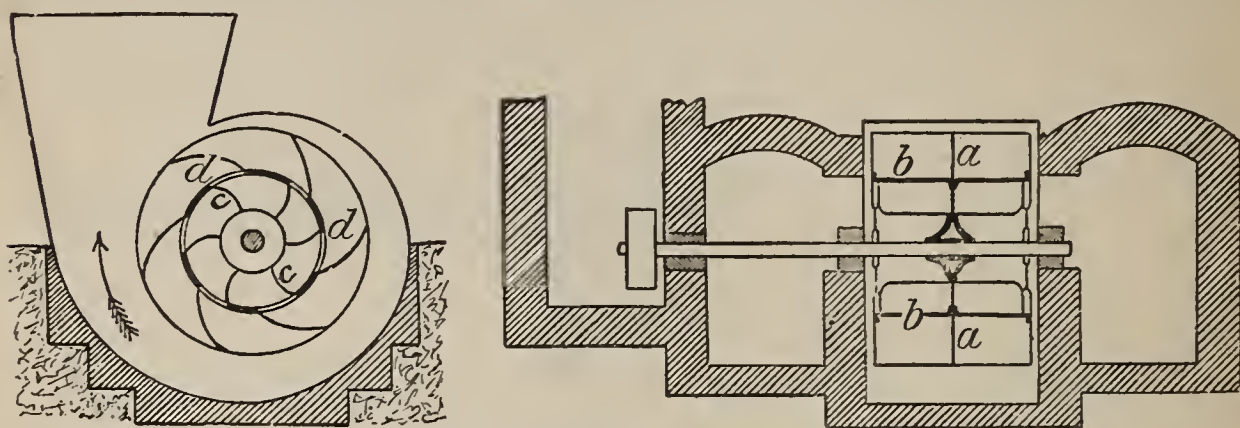


FIG. 591.—Schiele fan.

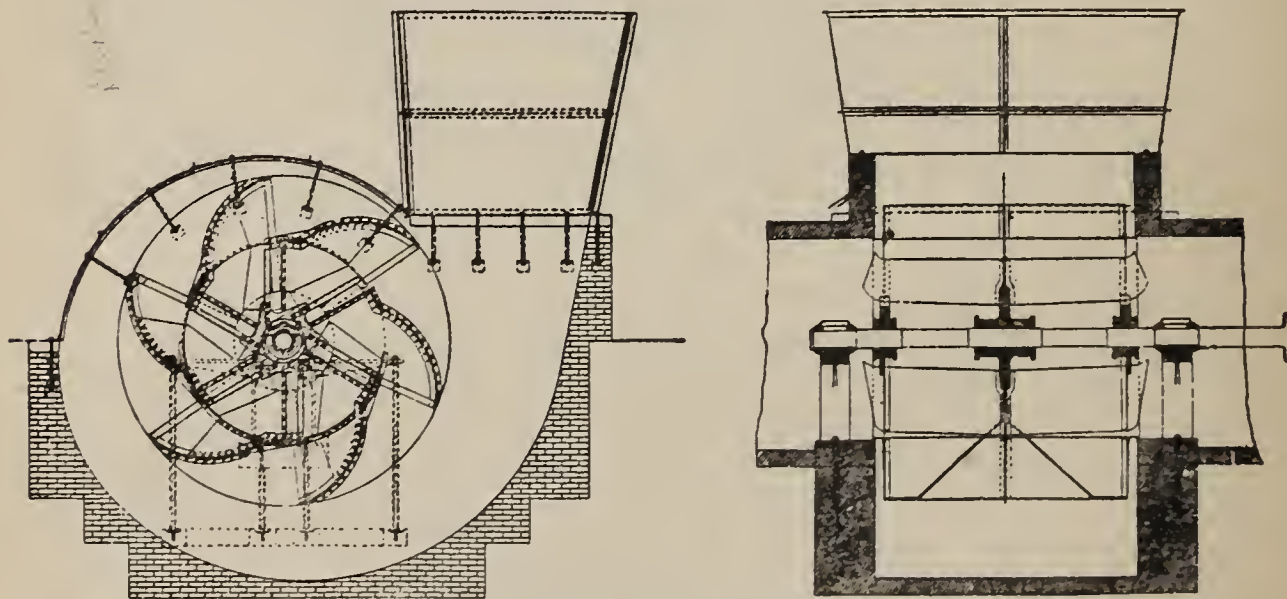
considerable portion of the shaft. Between the bosses are placed two discs of steel, of uniform thickness, bored in the centre to fit the fan shaft. Between these two discs, and gripped tightly by them, are fixed the iron arms of the fan in pairs. These arms extend from near the axis of the fan to its periphery, being supported half-way by the discs. In the small spaces between the discs which are not occupied by the fan arms, there are inserted annular plates. The whole portion outside the boss is then securely riveted together.

Angle irons are riveted to the fan arms where they extend beyond the discs, and to these, eight in number, are firmly secured the cross-section of the arm and vane in the form of a letter T, the top of the T representing the vane and the surface pressing against the air. The vanes, which spring tangentially from a small circle concentric



FIGS. 592 and 593.—Cappell fan.

with the fan shaft, are curved longitudinally to the arc of a circle of a certain radius, and are cut away from the edge of the inlet to the fan shaft to minimise central resistance. It is very necessary to minimise the slipping of the air between the sides of the vanes and the walls of the fan chamber. The vanes cannot be brought too



FIGS. 594 and 595.—Latest type of Cappell fan.

close to the walls, as, in the event of any side movement, they might catch and be injured. This clearance space is, therefore, filled up by attaching strips of pliable hoop iron to the sides of the vanes.

The fan is also fitted with the Walker anti-vibrating shaped shutter. In the ordinary Guibal fan, as each blade or vane passes the lower edge of the shutter, a pulsatory action of vibration takes place. This vibration is caused by the too abrupt cessation of the delivery of the air from the fan vanes or blades as they pass the

opening to the chimney, and for this the shape of the regulating shutter is responsible. The upper part of this opening, formed by the shutter as hitherto constructed, has a line parallel to the tips of fan vanes, and as the fan revolves these lines become identical; the delivery of the air is, as a consequence, abruptly interrupted. While discharging the air, the pressure is against the front of the vane, but immediately the latter enters the fan casing the load upon it is suddenly removed, and the pressure, owing to the vacuum within the casing, is instantaneously reversed, and causes an upward rebound of the previously depressed blade, with the result that a dangerous degree of vibration is set up. The Walker anti-vibrating shutter, as attached to the fan, removes this evil by effecting a perfectly gradual change in the pressure on the vanes, and so governs the discharge of the air as to cause it to pass, without objectionable

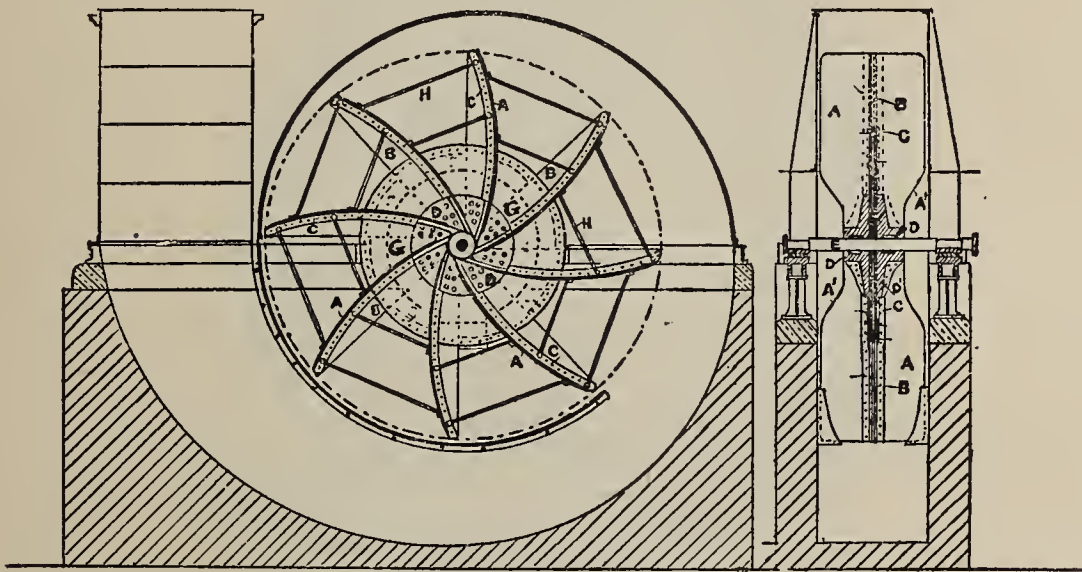


FIG. 596.—Walker fan.

eddyling, in a continuous stream from the fan vanes into the chimney instead of intermittently, and without the pulsatory action described. The shutter is constructed in sections, any of which can be removed for the purpose of adapting the area of the opening to varying duties of the fan. The fan is usually driven by rope gearing.

'Sirocco' Mine Fan.—The wheel of this fan, which is sold under the trade mark name 'Sirocco,' is constructed on an entirely different principle to that of other fans, the chief distinctive feature being the number and formation of the blades. These blades, of which there are 64 in a single-inlet, and 128 in a double-inlet fan, are very shallow radially, but very long axially. Their outward edges are curved forward in the direction of rotation, while the air passages between the blades are open at the ends towards the inflowing air. Owing to their formation and arrangement, the 'Sirocco' fan possesses the remarkable peculiarity that the velocity of the air issuing from the discharge exceeds the circumferential speed of the blades by about 80 per cent., whereas in other types of fans it

seldom equals and is generally less than the circumferential speed of the blades. Consequently, for a given duty, 'Sirocco' fans can be run at much lower speeds than other makes, or, at a given speed, are capable of imparting a much higher velocity to the air than other fans of equal diameter. This feature, and the fact that the inlet opening to the fan is approximately equal to the discharge, accounts for the 'Sirocco' fan being so small in size in comparison to its capacity.

The small diameter and comparatively high speed of revolution make it possible in many cases for the mine fan to be direct-coupled to modern high-speed engines and motors of high efficiency,



FIG. 597.—Single-inlet 'Sirocco' mine fan.

thereby eliminating the rope drive, which absorbs about 8 per cent. of the total power, and greatly improving the combined efficiency of set.

But notwithstanding that the fan works at a high speed of revolution, its peripheral speed is very much lower than that of other fans, with the result that the mechanical strain on the wheel is at least 25 per cent. less than in other types of fans. In fact, the peripheral speed is within 5 per cent. of the theoretical peripheral speed, as calculated by Murgue.

To comply with the new mining regulations the makers can arrange this fan to operate either as an exhaustor or blower, and thus reverse the direction of the air-current in the mine when desired.

The chief points in the special construction of 'Sirocco' centrifugal fans may be named as follows: The blades are very numerous, with their radial measurement (relatively to the diameter of the fan) very

shallow, and their axial measurement very long. Their outer edges are curved forward in the direction of rotation, and the air passages between the blades are usually open at the ends toward the inflowing air. The inlet for admitting the air to the fan and the outlet for its discharge are approximately of equal diameter to that of the fan itself. All these features constitute practically a reversal of previous theory and practice in regard to fan construction, but nevertheless the actual effect of this design is that the volume of air discharged per revolution is greater than in any other type of centrifugal fan of equal diameter.

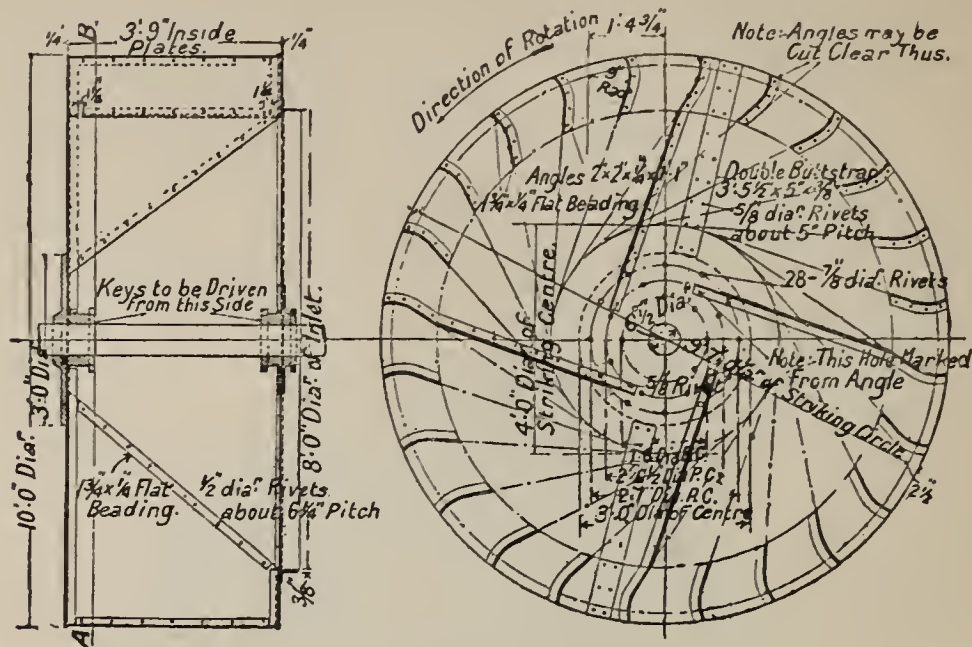
The inlet and discharge openings being relatively to the fan diameter about four times larger in area than in an ordinary fan, the frictional resistance to the passage of a given volume of air per minute through the 'Sirocco' is therefore only one-sixteenth that of other fans; thus, so far as fan resistance is concerned, a higher efficiency in actual work done for the power applied is obtained. The velocity of air into the inlet opening of the fan is almost equal over the whole area of the intake, and it is claimed by the makers that, owing to the adjustment and arrangement of the outer edges of the blades relatively to the inner edges, the velocity of the air issuing from discharge or delivery pipe of the fan exceeds the circumferential speed. But the duty of these fans is so good, in spite of the apparent reversal of general practice, that we hesitate to deny the truth of the contention.

The construction of the blades obviates the formation of eddies of air near the inlet. The periphery speed of the inner edges of the blades being only slightly less than that of the outer edges, the difference between the diameters of the respective circles is unusually small. It is well known that these internal eddies give rise to considerable noise with high velocities, but the 'Sirocco' is almost a silent fan, even at high speeds.

Barclay Fan.—This is a fan somewhat similar to the 'Sirocco' in construction, designed and built by Messrs Andrew Barclay, Sons, & Co., Ltd., Kilmarnock. It is built in sizes from 5 to 20 ft. diameter, the most usual sizes installed being 5 to 10 ft. diameter. The latter size of fan has twenty-four blades slightly curved to a tangent of the ear of the fan. The periphery is strengthened by four stiffening straps extending from the cheek to near the centre. A considerable number of these fans have now been installed at various collieries in Britain, designed to pass from 20,000 to 250,000 cub. ft. of air per minute, with ordinary and high water-gauges.

A series of interesting tests were made on two Barclay fans, one 7 ft. diameter by 7 ft. wide, and the other 10 ft. diameter by 7 ft. wide, by Mr David M. Mowat, M.E., at Bardykes Colliery, Blantyre, the full results of which are embodied in a paper read by him before the Mining Institute of Scotland (see *Trans. Inst. of Min. Engs.*, vol. xliv. pp. 100–101), and to which the student is referred

for detailed information. The 7-ft. fan was run at speeds varying from 215 to 364 revolutions per minute, and at the latter speed gave 153,800 cub. ft. of air per minute, with a water-gauge of 1.375 in. At



FIGS. 598 and 599.

the different speeds the manometric efficiency varied from 0.21 to 0.52, and the fan efficiency from 24.81 to 64.2 per cent. The larger fan, running at 216 revolutions per minute, gave 165,974 cub. ft.

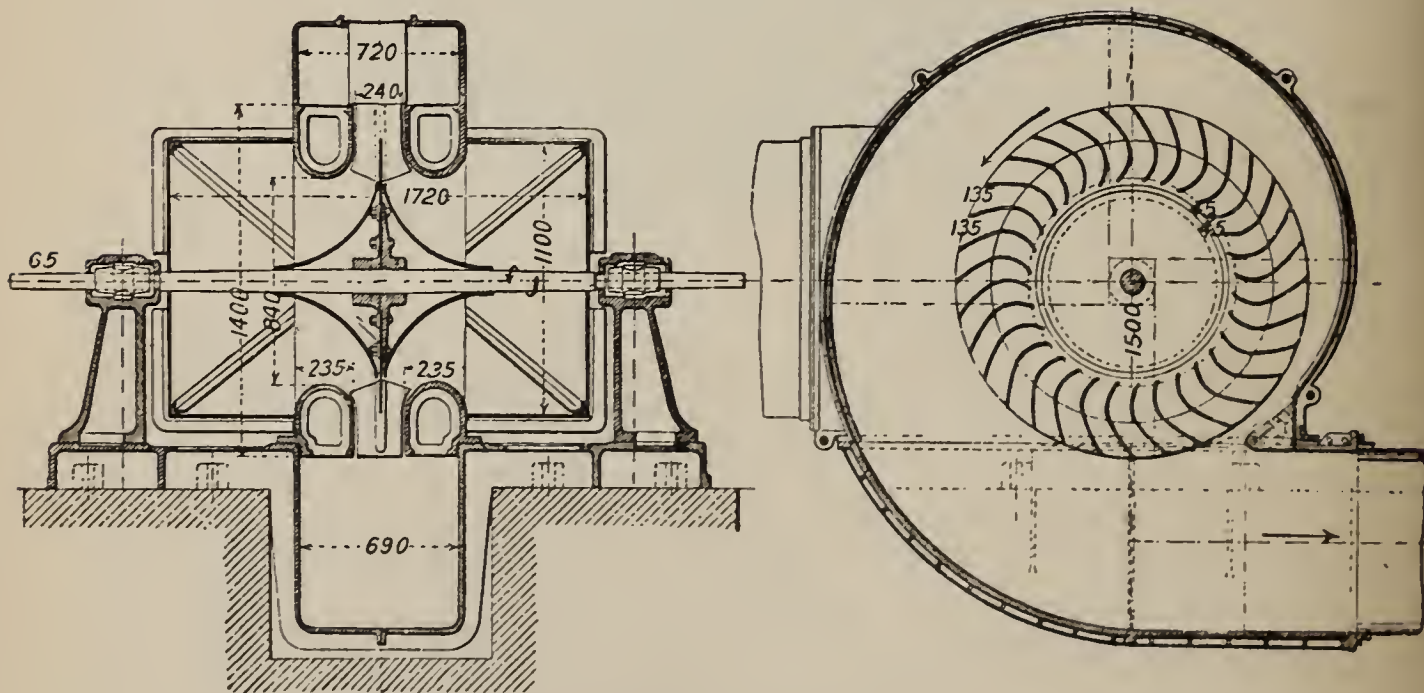
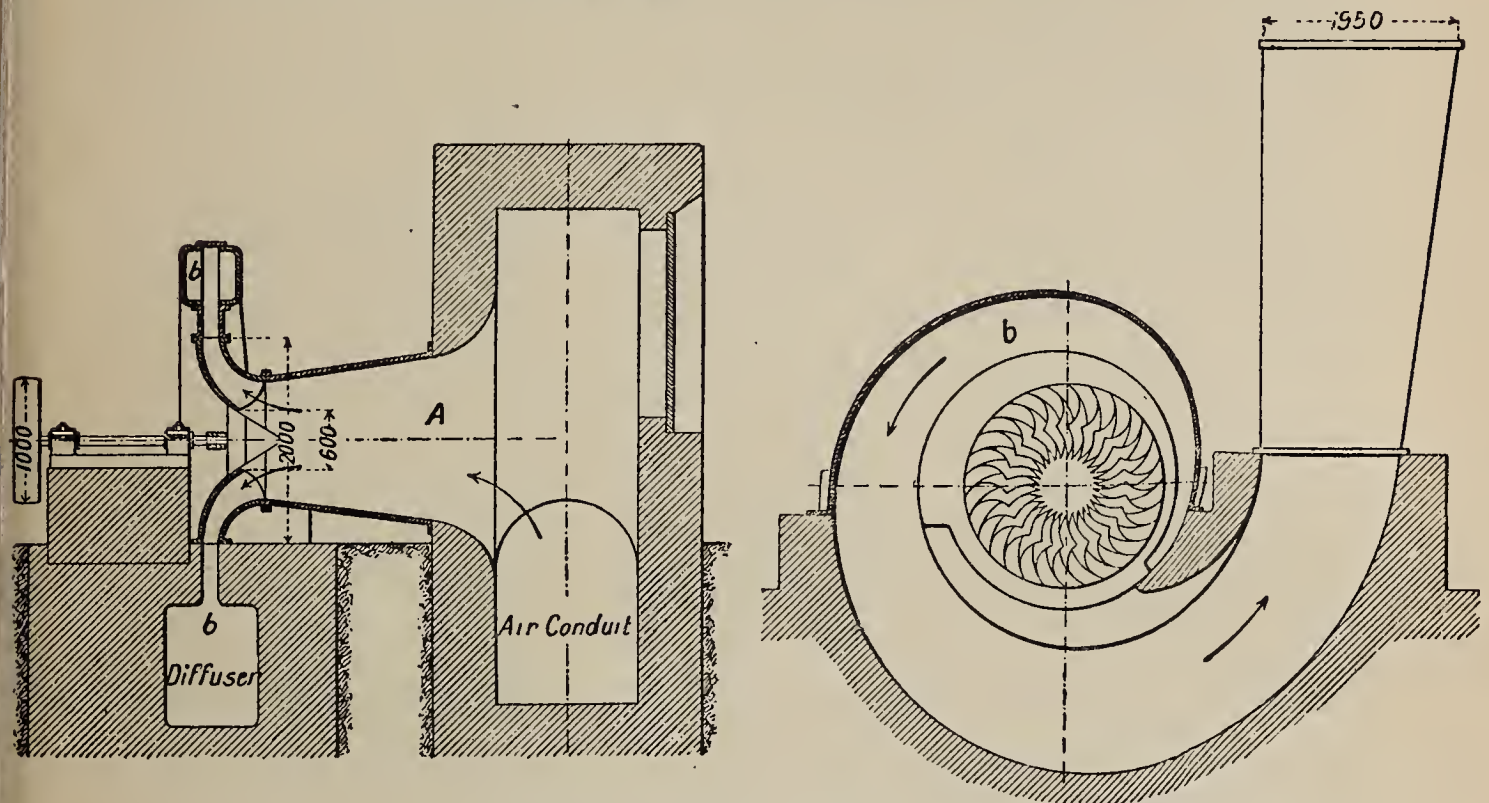


FIG. 600.—Ser fan.

of air per minute, the manometric efficiency being 0.438 and fan efficiency 50.20 per cent., with a water-gauge of 3.3 in.

Ser Fan.—This fan is largely used at French mines and is not unlike, in some respects, the 'Sirocco' fan. It has almost entirely displaced the large Guibal fans in France, especially in collieries

having small equivalent orifices.* It is fitted with a double inlet, with intake cones, and has a large number of vanes (thirty-two or more), curved forwards both inside and out, and provided with a spirally widening case and a diverging flue or chimney. The sudden widening of the fan diameter at the end of the vanes, where the spiral delivery chamber commences, will have an unfavourable effect on the efficiency of the fan. The angle made by the ends of the vanes with the peripheral tangents is 135° , whilst the angle at the base of the vanes (at the intake) is 45° . From tests made, a Ser fan 8 ft. diameter, running at 210 revolutions per minute, gave 93,480 cub. ft. of air per minute, with a water-gauge of 3.15 in.



FIGS. 601 and 602.—Rateau fan.

Rateau Fan.†—This is another fan that is largely used on the Continent, and first installed at a colliery in Belgium. It is fitted only with a single-inlet orifice about 4 ft. diameter. The bottom of the fan is of conoid form, and may be regarded as describing an arc about the axis of the fan. At the rim the fan bottom is perpendicular to the axis, and towards the inlet it is prolonged into a conical hood. The conical inlet (A, fig. 601) connecting the fan with the air-drift facilitates the entrance of the air and prevents vertical currents. The vanes are of a peculiar shape, are of steel, stamped in a hydraulic press and curved forwards both at the periphery and on the inner edge, just as in the Ser fan. This shape enables the air to enter without concussion, and also increases the rarefactive power

* *Ventilation in Mines*, by Robert Wabner, p. 189. Scott, Greenwood, & Co., London.

† *Ibid.*, p. 190.

(depression) of the fan. The outer edges of the vanes move close in front of the casing which joins the inlet A. The upper part of the casing is of cast iron, the lower portion of brickwork. The inner portion of the casing acts as a diffuser for reducing the velocity of the discharged air, whilst for the same purpose the outer portion widens spirally towards the chimney. This arrangement enables the kinetic energy of the air to be almost entirely converted into pressure. The angle made by the vane ends with the radius is 45° . The Rateau fan is said to give a high working efficiency.

A test made with a fan 6.78 ft. diameter, running at 184 revolutions per minute, gave an output of 70,577 cub. ft. of air per minute; water-gauge 1.73 in.; equivalent orifice 19.78 sq. ft.

There are a large number of other fans in use in Britain and on the Continents of Europe and America, but space cannot be given here for a description of them.

Selection of a Fan.—It is desirable, when erecting a fan to do a given amount of work, to select it so as to suit the varying conditions under which it must act. The height and condition of the underground roads are very important factors in determining the type of fan to be selected, also the amount of room at disposal for surface arrangements; but the principal considerations are: (1) the useful work or effect given out by any fan or engine; (2) the first cost of fan and engine, and of foundations, engine house, fan drift, etc.; (3) the space required at the surface for fan and engine houses, etc.; (4) the relative economy of fuel and stores consumed by the different types of fans; (5) the cost for repairs and freedom from stoppages and breakdowns.

In coming to any decision, especially on the basis of a maker's estimate, it is nearly always safe to make a deduction of 25 to 30 per cent., taking the amended estimate of the useful work that will be performed, for afterwards, in practical working, it may be found that the maker of the fan has overestimated the merits of his machine by that amount, and it is always better in any case to have a good deal of surplus power in case of an emergency.

Medium-sized and small fans, driven at a higher speed than is possible with the larger type of fan, are now being largely used, and have a good deal to recommend them, as they are economical and safe in working and take up little space. It is now not uncommon, both in Britain and on the Continent of Europe, to find fans working with water-gauges of 6 to 10 in. Where large volumes of air are required with a low water-gauge, a fan of large volume will be most suitable. The opinion is now generally held that the enclosed type of fan is superior and more efficient for mine ventilation than the open-running type like the Waddle fan. In addition to having an efficient fan, there should also be an efficient engine or motor to drive it, for, however highly efficient the fan may be, it will become practically valueless, so far as economy is concerned, if driven by an

inefficient engine or motor. For instance, a fan might be erected to give 70 or 80 per cent. of useful effect, but the efficiency might be entirely neutralised by attaching to it an engine and boiler consuming, say, 15 lbs. of coal per horse-power per hour in the air. Another fan of inferior design might yield the same results if attached to a highly efficient engine and boiler consuming 5 lbs. of coal per horse-power per hour in the air; *i.e.* one fan might consume coal to the value of £400 per annum, while an efficient fan, in consequence of being attached to an inefficient engine and boiler, might consume £1200 of fuel per annum.

It is desirable in most cases, and in fiery mines almost indispensable, to have a duplicate set of engines, either of which can be immediately attached to the fan in case of a breakdown. Sometimes a duplicate fan is set up, both fans being connected to the fan-drift, so that either can be worked separately for a week or a fortnight at a time.

Compressing Fans.—Fans, as already noted, may be either compressing or exhausting. The former require less power to drive them than the latter. The exhaust fan rarefies the air in the interior of the mine and accelerates the escape of gas from the waste workings. The compressing fan, on the other hand, renders the air in the mine denser and lessens the escape of gas. A falling barometer would intensify the evil of using an exhaust fan by rarefying the air to a still greater extent. At the same time, it is safer to employ a fan which promotes the escape of gas from the workings, provided there is an ample supply of air available with which to dilute it.

In the case of a colliery where choke-damp is given off in the workings, it might be advisable to use a compressing fan to prevent the escape of this gas into the working faces.

At the Clyde Collieries, Hamilton, experiments were made a good many years ago with compressing and exhausting fans, the results of which gave an apparent advantage to the compressing type, but the experiments were not carried out fully enough to allow of definite conclusions being arrived at.

There are cases, however, where compressing fans have been used to advantage, and give a larger quantity of air than an exhaust fan under the same conditions. One such case, with which the writer is acquainted, was at a small colliery where there were two shafts, one about 90 fms. deep and the other 50 fms.; and as they were separated by a distance of 800 or 900 yards, and no coal was being drawn from the deepest shaft, it was desired to remove the fan which was on this pit to the other one, to save boiler power. This was done; and when it was fitted up, at the 50 fms. pit, as an exhaust fan, it was found to be totally inadequate to supply the quantity of air required. It was then altered to a compressing fan, making the deep pit the up-cast, with the result that nearly 50 per cent. more air was circulated in the workings. In this case it was a distinct

advantage to have the fan forcing instead of exhausting the air; but the superiority of the compressing fan in this instance may have been largely due to the assistance it received from natural ventilation, owing to the deeper shaft having been made the up-cast, and the apparent inferiority of the exhausting fan may have been due to the fact that it had to draw the air from the deep shaft through a long stretch of old workings. If a forcing fan is used, the down-cast shaft must be covered over, which renders it inconvenient for winding, pumping, and haulage ropes, etc. Again, in the event of an explosion occurring, the disastrous effects and force of the explosion are nearly always felt most keenly in the down-cast shaft; and if the ventilation was carried out on the compressive principle, the fan would in all probability be wrecked by the force of the explosion, putting a stop to the circulation of air when it was most required. On the other hand, with an exhausting fan placed on the up-cast shaft, an explosion might occur and the fan remain intact, which is a great advantage under such circumstances. Taking everything into account, and the fact that a great majority of collieries are ventilated on the exhaustive principle, it would appear that the exhausting fan is the most suitable for general colliery work.

Reversing the Air-Current.—Within the last few years a great deal has been written and spoken on the reversing of the air-current in the event of an explosion underground or fire breaking out in the workings. The idea of reversing the air-current is to prevent the deadly after-damp resulting from an explosion of fire-damp, or gases from a fire, being driven forward on to workmen who may be employed on the return side of the place where the explosion or an underground fire has originated. The whole subject of reversing the air-current is a very difficult one, for while the reversing must be done at the surface, it might be hours after an explosion has taken place or a fire broken out before the exact locus could be determined. To reverse the air-current without first ascertaining the exact conditions prevailing after an explosion or a bad gob fire might increase danger instead of minimising it, and might lose lives instead of saving them. For instance, an explosion or fire might originate in a place near to the end of the return current, and to immediately reverse the air without investigation would be to bring about disaster. Unless an explosion or fire occurred on the main roadways or near the shafts, where the danger zone could be easily located, there are very few instances where reversal of the air-current could be carried out with safety, or be of the slightest use in saving lives.

In most cases it will be a very risky experiment, a risk which few colliery managers will care to take. Apart from that, the Coal Mines Act, 1911 (Sect. 31 (3)) enacts that adequate means for reversing the air-current must be provided and maintained in a condition to be put into immediate operation. The method of

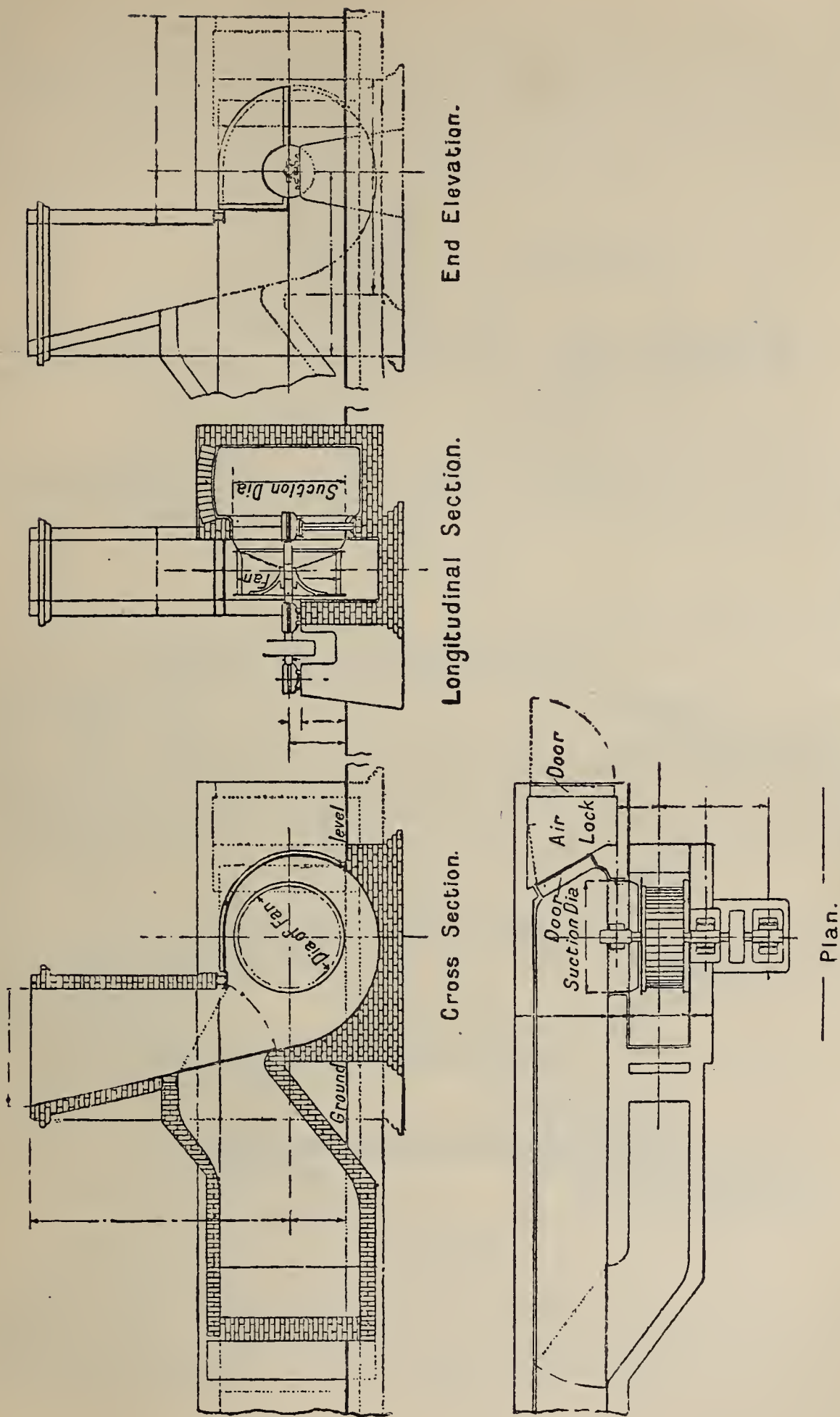


FIG. 603.—Showing arrangement of reversing air-current,

reversing the air-current will depend on the type of fan in use, each type requiring a different arrangement to suit the conditions. The usual method, with enclosed types of fans, is to employ auxiliary doors, so that by the opening or closing of these doors the fan can be made to act as an exhaust or forcing fan when required.

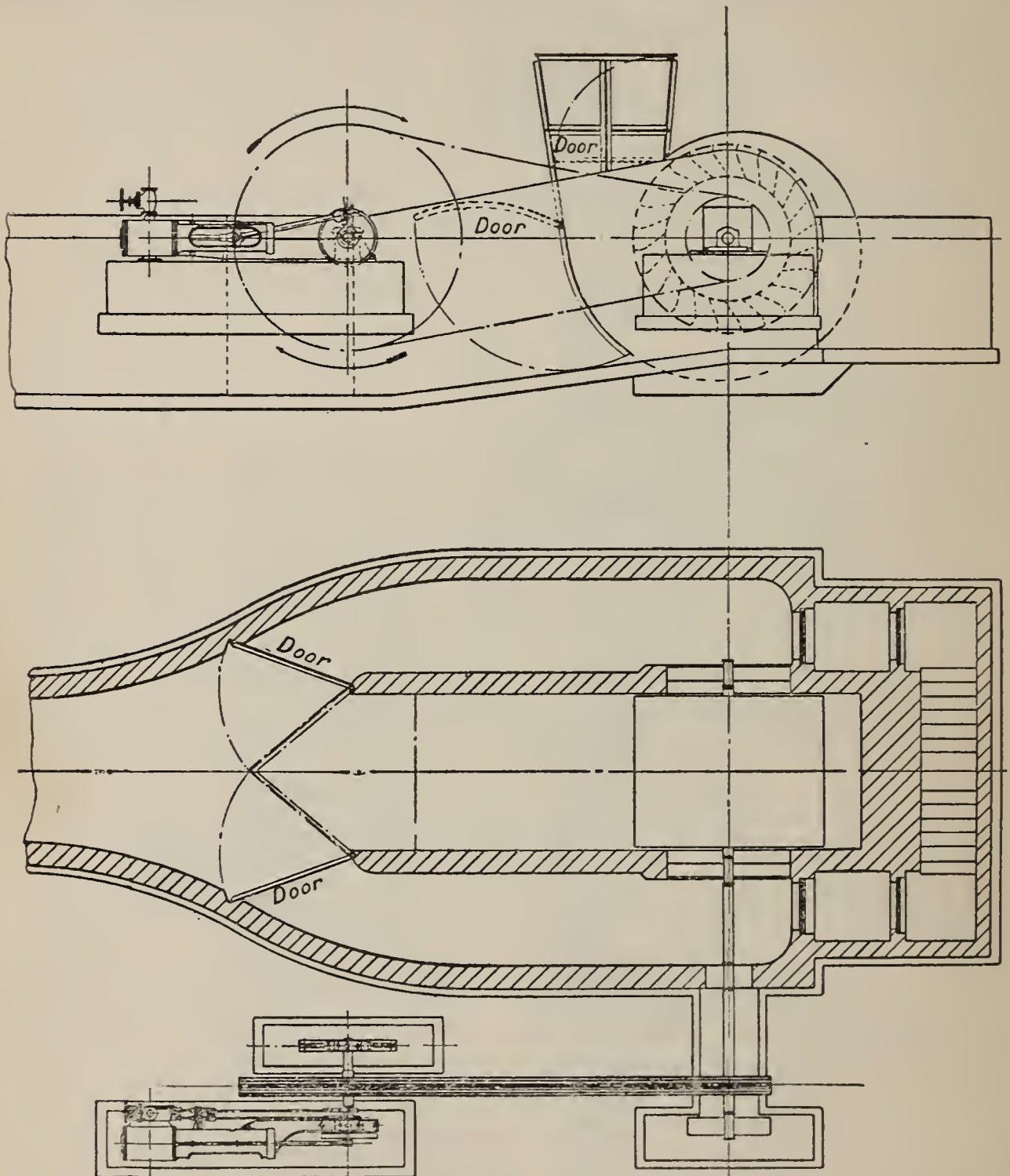


FIG. 604.

Fig. 603 shows a method of reversing the current with a single-inlet fan, and fig. 604 illustrates a method of reversing with a double-inlet fan, the arrangements of which will be understood from the diagrams.

Equivalent Orifice.—The theory of the equivalent orifice of a mine was first investigated by the French engineer M. Murgue, and may be stated thus:—The

air passing through the workings of a mine meets with a certain amount of resistance. If we imagine that all the air is entering, not through the mine workings, but through an orifice or aperture in a thin plate, then the equivalent orifice for any mine would be the aperture in that plate, *of an area such that a resistance would be offered to the entrance of the air equal in amount to that caused by the mine workings.*

If we call Q the quantity of air in cub. ft. per minute, h the water-gauge in inches, and A the area of equivalent orifice in sq. ft., then

$$A = \frac{0.00039Q}{\sqrt{h}} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (a)$$

$$Q = \frac{A\sqrt{h}}{0.00039} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (b)$$

$$\sqrt{h} = \frac{0.00039Q}{A}, \text{ or } h = 0.169 \left(\frac{Q}{A} \right)^2 \quad . \quad . \quad . \quad . \quad (c)$$

or, assuming the normal densities of the air and water to be in the ratio of 1.2 : 1000, and Q as before, = the volume of air in thousands of cubic feet per minute; h = inches of water-gauge, then

$$A = 0.403 \frac{Q}{\sqrt{h}}.$$

Quantity of Air delivered by a Fan.—The quantity of air delivered by a fan is not easy to ascertain, as no two fans, even of the same make, will give the same quantity, even when tried under the same conditions. A great deal depends on the condition of air-courses, type of fan, etc.

The height of the motive column H in feet of air, produced by a difference of temperature or by mechanical means, is the cause of air currents circulating in mines, and the velocity of such air currents (provided there is no resistance or friction) would be equal to that acquired by a body falling from the same height as that represented by the motive column $v = \sqrt{2gH}$, and therefore the theoretical volume of air passing through any opening (equivalent orifice) of area a will be

$$V = va, \text{ but as } v = \sqrt{2gH} \quad \therefore V = a \sqrt{2gH};$$

$$\text{but } H = \frac{u^2}{g} \quad \therefore V = a \sqrt{2g \frac{u^2}{g}} = a \sqrt{2u^2}$$

The theoretical volume produced per second, allowing for the action of the *vena contracta*, will be

$$V = 0.65a \sqrt{2u^2}.$$

Applying this to a fan with a tangential velocity in feet per second = u , and a manometrical efficiency = K ,

then $V = .65a \times \sqrt{2u^2} \times K.$

or if Q = total quantity of air in cubic feet per minute,

d = diameter of fan in feet,

R = revolutions of fan per minute,

u = tangential speed of fan in feet per minute = $d \times 3.1416 \times R$,

a = equivalent orifice of mine in square feet,

K = coefficient representing the manometrical efficiency,

0.65 = coefficient of the *vena contracta*,

then $Q = 0.65a \sqrt{2(d \times \pi \times R)^2} \times K = 0.65 \times 1.41\pi, a, d, R, K$
 $= 2.88 a, d, R, K$

$$\text{or } d = \frac{Q}{2.88 a, R, K}. \text{ This would apply to a fan}$$

with radial blades.

Example.—What would be the total quantity of air, in cubic feet per minute, produced by a Guibal fan 35 ft. diameter and moving at a speed of 40 revolutions per minute; the equivalent orifice of the mine being 30 square feet, K being = .69

$$Q = 2.88 \times 30 \times 35 \times 40 \times .69 \\ = 83,462.40 \text{ cubic feet per minute.}$$

Example.—What size of fan, making sixty revolutions per minute, and giving an efficiency of 60 per cent., would be required to pass 60,000 cub. ft. of air per minute, the equivalent orifice of the mine being 20 sq. ft.?

This problem may be worked out in two different methods, but we may proceed by the formula, $a = \frac{0.37Q}{\sqrt{h}}$.

To find the water-gauge—

$$20 = \frac{0.37 \times \frac{60000}{1000}}{\sqrt{h}} \quad \therefore 20\sqrt{h} = 22.20 \text{ and } h = \left(\frac{22.20}{20}\right)^2 = 1.23 \text{ ins.}$$

We may find the velocity of the periphery of the fan by formula $h = .00045 u^2$. Then

$$1.23 = .00045 u^2, \text{ or } u^2 = \frac{1.23}{.00045} \text{ and } u = 52.32 \text{ ft. per second.}$$

As the fan is making sixty revolutions per minute, the diameter, supposing its efficiency to be 100 per cent., would be

$$\frac{v \times 60}{R \times 3.1416} = \frac{52.32 \times 60}{60 \times 3.1416 \times .69} = 24.22 \text{ ft.}$$

But the fan has an efficiency of 60 per cent. only, and allowing .69 for manometrical efficiency,

$$\therefore \text{the diameter} = \frac{16.6 \times 100}{60 \times .69} = 40 \text{ ft.,}$$

so that to pass a volume of 60,000 cub. ft. per minute, with an equivalent orifice of 20 sq. ft., we would require a fan 40 ft. diameter, making sixty revolutions per minute.

Working it out by the more direct formula given above, we have

$$d = \frac{60000}{2.88 \times 20 \times 60 \times .69} = 25 \text{ ft.}$$

and allowing for 40 per cent. in loss of efficiency

$$d = \frac{25 \times 100}{60} = 41.6 \text{ ft.}$$

The following formula is given by a French mining engineer:

$$D = 60 \frac{\sqrt{g \frac{0.38Q}{a} + h}}{\pi R \sqrt{w}},$$

where Q = volume in cubic feet per second,

a = equivalent orifice in square feet,

h = depression, or water-gauge, in inches,

R = revolutions of fan per minute,

w = weight of cubic foot of air at normal temperature and pressure,

$\pi = 3.1416$.

It should be remembered that these formulæ only give approximate results and are not mathematically correct. A more exact approxi-

mation of the *actual* volume of air produced by a fan would be found by the formula

$$Q = 40u \sqrt{\frac{2Ka^2O^2}{a^2 + O^2}},$$

where a = equivalent orifice in sq. ft. and O = orifice of discharge of the fan in sq. ft., or an orifice representing the difficulty of the passage of the air through the ventilator.

The value of O is a variable quantity, depending on the equivalent orifice and the water-gauge, and may be found from the expression

$$O^2 = \frac{h_1 a^2}{h - h_1} = \frac{h a^2}{0.00045u^2 K - h}; \quad h \text{ and } h_1 \text{ being respectively the theoretical and observed water-gauges.}^*$$

Manometrical Efficiency.—The manometrical efficiency of a ventilator is the ratio of the head or water-gauge observed in practice to that calculated by the formula $h = \frac{u^2}{g}$; or it is simply the ratio between the observed and the theoretical water-gauge at the normal velocity.

The theoretical depression produced by a perfect centrifugal ventilator moving without friction calculated in height of motive column in feet, $H = \frac{u^2}{g}$, or in inches

$$\text{of water-gauge, } h = \frac{u^2}{g} \times \frac{d_1}{d_2} \times 12,$$

where u = peripheral velocity of fan in feet per second,

g = force exerted by gravity (32.2),

d_1 = mean density of air,

d_2 = „ „ water,

d_1 and d_2 are in the ratio approximately of 1.2 : 1000

$$\therefore h = \frac{u^2}{32.2} \times \frac{1.2}{1000} \times 12 \quad \left[\frac{1.2 \times 12}{32.2 \times 1000} = 0.000447, \text{ usually taken as } 0.00045 \right]$$

$$= 0.00045u^2.$$

Take the speed of fan periphery at 90 ft. per second, $H = \frac{90^2}{32} = 253.12$ ft., and with a column of 253.12 ft. of air, say at 70° F. and 30 in. bar., the pressure $= \frac{1.3253 \times 30}{70 \times 459} \times 253.12 = 19.02$ lbs. per sq. ft., and $h = \frac{19.02}{5.2}$, or 3.65 in. of water-gauge.

The value of H or h is not usually realised in practice owing to the various imperfections common to all centrifugal ventilators, and the equation becomes, for the value of the *initial* water-gauge produced when the fan is exhausting from a closed chamber adjoining the inlet,

$$h = K \frac{u^2}{g} \times \frac{d_1}{d_2} \times 12$$

$$= K \times 0.00045 \times u^2,$$

in which K is a fraction representing the maximum manometric efficiency realised

* Students who desire a fuller treatment of fan ventilation should consult *Theories and Practice of Centrifugal Machines*, by D. Murgue (E. & F. N. Spon, 1883).

by the ventilator ; the average value of K being, for Guibal fan, 0.69, for Waddle fan 0.46, and for Schiele fan 0.50.

If the ventilator be connected to a mine and run at a constant speed, and if it be assumed that the mine can be substituted by an orifice in a thin plate (equivalent orifice) whose area can be varied as desired ; if also the resistance of the air in passing through the ventilator be represented by an orifice O ,

$$\begin{aligned}\text{then } h &= K \frac{u^2}{g \left(1 + \frac{a^2}{O^2}\right)} \times \frac{d_1}{d_2} \times 12 \\ &= K \times 0.00045 \times \frac{u^2}{\left(1 + \frac{a^2}{O^2}\right)}.\end{aligned}$$

The manometrical efficiency is, therefore, not a constant expression, but varies with the resistance of the mine, the maximum being obtained when the orifice is at its minimum area or entirely closed, and decreases to zero when the area of the equivalent orifice is infinite. If all other conditions remain constant, the depression varies as the square of the peripheral velocity of the ventilator.

If h = theoretical depression or water-gauge, and h_1 = actual or observed water-gauge, then the manometrical efficiency $dH = \frac{h_1}{h}$. For instance, if the theoretical water-gauge = 2.05 in., and the observed water-gauge = 4.46 in., then $d = \frac{2.05}{4.46} = 0.459$; thus the manometrical efficiency would be 0.459, or $0.459 \times 100 = 45.9$ per cent.

The higher the manometrical efficiency is in the case of any ventilator the smaller is the peripheral velocity required to obtain a given result. A fan may, however, have a low manometrical efficiency and at the same time have a high mechanical efficiency.

The mechanical efficiency (E) of a ventilator is the ratio of the useful work (W_1), performed per second or per minute, to the work done upon the driving shaft (W_2) of the machine in the same time ; or simply it is the ratio of the work put in to the work got out ; hence $E = \frac{W_1}{W_2}$.

$$\begin{aligned}W_1 &= w \times H \times V \\ &= 0.075 \times 69.3 \times h \times V \\ &= 5.2 \times h \times V\end{aligned}$$

$$W_2 = W_1 + T + t$$

Where w = weight of 1 cub. ft. of air at shaft temperature = 0.075 lb. at 60° F.

H = height of column of air to exert same pressure as the water-gauge (1 in. = 69.3 ft. of air column).

V = volume of air in cubic feet per second.

T = work lost by friction and eddies of air due to the internal resistance of fan.

t = work lost by friction of shaft and bearings.

$$\therefore E = \frac{W_1 + T + t}{5.2 \times h \times V},$$

$$\begin{aligned}\text{or more simply, } E &= \frac{\text{H.P. in air}}{\text{H.P. in engine}} = \frac{5.2 \times h \times V}{D^2 \times 0.7854 \times P \times L \times 2 \times \frac{R}{60}} \\ &= \frac{33000}{33000}\end{aligned}$$

The amount of *useful effect* produced by a fan is found by carefully measuring the quantity of air put in circulation and measuring the water-gauge ; the H.P. in the air = $\frac{Q \times \text{W.G.} \times 5.2}{33000}$.

While the air measurements are being taken, the speed of the fan engine is

carefully noted and *indicator diagrams* taken, from which the mean effect of steam pressure acting in the cylinder is ascertained.

Then the H.P. of engine = $\frac{D^2 \times .7854 \times P \times L \times 2 \times R}{33000}$, where D is the diameter of cylinder of engine in inches ; P, the effective pressure of steam acting on piston in lbs. per sq. in. ; L, the length of stroke in feet ; and R, the revolutions of crank per minute.

Then the useful effect of fan will be as H.P. in air : H.P. in engine.
Example.—If the total quantity of air passing is 160,000 cub. ft. per minute with a W.G. of 2 in., the fan engine being 20 in. diameter with a 3½ ft. stroke, going at forty revolutions per minute, with an effective steam pressure of 40 lbs. per sq. in., what would be the efficiency of the fan ?

H.P. in air = $\frac{160000 \times 2 \times 5.2}{33000} = 50.42$

H.P. in engine = $\frac{18^2 \times .7854 \times 40 \times 3.5 \times 2 \times 40}{33000} = 86.36.$

The efficiency of the fan will therefore be as 50.42 : 86.36 or .58, which multiplied by 100 = 58 per cent.

The mechanical efficiency is one of the most important characteristic qualities of a ventilator, for if economy is effected in the work to be transmitted to the driving shaft, a resulting proportional economy in the coal consumed will be attained, with a corresponding reduction in the size of engine and horse-power of boilers required. The mechanical efficiency of a ventilator varies as the area of the opening, and if the opening remains constant, the useful work (W₁) is in proportion to the cube of the *peripheral velocity*. Thus if the peripheral velocity was increased from 40 to 50 ft. per second, the useful work (W₁) would be in the ratio of

40³ : 50³, or 64,000 : 125,000, or 1 : 1.95 ;

giving a gain of nearly 95 per cent. for an increase in velocity of 10 ft. per second. But the work lost by being transformed into heat is also proportional to the cube of the peripheral velocity, hence there is very little advantage to be gained by increasing the speed of a fan unless the equivalent orifice can be increased at the same time.

TABLE SHOWING MEAN COMPARATIVE PERCENTAGES OF USEFUL EFFECT FOR VARIOUS AREAS OF EQUIVALENT ORIFICE.

Type of Ventilator.	Percentages of Useful Effect for Areas of Equivalent Orifices of				
	10 Sq. Ft.	20 Sq. Ft.	30 Sq. Ft.	40 Sq. Ft.	50 Sq. Ft.
Guibal, . .	24.5	43.4	52.7	54.8	53.2
Schiele, . .	31.5	46.4	49.8	49.5	48.4
Waddle, . .	21.6	35.7	42.2	43.9	44.0

The foregoing table is important, showing, as it does, that the equivalent orifice has a great influence on the efficiency of a ventilating plant. It also goes to prove that the maximum useful effect may be realised upon mines whose equivalent orifice has an area of 35 to 40 sq. ft., as beyond this the useful effect begins to diminish, owing to a greater proportion of the power being absorbed in overcoming the internal resistance of the ventilator due to the larger volumes of air passing through it. For small equivalent orifices, the useful effect is low, and reaches zero when the mine is closed off from the fan, the work of the engine being entirely absorbed by the passive and other resistances of the ventilator and engine.

The quantity of air produced by a centrifugal ventilator, if the speed remains constant, varies (1) inversely with the resistance of the mine, and becomes zero when the resistance of the mine is infinite or when the inlet from the mine is completely closed; and (2) varies as the orifice of the fan (O) or the orifice of the mine (a) is increased or decreased. The quantity of air produced also varies with the speed of the fan, other conditions remaining constant. The above formula also demonstrates that the quantity of air produced by small fans can never equal that produced by large fans, as the orifice of the ventilator, which measures the resistance of the fan to the passage of the air, must necessarily be less for a small than for a large ventilator, hence better and more efficient results can be got by employing medium-sized fans, 20 to 30 ft. diameter, especially where the seams worked are a fair height and where the airways are of fairly good dimensions and can be kept in good order.

Capacity of Output of Air.—Capacity of output is simply the weight, in lbs., of air which passes through the ventilator per second.

Let x = weight in lbs. of air passed through the fan per second,
 w = weight of 1 cub. ft. of air at shaft temperature and pressure,
 Q = volume of air in cub. ft. per second,

$$\text{then } x = w \times Q,$$

$$\text{or } x = \frac{1.3253 \times B}{460 + t} \times Q = 0.0776Q \text{ approximately,}$$

where B = height of barometer in inches, and t = temperature of air in degrees F.

“The capacity of output varies from zero when the opening is nil to a maximum when it is infinite. Between these two limits there exists a capacity of output of special importance, which corresponds to the normal speed. Again, the theoretical centrifugal ventilator, free from friction and producing the theoretical depression, will produce volumes of air proportional to the *areas* of the *equivalent orifice*.” From actual practice it has been shown that the volume of air produced is highest from those of largest diameter in some fans,

like the Schiele, while in others it may decrease as the diameter increases, as in the Waddle.

TABLE SHOWING COMPARATIVE VOLUMES OF AIR BY SEVERAL VENTILATORS WHEN RUNNING AT THE NORMAL SPEED OF 6000 FEET PER MINUTE FOR VARIOUS AREAS OF EQUIVALENT ORIFICE.

Type of Fan.	Diameter of Fan.	Volumes of Air per Minute corresponding to Areas of Equivalent Orifices of				
		10 Sq. Ft.	20 Sq. Ft.	30 Sq. Ft.	40 Sq. Ft.	50 Sq. Ft.
	Ft.	Cub. Ft.	Cub. Ft.	Cub. Ft.	Cub. Ft.	Cub. Ft.
Guibal, .	30	41,750	83,750	122,000	156,250	190,000
Waddle,	30	36,700	73,250	108,500	142,750	175,750
Guibal, .	36	43,250	86,500	129,750	170,000	205,250
Schiele, .	12	38,250	74,000	107,500	138,750	169,000
Waddle,	45	38,250	74,000	107,000	138,500	169,750
Schiele, .	15½	40,500	77,750	111,750	145,500	177,750

Those bracketed together were working on the same mine under similar conditions.

The above table shows the importance of having large equivalent orifices, which has more influence on the capacity of output than the size or speed of fan.

Characteristic Curve.—The capacity of output, the mechanical efficiency, the manometric efficiency, and the reduced opening (gradually decreasing equivalent orifice) are simply coefficients. They have certain relations amongst themselves which are not easily explained. We know, however, that the volume of air increases as the depression falls when the equivalent orifice of the mine becomes larger. This will be made clearer if a series of increasing values of the equivalent orifice are plotted as abscissæ, and the corresponding mechanical efficiencies, outputs of air, and manometrical efficiencies plotted as ordinates, three curves are obtained whose positions are fixed for each type of fan. This would be termed the characteristic curve of that type. The nearer the curve approximates to a straight line the more efficient will the ventilator be.

Speed of Fan.—This will altogether depend on the size and weight of the fan, for there is a limit to the angular velocity at which a fan may be driven, and to go beyond which would be dangerous. As a rule the velocity of the periphery may vary from 90 to 110 ft. per second, the latter being a practical limit for medium or large-sized fans.

Advantage of Fans over Furnaces.—(1) They will not ignite gas ; (2) are more under command and better suited for repairs ; (3) the quantity of air passed is more regular and more easily varied at will ; (4) less cost incurred for upkeep and attendance, and smaller consumption of fuel, stores, etc.

Cost of Fans. — The price of fans is a very variable item, but the following are taken from some makers' estimates for fans to deliver 120,000 cub. ft. of air per minute, with a water-gauge of 3 in.*

<i>Schiele Fan</i> , 10 ft. diameter, with engine 20 in. diameter × 24 in. stroke, with driving belt, etc.,	£600
<i>Waddle Fan</i> , 21 ft. diameter, with engine 20 in. diameter × 21 in. stroke, complete,	590
<i>Guibal Fan</i> , 27 ft. diameter, with engine 18 in. diameter × 36 in. stroke, complete,	350
<i>Chandler Fan</i> , 13 ft. diameter, with patent silent running Chandler engine,	700
<i>Walker's Patent Fan</i> , with patent anti-vibration shutter, engine 16 in. diameter × 14 in. stroke,	503
<i>Cappell Fan</i> , 12 ft. × 5 ft. 6 in. open running, with semi-compound engine 16 in. diameter × 36 in. stroke,	360

In addition to the above prices, the brickwork for a Guibal fan would cost on an average £130 to £150, and that for a Cappell fan about £70.

Auxiliary Underground Fans:—In underground work it frequently happens that long close or narrow drifts require to be driven in the solid coal or in stonework, such drifts being only in communication with the main ventilating current at one end. When this occurs the drifts are difficult to ventilate—especially if they are long and fire-damp is given off—even with the aid of an efficient mid brattice. Under such circumstances a small auxiliary fan may be employed to assist the ventilation. If the quantity of air required for such work is not large, then a small fan rotated by manual labour may be employed, but if the drift is of considerable length and requires a larger quantity of air to keep it sufficiently ventilated, a fan of larger dimensions, say 2 to 3 ft. diameter, will require to be used, driven either by a compressed-air or electric motor. The latter is the preferable method, owing to the ease with which a small motor can be installed at any position desired, and the higher efficiency which can be got from it, compared with a compressed-air motor. The fan may be connected to the drift by either a dividing wood or brick brattice, wooden boxing, or large sheet-iron circular tubes, 1½ to 2 ft. diameter.

There are many occasions when the judicious application of underground fans, arranged to act in conjunction with the main

* These prices were quoted in 1896 ; in 1900 they were 20 to 25 per cent. higher. At the present date (1913) they may be taken about the same price as for 1900.

surface fan, can have a most favourable effect in the proper ventilation of the workings and on the running costs of the ventilating plant. In driving long stone drifts, or long narrow roads in solid coal, it becomes difficult to carry in sufficient air by means of the ordinary system of bratticing. Again, when the workings extend a long distance from the shaft there is a great amount of leakage between the intake and return airways, and it is difficult to carry forward the necessary quantity of air to maintain sufficient purity of the atmosphere and to dilute and render harmless the gases given off in the workings. It is in such conditions as these that an auxiliary fan placed underground can be used to great advantage. In the case of very extended workings the auxiliary fan should be placed as near as possible to the far end of the return airway. Fig. 605 shows an arrangement for installing a small auxiliary fan in

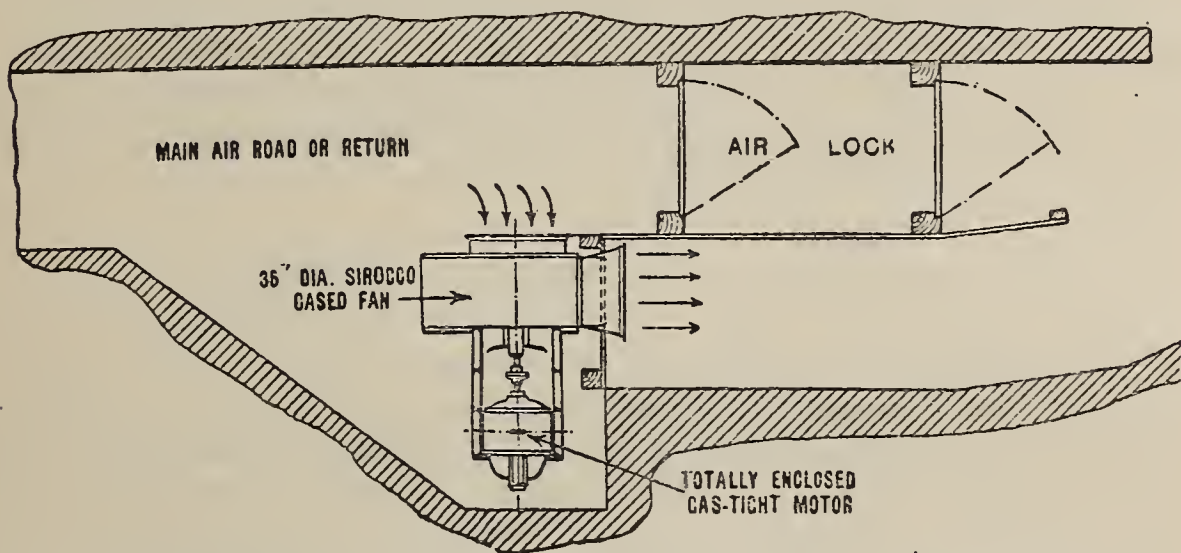


FIG. 605.—Showing arrangement of auxiliary fan underground.

such circumstances. Small fans driven by manual labour have been frequently employed for augmenting the ventilation under certain conditions, but this system is at best very inefficient, and can only be looked upon as a temporary expedient. Where compressed air or electricity is used underground it is easy to operate the fan by one or other of these powers. The installation of such auxiliary underground fans has also other advantages which should not be overlooked. When the workings are remote from the shaft, especially in thin seams, it will be necessary to operate the surface fan at a higher speed to produce the necessary water-gauge. A small auxiliary fan can be used in such circumstances to 'boost' up the air-current where it is needed, and it will then only be necessary to run the main ventilator on the surface at a lower speed to maintain the water-gauge and meet the requirements of the other parts of the mine. By thus reducing the speed and water-gauge at the main ventilator, surface leakage, which is considerable at high water-

gauges, particularly where the up-cast shaft is used for winding purposes, is reduced to a minimum. In one instance where an auxiliary fan was installed underground the leakage was reduced 50 per cent. Far too little attention is frequently paid to the question of leakage at the surface in the up-cast shaft. Many up-cast shafts are protected by timber casing, which in time gets worn and causes a great deal of leakage. The covering or lifting doors at the pithead level are also too frequently of very faulty construction and not designed to reduce leakage to a minimum. In tests which the author has carried out he has found the leakage (*i.e.* the difference between the quantity circulating in the fan drift and in the up-cast shaft) amount to 40 to 70 per cent of the total quantity of air passing. This represents a very serious loss of power and money.

All up-cast shafts should be surrounded with a good brick structure (brickwork set in cement), a good type of lifting doors or covers provided, and a double set of doors in the air-lock. If these details were attended to, it would mean some little outlay of money, but it would also mean a greatly increased efficiency of the ventilating plant and lower running costs.

Method of Driving Fans.—Fans working on the surface may be driven either—(a) direct; (b) by belting; (c) by ropes.

Large fans of the Guibal or Waddle type, which are run at moderate speeds, are usually driven direct; *i.e.* the connecting rod of the engine is connected direct to the fan shaft through a crank or disc, and this, on the whole, gives good results and saves the extra cost of driving pulleys and belts or ropes. Small quick-running fans, of the Cappell or Schiele type, are usually driven by belts or rope gearing, the latter giving the best results, for although ropes are more expensive in first cost, they wear much better and are not so apt to give way as belting. The ropes may be either made of hemp or cotton, the latter being now largely employed.

Tests of Fans.—In putting down a new fan it is necessary that a full test should be made to ascertain if all the requirements of the specification are carried out by the makers of the fan and engine. In order to do so the following data will be required to arrive at results:—

I. *Dimensions of Fan:*

Type (Guibal, etc.) fan.

Diameter,	.	.	.	.	ft.	in.
-----------	---	---	---	---	-----	-----

Width at inlet,	.	.	.	.	„	„
-----------------	---	---	---	---	---	---

„ periphery,	.	.	.	.	„	„
--------------	---	---	---	---	---	---

Mean speed of fan, revolutions per minute.

Normal velocity (usually taken at 100 ft. per second).

II. *Dimensions of Engine :*

- No. of cylinders.
- Diameter of cylinders.
- „ piston rod (back end).
- „ „ (fore end).
- Mean effective steam pressure on piston (got from indicator diagrams).
- Type of drive (rope, belt, or direct).
- Ratio of transmission.
- Mean revolutions of engine, per minute.

III. *Dimensions of Shafts :*

- | | | | | |
|---------------------------|---|---|------|-----|
| Depth of down-cast shaft, | . | . | fms. | ft. |
| „ up-cast „ | . | . | „ | „ |
| Size of down-cast shaft, | . | . | „ | „ |
| „ up-cast „ | . | . | „ | „ |
- Difference in surface levels of shafts.
 - Dimensions of fan drift.
 - Sketch plan of shafts, with obstructions, etc.

IV. *Temperature and Pressure :*

- Temperature of atmosphere.
- „ in fan drift.
- „ at bottom of shafts.
- Pressure of atmosphere ; inches of mercury.
- Water-gauges in inches.

V. *Other Data :*

- Number of tests.
- Duration „
- Volumes of air in cubic feet per minute.

In making a test a high-speed standardised anemometer should be employed, and two or three water-gauges fixed at different parts of the drift, the mean of these being taken. It is usual to divide the fan drift into a number of squares by means of wires or cords, and a reading of the anemometer taken in each square for $\frac{1}{2}$ or 1 minute. A reading should also be taken over the whole of the sectional area, say for 2 or 3 minutes, moving the anemometer in a sinuous path all over drift. This will give a check on the other readings.

The party making the test should have an assistant to take measurements, take the time of each reading, etc. At the same time as the air volumes are being measured indicator diagrams of the engine should be taken, or, if electricity is the motive power, readings taken by a watt-meter, to ascertain the consumpt of power.

The following tables show the results of a test made by the author on a Waddle fan, with explanatory notes :—

TABLE I.—SHOWING RESULTS OF TEST OF WADDLE FAN.

Number of Test.	Revolutions of Fan per Minute.	Area of Passage in Square Feet. (a).	Velocity of Air in Feet per Minute. (v).	Theoretical Depression in Inches. ($h = .00045 v^2$).	Observed Depression in Inches. (h_1).	Equivalent Orifice in Sq. Ft. $O = .00039 \frac{Q}{\sqrt{h_1}}$.	Total Quantity of Air in Cubic feet per Minute. ($Q = v \times a$).	Horse-power in Air. ($H.P. = \frac{Q \times h_1 \times 5.2}{33,000}$).	Indicated H.P. of Engine. $H.P. = \frac{d^2 \times 7854 \times P \times L \times 2 \times R}{33,000}$.	Combined Mechanical Efficiency of Fan and Engine. ($E_1 = \frac{H.P. \text{ in Air}}{H.P. \text{ in Engine}}$).	Manometrical Efficiency of Fan. ($E_2 = \frac{h_1}{h} \times 100$).
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)
										Per Cent.	Per Cent.
1	92	...	...	4.60	1.40	...	...	...	18.30	...	30.40
2	92	6.00	5648	4.60	.45	19.72	33,888	2.40	32.80	73.10	9.78
3	92	104.50	1045	4.60	2.16	29.17	109,202	37.17	52.62	70.60	48.69
4	92	104.50	1402	4.60	1.76	43.28	146,509	40.63	78.20	51.70	38.26
5	92	104.50	1431	4.60	1.66	45.56	149,539	38.97	77.72	50.10	36.03
6	100	104.50	1092	5.44	2.77	26.80	114,114	49.90	68.00	73.30	50.00

The fan is of the modern Waddle type, with diverging outlet, 21 ft. diameter, 2 ft. 9³/₄ in. wide at the centre ; 10 in. wide at the circumference or outlet, diverging to 2 ft. 4 in. at the periphery, and has a single inlet 9 ft. in diameter. It is driven by a single-cylinder horizontal engine 18 in. in diameter by 18 in. stroke, and working at a steam pressure at the boilers of 80 lbs. per square inch. The fan is direct driven, and the usual working speed is 92 to 95 revolutions per minute. It is connected to the up-cast shaft by a drift 11 ft. wide by 9½ ft. high, and about 9 ft. in length, dipping at a gradient of 1 in 3 toward the shaft. Fig. 606 shows the relative positions of the engine *a*, fan drift *b*, and up-cast shaft *c*.

The test was made with a view of ascertaining : (*a*) the quantity of air the fan was able to circulate in the fan drift ; (*b*) the actual quantity circulating in the workings underground ; (*c*) the amount of difference, or leakage, between the quantity circulating in the fan drift and the quantity circulating underground.

For these purposes we submitted the fan to a series of six different tests as follows :—

(1) The mine entirely shut off from the fan by means of a timber partition in the fan drift. This test was made to obtain the initial depression or water-gauge, and is mainly of value for comparison

with other fans under similar conditions. The initial depression in this case amounted to 1.40 in., as shown in column 6, Table I.

(2) This test was made with a restricted opening, 3 ft. by 2 ft., in the drift partition. This test also is chiefly for comparison, and shows the disadvantage of the fan drawing the ventilating current through a restricted passage, as the manometrical efficiency of the fan under these conditions works out at the low figure of about 10 per cent., as shown in column 12, Table I.

(3) The fan drift in this case was fully open, with the drift and mine in normal working conditions. With this test the total quantity of air measured in the fan drift amounted to 109,202 cub. ft. per minute (see column 8), with a ventilating pressure of 2.16 in. of water-gauge.

(4) In this test the fan drift was fully open, and, in addition, the large doors in air-lock *d* enclosing the up-cast shaft at surface level were kept half opened. This test shows the largely increased quantity of air that the fan could under these conditions circulate, the greater part of the air passing direct from the outside atmosphere to the fan, with consequently little resistance.

(5) Both the fan drift and the large doors in the air-lock were kept fully open during this test, and the whole of the air, 149,539 cub. ft. per minute, would be practically drawn direct from the outside atmosphere, and none or very little from the underground workings.

(6) The fan drift was fully open, the large doors *d* closed in the air-lock, and the mine and fan brought to normal working conditions as in test (3), but with the fan speed increased to 100 revolutions per minute. In this test the quantity of air passing was increased from 109,202 cub. ft. per minute to 114,114 cub. ft. per minute, or approximately an increase of 5000 cub. ft. per minute. In other words, with an increase of about 11 per cent. in the speed of the fan, only $4\frac{1}{2}$ per cent. of increase is obtained in the quantity of air passed. During the time the last four tests were being made the fan drift was divided by cords, vertical and horizontal, to form eighteen divisions, separate readings were taken in each division, and the total quantity ascertained from these. To check these results a number of readings were taken without the dividing cords, the anemometer being slowly moved all over the drift in a sinuous course, and the mean of these readings recorded.

In addition to these tests the air was carefully measured in the underground return airways to ascertain the actual volume circulat-

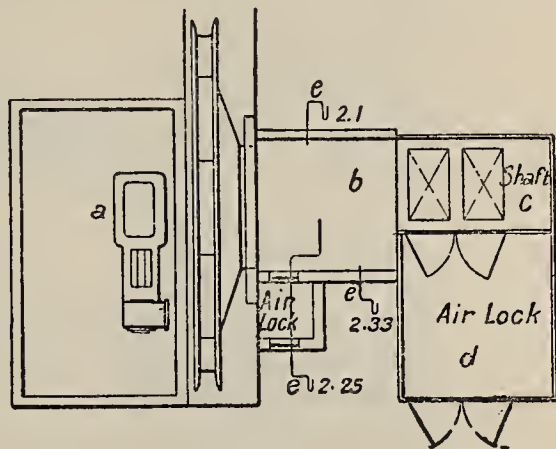


FIG. 606.

ing in the workings. The air was measured and the ventilating pressures obtained by anemometers and water-gauges supplied by Messrs Kelvin & White, Ltd., Glasgow, and guaranteed by them as corrected and tested at Kew. Three anemometers and three water-gauges were used in the tests, and the mean of the readings taken, the water-gauges *e* being placed in openings drilled into the drift, and in the position shown in fig. 606. During the first five tests the fan and engine were driven at a uniform speed of 92 revolutions per minute, which was increased to 100 revolutions during the sixth and last test. While these tests were being made, indicator diagrams of the engine were taken by a competent mechanical engineer.

TABLE II —SHOWING RESULTS OF SECOND TEST OF WADDLE FAN.

Average number fan revolutions during test, ...	92
<i>Fan Drift:</i>	
Area, square feet, ...	104
Temperature of air, ...	53° F.
Average velocity, feet per minute, ...	903
Volume, cubic feet per minute, ...	93,900
Average water-gauge, inches, ...	2.25
<i>No. 1 Pit, Main Return:</i>	
Area, square feet, ...	51.5
Temperature of air, ...	57° F.
Average velocity, feet per minute, ...	840
Volume, cubic feet per minute, ...	43,260
Water-gauge, inches, ...	1½
<i>No. 2 Pit, Main Return:</i>	
Area, square feet, ...	51
Temperature of air, ...	59° F.
Average velocity, feet per minute, ...	420
Volume, cubic feet per minute, ...	21,420
Water-gauge, inch, ...	1½
No. 1 pit volume, ...	43,260
No. 2 pit volume, ...	21,420
Unworked seam, ...	3,000
Total, ...	67,680
Volume in drift, ...	93,900
Volume underground, ...	67,680
Difference due to leakage, ...	26,220 = 28%

Guiding and Conducting the Air-Current.—In no part of colliery work does so much thought and care require to be expended as in the arrangements for guiding and conducting the different underground air-currents. In many collieries large quantities of air can be measured in the intake and return airways near the shafts, but the actual amount that really reaches the working faces—the most important point—is often comparatively small. To ensure the maximum quantity reaching the working faces, the main air-current

will have to be split into as many currents as are necessary for each individual colliery, and each of those currents carefully conducted to the working faces by means of air-crossings, trap-doors, bratticing, screens, etc.

Air-Crossings.—When two currents, an intake and return, have to cross each other, an air-crossing will require to be constructed, so that they may not intermix. Where the strata are hard and free from open fissures the air-crossings may easily be constructed to pass over each other (fig. 607), but where the strata are soft and friable, an air-crossing has often to be built of brickwork and wood, or brickwork and iron. Whatever method is adopted, the crossing ought to be thoroughly air-tight.

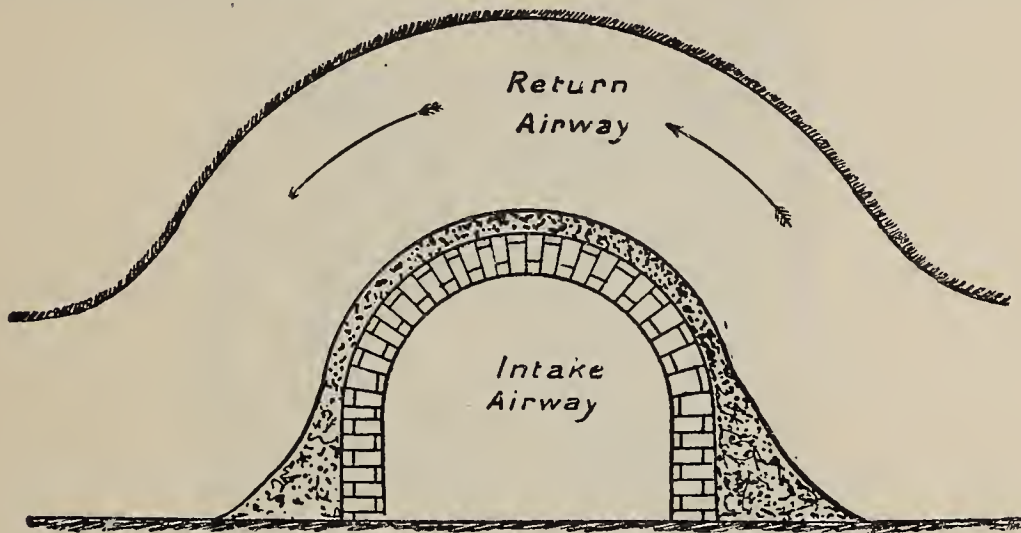


FIG. 607.—Air-crossing.

Trap-Doors.—In all collieries a number of trap-doors are required, either on the main roads or on some of the subsidiary haulage roads. As few doors as possible ought to be used on the main haulage roads, as they are often dangerous in these positions, unless attended by a 'trapper,' and they are also expensive to keep in order and repair, owing to the continual squeeze on the strata, which tends to constantly crush the frames out of position and shape. Fig. 608 shows the method of fixing an ordinary trap-door on underground roads. The door is usually made of wood, 1 to $1\frac{1}{4}$ in. thick, hinged to frames 6 or 7 in. square. An iron bow is fixed in front for the tubs to strike and to facilitate their passage. All doors should be made so that they will close automatically, which can be easily accomplished if they are properly hung with the right amount of 'lean-to' at the top. Sometimes a weight and pulley are attached to trap-doors to assist in closing them, but unless these are carefully fitted up and often examined they are sure to get out of order, and do more harm than good. Fig. 609 shows a very simple and effective method of automatically closing trap-doors.

To the door D is fixed a bar of wood A, 6 or 7 in. square, set at such an angle as to clear the approaching tubs at the end un-

connected to the door. This end works on a roller pulley B, fixed at the side of the road, and clear of the rails. As the tubs approach they first strike the horizontal bar and gradually push the door open, the sides of the tub rubbing against the bar until they pass through. After they have passed, the door closes of itself, the weight of the attached bar of wood helping to draw it back. This apparatus is easily attached to any door, and will work well on haulage roads, especially with slow-speed haulage, without a 'trapper' being required to attend to it.

An automatic door, such as is used for main haulage roads in some districts in Germany, is shown in figs. 610, 611. It is made of wood, covered with felt, and hinged to the brickwork. To facilitate

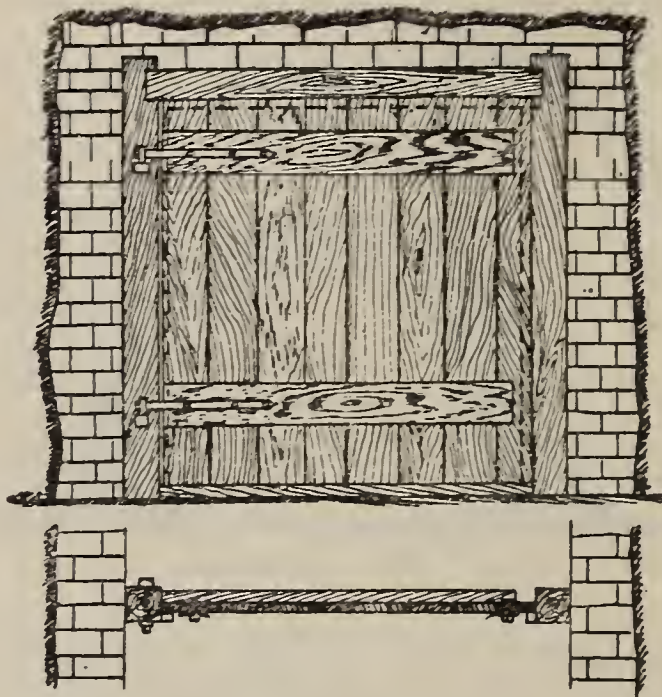


FIG. 608.—Trap-door.

working it is hung slanting. On the inside there is fastened a half-hoop of strap-iron *a*, lower than the top of the tub. Behind the door is fixed a prop, or else a bolt *b*, made of round bar iron and held in a socket fastened to the wall. When the door is opened the bolt is about 4 in. from the edge, and on turning engages with it, and when it closes the bolt hangs down. On the approach of the tubs the door is opened by the pony-driver, and is caught by the bolt *b*. After the horse has passed the door, the first tub presses against the hoop *a*, and so pushes the door against the wall, in consequence of which the bolt, now liberated, returns to its original position. The door is held open by the tubs as they pass, and after the passage of the last tub closes of itself.

An ingenious self-closing door,* which was first exhibited in working order at the Newcastle Mining Exhibition, is shown in figs.

* *Trans. Min. Inst. Scot.*, vol. ix. p. 123.

612, 613. The door is made in two halves, and hung by pulleys which travel on rails, inclined from both sides towards the centre of the road, which allows the two halves to come together by their own weight. About 2 ft. from the bottom of each division, and attached to the edge of each door, is an angle-iron aa , 7 or 8 ft. long, the outer end of this angle-iron being attached to an eye-bolt fastened to a prop cc clear of the rails. As the tubs approach, they first strike the angle-iron, gradually opening the two leaves of the door,

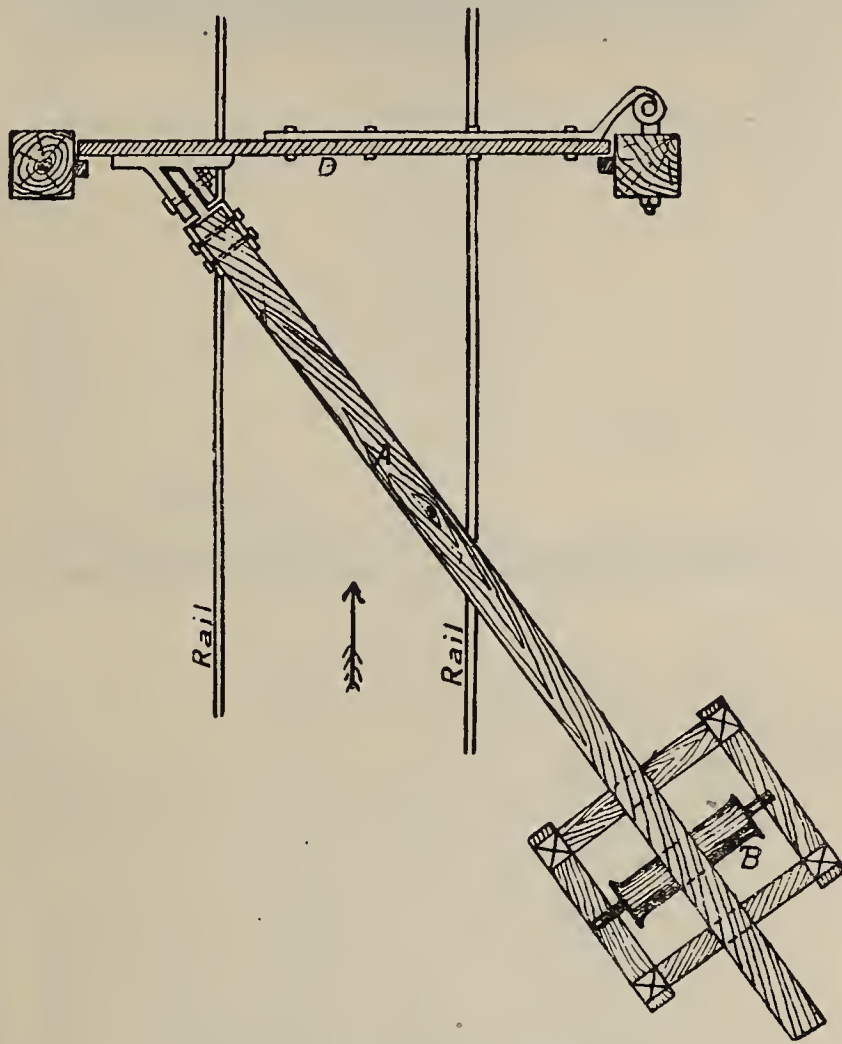


FIG. 609.—Automatic closing door.

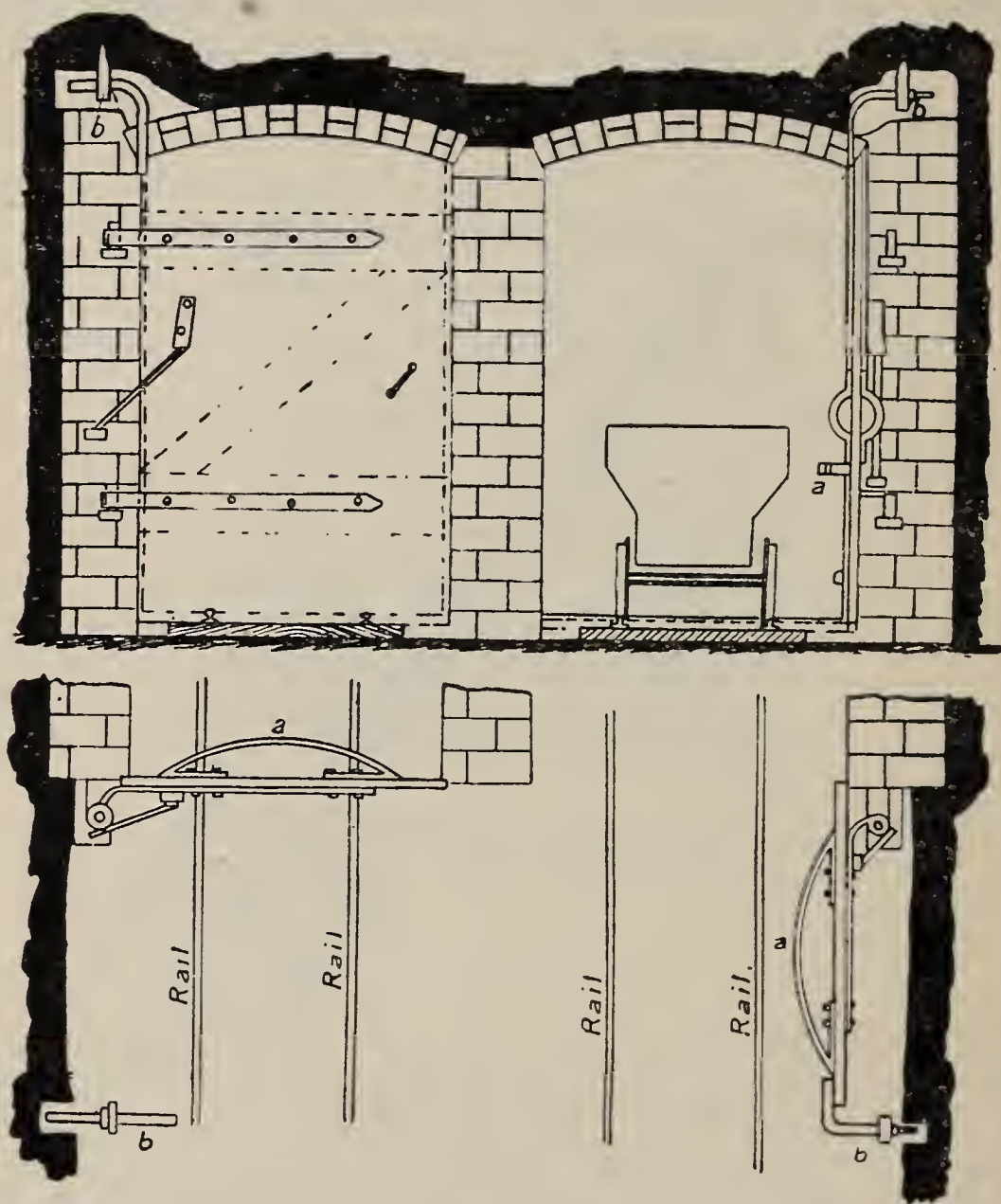
which close again as the tub passes out from between the corresponding bars on the opposite side. The door is opened and shut in a similar manner when a tub passes through it in the opposite direction.

In the ordinary trap-door the tubs often strike very hard, but in this case the doors are opened without shock. On main roads, where the doors serve as the only partitions between the main intake and return airways, there should be at least two doors, each separated by a distance sufficient to accommodate the full train of tubs, and permit the first door to close before the second one opens.

Brattice.—In most underground workings there will be certain

places that require air conducted to them by means of 'bratticing'; this is especially the case in pillar and stall workings, or where close or stone drifts are being driven.

When a close or stone drift has to be driven to any considerable length, and where the road has to be divided to form an intake and return airway, the central division is best made of masonry (fig. 614).

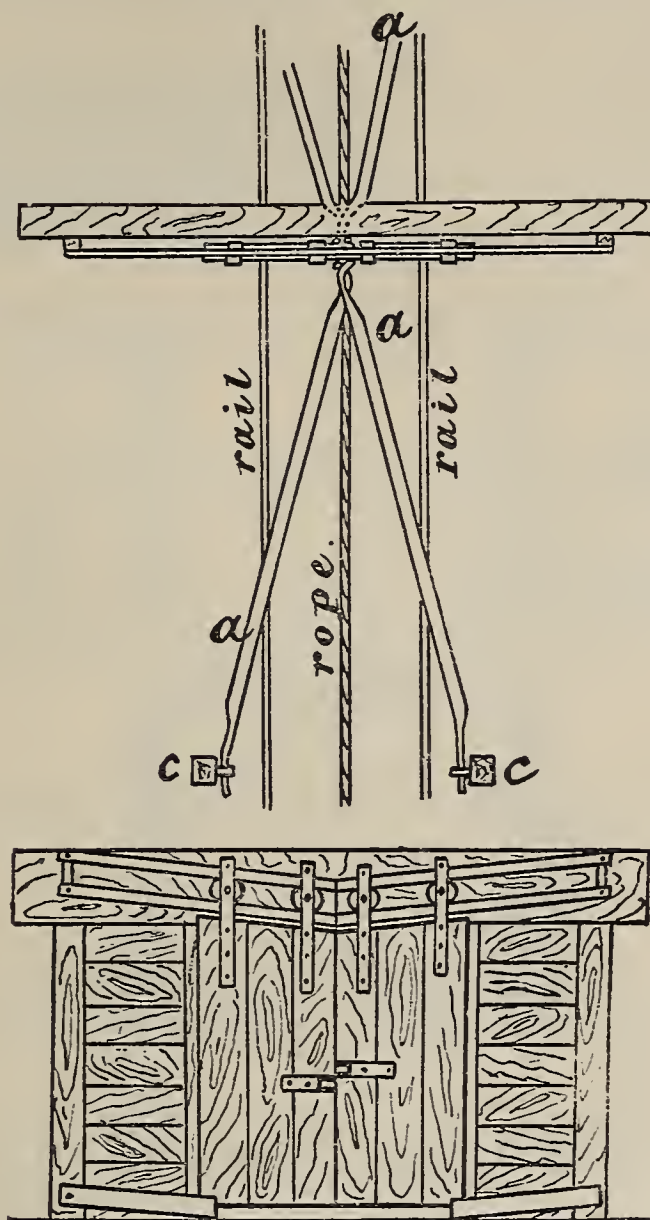


FIGS. 610 and 611.—Automatic doors.

If the roof is good, and no great crush anticipated, a $4\frac{1}{2}$ -in. brick wall may suffice, but it is best to build it $9\frac{1}{2}$ to 14 in. thick. Sometimes only a small brick quadrant is built in roads where traffic is difficult, such as steep inclines or low-roofed ways (fig. 615). A half wall may be built, and a longitudinal plank laid on it, from which boards are laid across to the side of the heading (fig. 616), the seams in the top boards being well filled with good clay, or mortar, and afterwards covered over with brattice cloth.

Wood brattice, or a combination of wood with cloth, is largely used for short distances, such as occur in pillar and stall workings.

To fix the brattice cloth, planks 11 in. $\times$ $\frac{3}{4}$ in. are nailed along the top of the props and at the bottom close to the floor. The cloth is fixed tightly to these planks, and also to the props, by means of short, flat-headed nails. The greatest care must be exercised in fitting the joints of the cloth, and in having such joints opposite a post, as well as in the fixing, top and bottom, for unless this sort of brattice is well



FIGS. 612 and 613.—Self-closing door.

fitted together, it becomes difficult to convey a current of air for more than 30 or 40 yards.

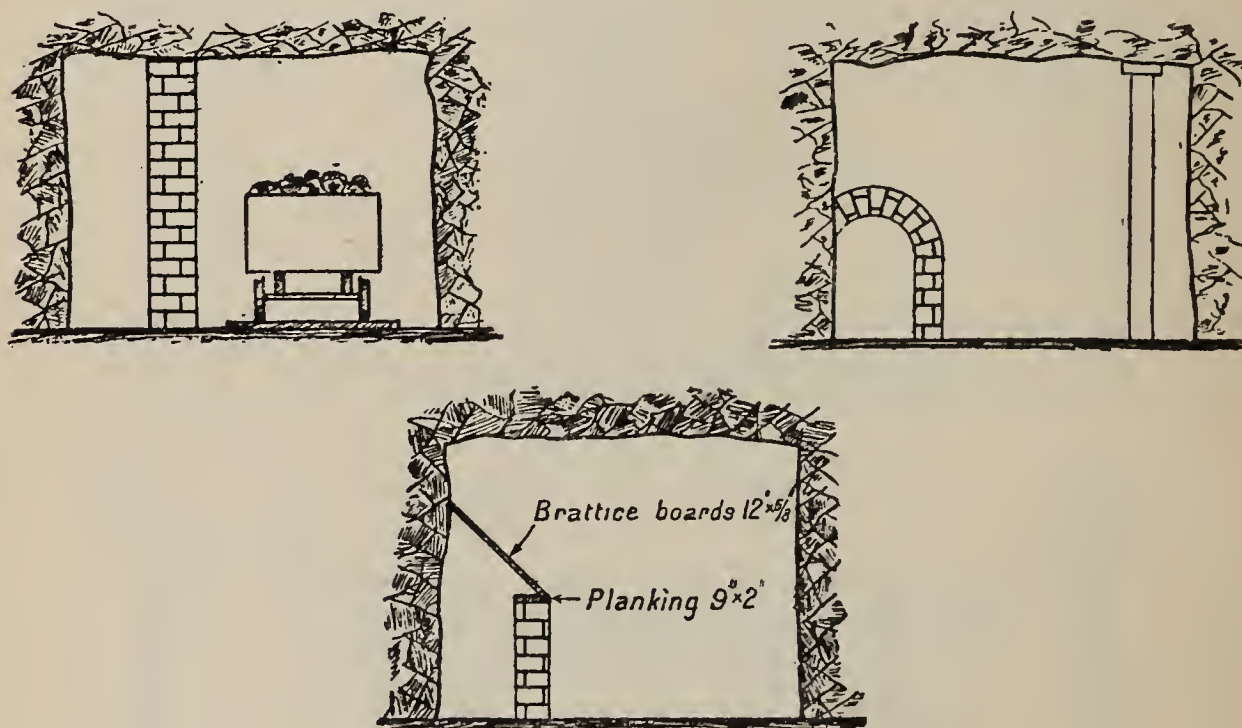
In places where much fire-damp is given off, it is often difficult to keep the face clear by means of brattice at a distance of 20 yards.

In dividing the road by bratticing, the spaces for intake and return are generally of unequal area; and in such cases it is the general rule to make the narrow way the intake, and the wide tram road the return. If the current of air is sluggish, this is undoubtedly the best method, as by making the small area the intake it will increase the velocity, and a supply of air may suffice to keep the face clear,

which if brought in by the wider way would have been quite inadequate for such a purpose. If, however, the air can be supplied in large quantities, it can best be brought through the wider way.

In longwall workings, fire-damp often accumulates at the face of the ripping, and in such a case a 'hurdle' screen ought to be set up to clear it out (figs. 617, 618). A hurdle screen is fitted up by fixing a crown or strap across the road, and leaving a space 18 in. or 2 ft. between it and the roof. Two legs or props are set up to the cross-piece, and the screen-cloth firmly nailed to it.

In all branch roads of a longwall working, screens must be fixed to prevent the air, which is generally required to travel round the



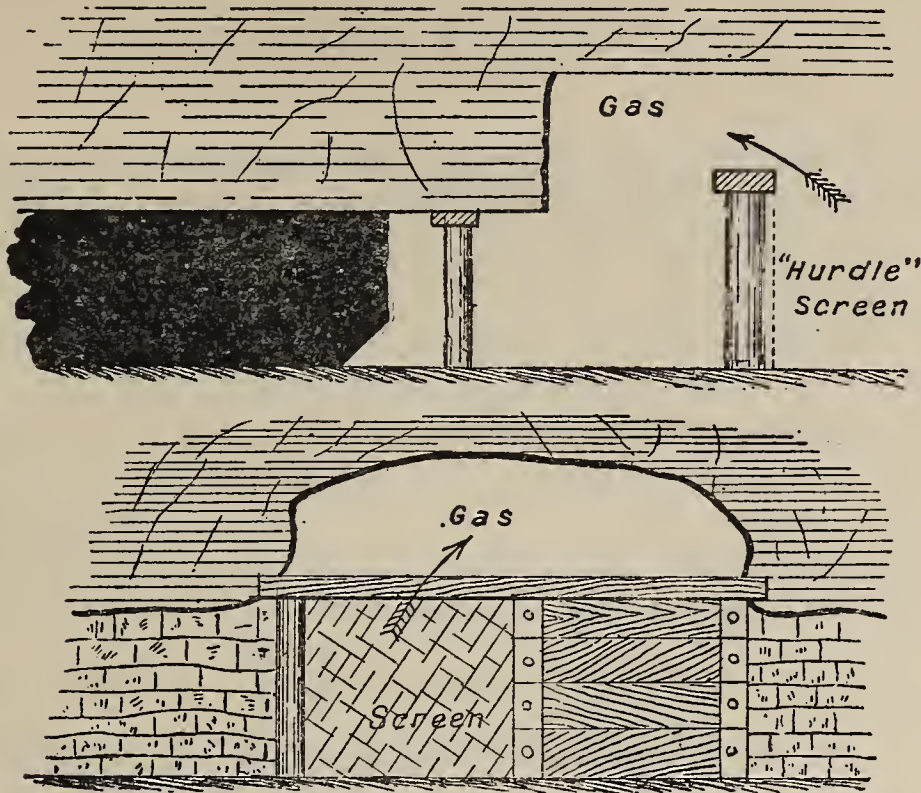
FIGS. 614, 615, and 616.—Ventilating close drifts.

faces, from passing away, unless under exceptional circumstances. Fire-damp frequently accumulates in the open spaces between the packs in longwall, and is often very troublesome. When this occurs, it is cleared out either by conducting a current into such spaces by fixing a screen along the face, or by resorting to 'split buildings.' In this latter method the building or pack wall is not built right along, but at certain intervals a space of 2 to 4 ft. is left, and an airway formed through the waste, so as to clear out any accumulations of gas which may have lodged (fig. 619). Such airways are often difficult to maintain and keep in repair, and new ones will require to be opened at short intervals.

Air Pipes.—Instead of using brattice, wooden boxes or sheet-iron pipes are sometimes used, and are very convenient, but great care must be taken that they are efficient. Where such means are used, the volume of air conveyed to the face may be sufficient to clear away the gas at the point of delivery, but at a short distance back

from the face, where the velocity of air is much reduced, gas may accumulate in dangerous quantities.

The wooden air-boxes are made from 1 ft. to 2 ft. square, or even



FIGS. 617 and 618.

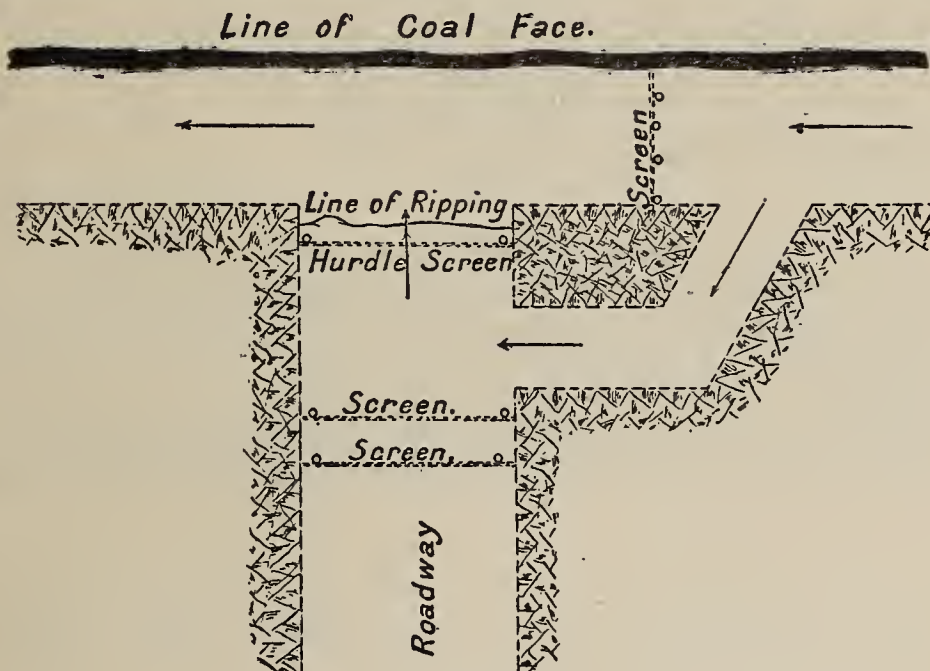


FIG. 619.

larger, and are made of $\frac{3}{4}$ -in. boards fitting closely. Sheet-iron pipes from 1 ft. to 2 ft. diameter are also used.

Laws affecting the Air-Current.—A current of air, either on the surface or underground, is produced, as has been explained, by differences of pressure or density. A very small difference of pressure or force is required to impart motion to the air, but a current travel-

ling in a circumscribed area meets with a very large amount of resistance, which is termed friction, and of the total force or pressure required to set a ventilating current in motion, a very large percentage is taken up in overcoming this friction. Roughly speaking, 90 to 95 per cent. of the actual force is expended in overcoming the resistance met with in the workings, while only 5 to 10 per cent. of the pressure is effective.

If an air current in passing through the underground workings met with no resistance or friction, it would flow with a velocity equal to that attained by a body falling from a given height, and could be expressed by the formula $v = \sqrt{2gH}$, or approximately $= 8\sqrt{H}$, where v = velocity attained in feet per second; H = height in feet fallen through; and g = value of gravity = 32.2. Before air will flow from one point to another, there must be a difference of pressure between the two points, and this difference in pressure can be measured as a head of air or motive column H , which may be substituted for the height, in above formula, through which the falling body has traversed. We have already shown that a head of air or motive column can be expressed in inches of water-gauge $h = \frac{w \times H}{5.2}$, w being = weight of

1 cub. ft. of air at the prevailing temperature and pressure. But the value of $w = \frac{1.3253 \times B}{460 + t}$ and $H = \frac{v^2}{2g}$; therefore the pressure in inches of water-gauge required to set an air current in motion (with

no resistance) will be $h = \frac{\left(\frac{1.3253 \times B}{460 + t}\right) \times \frac{v^2}{2g}}{5.2}$.

The laws of air friction are usually expressed as follows:—

The pressure required to overcome the resistance due to the friction of air, when the velocity is constant, varies in proportion to the surfaces in contact.

The pressure per unit area required to overcome the resistance due to friction varies inversely, as the sectional area of the airway, or

$$P \propto \frac{1}{a}.$$

The pressure required to overcome the friction varies in proportion as the *square of the velocity*.

Assuming that this latter law is correct (it is approximately correct for all except very high velocities), the force required to overcome friction is expressed by the formula, $F = ksv^2$.

Where F = force of friction measured in lbs.

K = coefficient found by experiment.

S = total rubbing surface of the airway in sq. ft. = total length $\times$ the mean perimeter in feet.

v = velocity of flow of air in feet per second.

But $v = \frac{Q}{a}$, Q being = total quantity of air circulating in cub. ft. per second, and a = area of cross section of airway in sq. ft

$$\therefore F = ks \left(\frac{Q^2}{a^2} \right).$$

If we assume that the resistance is overcome by a head of air, or motive column H , then we can also assume that it is overcome by an urging force $F = Haw$, which is the equivalent of this head.

$\therefore Haw = Ks \left(\frac{Q^2}{a^2} \right)$ (w being = weight of 1 cub. ft. of air at given temp. and bar.), from which we have $H = \frac{k}{w} \times \frac{sQ^2}{a^3}$, or in inches of water-gauge $h = \frac{k}{w^2} \times \frac{sQ^2}{5 \cdot 2(a^3)}$.

The factor $\frac{k}{w}$ is determined by direct experiment in mines.

These are the three principal laws enunciated by the late J. J. Atkinson, and two other laws may be stated, which as a natural consequence follow from them, viz. :—

The power required to overcome the resistance to an air current in the workings of a given mine varies as the *cube* of the velocity, and the resistance measured in lbs. per cub. ft. of air current is equal to the product of the coefficient of friction, the total contact surface and the velocity squared and divided by the sectional area of road, *i.e.*

$$P = \frac{KSV^2}{a}.$$

The rubbing surface spoken of is the whole extent of surface exposed to the ventilating current, *i.e.* the roof, pavement, and two sides of the road, the sum of these four sides in section being called the *perimeter*. The total rubbing surface will therefore be the total length of airway multiplied by the perimeter.

Example.—In an airway 8 ft. wide, 6 ft. high, and 200 fathoms long, what would be the total surface in contact?

$$\text{Frictional or rubbing surface} = (8 \times 2) + (6 \times 2) \times 1200 = 33,600 \text{ sq. ft.}$$

Since the pressure increases in direct proportion to the length, it follows that if we double the length of an airway the pressure will also require to be doubled, to maintain the same quantity of air in motion; or, if the length be reduced one-half, the pressure required will be halved.

Another point to be noticed is, that the pressure required to overcome the resistance due to friction depends largely on the size and shape of the airways, *i.e.* whether rectangular, square, or circular. In airways of each of these shapes, and having the same area, viz., 78.54 sq. ft., the perimeter of a circle would be 31.416 ft., of a rectangle, 26.12 $\times$ 3, 58.36 ft., and of a square, 35.2 ft.

From this it is evident that a circular-shaped airway presents the least area of rubbing surface of the three shapes enumerated, and hence would oppose the least resistance to a current of air passing through it; but, on the other hand, it is the least practicable of the three, and one that is very seldom adopted, except in the case of vertical shafts.

Since the pressure required to overcome the friction varies *inversely* as the area, it follows that in two airways, the areas of which are as 1 : 2, the pressure necessary to overcome the friction of an air current in them will be as 2 : 1, *i.e.* it would take a pressure of 1 lb. in the airway whose area is 1 to do the same work as a pressure of $\frac{1}{2}$ lb. in the airway whose area was 2.

In the case of two airways of equal lengths and perimeters, and consequently of equal rubbing surfaces, the velocity being the same in each case, but having unequal sectional areas, all the conditions upon which friction depends will be equal, therefore the amount of work required to overcome friction will be equal in both cases.

We have assumed that the velocities are equal, therefore the *total pressure* must be the same in both airways; but as the airways have different areas, the pressure per sq. ft. must be different, and will be less in the large than in the small airway.

Example.—In two airways of equal lengths, one 10 ft. $\times$ 8 ft. and the other 14 ft. $\times$ 4 ft., the areas are as 80 : 56, while the rubbing surfaces are equal, and therefore the total resistance or pressure must be the same in each. Suppose we have a velocity of 350 ft. per minute in each airway, and that the total energy expended in overcoming resistance equals 120,000 ft. lbs., the total pressure for each airway will equal $\frac{120,000}{350}$, *i.e.* 342.85 lbs.; the pressure per sq. ft. will, however, be different, for in the 10 ft. $\times$ 8 ft. airway it will equal $\frac{342.85}{80}$, or 4.28 lbs. per sq. ft., and in the 14 ft. $\times$ 4 ft. airway $\frac{342.85}{56}$ or 6.12 lbs. per sq. ft., which is in the ratio of 1 to 1.42. From this it will be seen that the smaller airway requires nearly half as much more pressure per sq. ft. than the larger, and that the pressure varies in *inverse* proportion to the sectional area of the airways through which the currents have to pass.

The pressure required to overcome friction is proportional to the square of the velocity. From this it follows that if the velocity be doubled, the pressure required to overcome the resistance due to friction will have to be increased four times; and if we halve the velocity, the pressure required will only be $\frac{1}{4}$.

If we treble the quantity, the velocity must also be trebled, which will increase the original friction nine times.

From the above laws are deduced nearly all the formulæ used in connection with problems in the ventilation of mines, and of which some may be here given.

If p is the pressure per sq. ft., a the area of airway in sq. ft., s the total rubbing

surface of airway, v the velocity of air current in thousands of cub. ft. per minute, and K the coefficient of friction,*

$$\text{then (1) } p\alpha = ksv^2, \quad (2) \quad p = \frac{ksv^2}{\alpha},$$

$$(3) \quad \alpha = \frac{ksv^2}{p}, \quad (4) \quad k = \frac{p\alpha}{sv^2},$$

$$(5) \quad s = \frac{p\alpha}{kv^2}, \quad (6) \quad v^2 = \frac{p\alpha}{ks}, \text{ or } v = \sqrt{\frac{p\alpha}{ks}}$$

$$\text{while the H.P.} = \frac{ksv^3}{33000} \quad (7).$$

Practical considerations in reducing Friction.—*Reducing the Length.*—This can be most conveniently done when the airways are being constructed at first, and if the distance which the air current has to travel can in any way be reduced, even after the airways have been made, this ought to be done.

There are many advantages to be gained by having airways as short as possible, amongst which may be mentioned: The smaller cost and reduced expense for repairs and upkeep, and the larger volume of air which can be obtained by a given expenditure of power. All sharp angles should be avoided as much as possible.

Increasing the Area.—As to increasing the sectional area, there is a practical limit to which this can be done, depending largely on the nature of the strata through which the airways pass; for sometimes it is only with the greatest difficulty that a road can be enlarged, especially if the roof or floor is very bad, and if there is a continuous crush on the workings. For this reason it is often cheaper and easier to make an additional airway parallel to the one already existing.

By doing this the rubbing surface is doubled, and the resistance due to friction correspondingly increased; therefore the pressure will also require to be doubled. At the same time, however, doubling the rubbing surfaces has also doubled the *area*, and consequently the *velocity* will be *halved*, and the resistance and pressure reduced to one-fourth, the decrease of friction, owing to the reduced velocity in the larger area, being always much greater than the increase due to the extra rubbing surface.

Example.—If an airway measuring 9 ft. $\times$ 6 ft. is enlarged to 9 ft. $\times$ 8 ft., the area will be increased from 54 sq. ft. to 72 sq. ft., or in the ratio of 3 : 4, and the velocity will be reduced *inversely* in the same ratio, *i.e.* from 4 : 3, while the rubbing surface will, at the same time, be increased from 30 sq. ft. to 34 sq. ft., or in the ratio of 15 : 17. The resistance, however, varies as the square of the velocity, or as

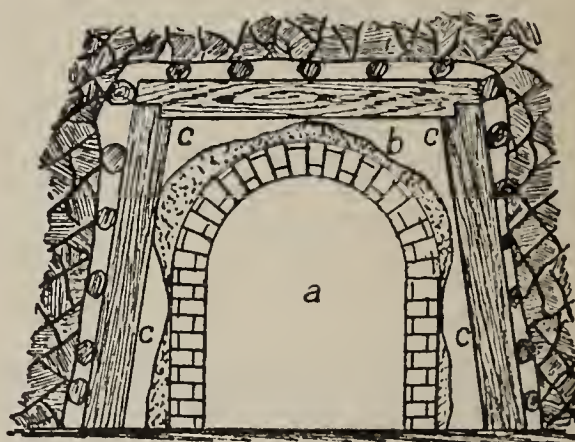
* For every foot of rubbing surface, and for a velocity in the air of 1000 ft. per minute, the friction is equal to 0.26881 ft. of air column of the same density as the flowing air, which is equal to a pressure, with air at 32° F., of 0.0217 lb. per sq. ft. of area of section. This is known as the coefficient of friction.

16:9, therefore the resistance, after increasing the area, will be reduced to $\frac{9}{16}$ of $\frac{17}{15}$ ($=\frac{153}{240}$), the net gain being over one-third.

Another important factor in limiting the area of airways is the size of the shafts, for with small shafts of insufficient size to permit of the passage of the requisite quantity of air, it is useless to increase the area of air-courses beyond a certain limit.

In making airways so as to present the least possible resistance the following points should be attended to. They should be as nearly as possible of the same sectional area throughout, have as few sharp angles as possible, and no side projections and narrow places in them. They should also have a good-sized area, and be as nearly circular, or, if this is impracticable, as square, as possible.

In experiments made recently by D. Murgue on the quantities of air passing in airways of varying construction, he found that there was a great deal of difference in the respective behaviour of arched airways, timbered and untimbered airways (fig. 620). In workings of the same area he found that the quantities passed would be 46,000 cub. ft. per minute for arched airways; 36,000 for airways without timber; and 32,000 for timbered courses, or in the proportion of $1\frac{7}{18}:1\frac{1}{8}:1$. Readings of the water-gauge for the same airways with these quantities were:—Arched airways, 0·253 in., airways without timber, 0·760 in., and airways with timber, 1·275 in., or in the ratio of 1:3:5 (approximately).



a = arched airway; *b* = unlined airway;
c = timbered airway.

FIG. 620.*

These results show in a very striking manner the advantage of having good smooth passages for the air to travel in, and that often better results could be obtained by giving more attention to the airways than by trying experiments with better ventilating machines. For the different airways Murgue gives the following coefficients of friction, expressed as pressure per sq. ft. in decimals of a lb. for each square foot of rubbing surface and an air velocity of 1000 ft. per minute:—

Arched passages, average coefficients	·001716
Unlined ,, ,, ,,	·00494
Timbered ,, ,, ,,	·00821

The laws of ventilating may be summed up and briefly stated as follows:—

* The illustration is intended to show graphically that the arched airway will pass as much air with the same ventilating pressure as either the unlined or timbered airways described on its outside.

The quantity of air circulating in a mine is as the square root of the pressure or the square root of the water-gauge reading.

In airways of the same sectional area, and which vary in length only, the volume and velocity of air currents are *inversely* proportional to the square root of their lengths.

The quantity of air passing in airways of different areas is, other things equal, in proportion to the square root of the area multiplied by the area itself.

The resistance varies directly as the length.

The pressure required to propel air through airways is inversely proportional to the areas, other conditions remaining the same.

The quantity of air circulating is proportional to the cube root of the power applied.

Since the quantity of air circulating varies as the cube root of the power applied, and as the number of revolutions of a fan vary in the same ratio, it follows that the quantity of air circulating at any moment depends *directly* on the speed of the fan.

Other general formulæ, as arranged by Merivale, may here be given :—

- a. Ventilating pressure varies as depth of up-cast shaft (in furnace ventilation).
- b. Ventilating pressure varies as difference of temperature between up-cast and down-cast shafts.
- c. Ventilating pressure varies as horse-power of fan or furnace.
- d. Ventilating pressure varies as quantity of coal burned.
- e. Quantity of air circulating varies as revolutions of fan.
- f. Quantity of air circulating varies as square root of pressure.
- g. Quantity of air circulating varies as square root of depth of up-cast (in furnace ventilation).
- h. Quantity of air circulating varies as difference of temperature between up-cast and down-cast shafts.
- i. Quantity of air circulating varies as cube root of horse-power.
- j. Quantity of air circulating varies as cube root of coals burned.

'Splitting' the Air.—The benefits to be derived by increasing the area of airways, and also by adding another airway, have been already pointed out. The latter method of solving this problem is termed *'splitting the air.'* In former times (and even in some collieries at the present day) it was the practice to carry all the air round the workings in one current, which is a very bad method, and one that should be avoided if possible. The air should be taken round the workings in a number of separate currents, as only by this system can the greatest efficiency be obtained.

Splitting the air, as already shown, increases the rubbing surface, but it reduces the velocity with the same pressure, and as the pressure varies as the rubbing surface and the velocity squared, a greater quantity of air can be propelled by a given power.

There are other advantages gained by splitting the air, and ventilating each section separately with main splits, among which are the following :—

A greater total quantity of air is obtained.

By splitting the air each section gets a more uniform and purer supply.

A fall occurring in one split or section does not injure the others.

An explosion taking place in one section is less liable to occasion disastrous effects in the other sections.

Fewer trap-doors are required on the main roads.

To obtain the greatest possible advantage from 'splitting,' the main splits ought to commence as near the down-cast and end as near the up-cast shafts as possible.

In 'splitting,' the main current of air should never be split to such an extent that its velocity becomes insufficient to keep the working faces clear of gas. If this be the case, fire-damp or choke-damp may accumulate, and render the working-places dangerous.

The velocity of the air travelling along the faces in fiery mines where unshielded lamps are used should never be less than 120 to 150 ft. per minute, nor more than 350 ft. per minute.

In ordinary circumstances, the velocity along the face should be from 2 to $3\frac{1}{2}$ ft. per second, in splits $3\frac{1}{2}$ to 5 ft. per second, in main airways 5 to 10 ft. per second, and in shafts it often reaches 20 to 40 ft. per second..

The relative proportion of air to be distributed among any given number of splits will depend greatly on the amount of natural gases given off, apart from the actual quantity required for men and animals, the burning of lamps and the blasting of explosives, so that in distributing a current it is best to divide it according to local circumstances; *i.e.* the section giving off the most gas should get the largest supply of air. The average quantity allowed by some authorities for mines free from fire-damp is 150 to 200 cub. ft. per minute per man and boy, and three to six times these quantities for each horse employed underground. In fiery collieries 200 to 350 cub. ft. per minute per man and boy ought to be allowed, and an additional quantity for 'scale,' as a certain proportion never reaches the working faces, being lost by leakage through defective stoppings, bratticing, and screens. The above quantities are only approximate.

For a fiery mine employing 200 men and 20 horses, the quantity required would be:—

$$\begin{array}{l} \text{men } 200 \times 350 = 70,000 \text{ cub. ft.} \\ \text{horses } 20 \times 2100 = 42,000 \quad , , \quad = 112,000 \text{ cub. ft. per minute.} \end{array}$$

Equal Splits. — The expression 'two equal splits' means that the original single air-current has been divided into two currents, and each current traversing an airway of *half* the original length, but both with the *same area* as the original one. 'Equal splitting' is more of a theoretical than a practical expression, for to split an air current into two or more equal volumes can rarely be carried out in practice, although in some cases it may approximately be done.

When the original single current is divided into two equal splits, they may be considered as one current with *double* the area, but with the *same* rubbing surface; likewise, when divided into three equal splits, they may be considered as one with

three times the original area but with the same rubbing surface. This may be illustrated as follows:—

	Area.	Rubbing Surface.
Original airway, . . .	50 sq. ft.	160,000 sq. ft.
Two equal splits, . . .	{ 50 „	80,000 „
	{ 50 „	80,000 „
	<u>100 „</u>	<u>160,000 „</u>
Three equal splits, . . .	{ 50 „	53,333·3 „
	{ 50 „	53,333·3 „
	{ 50 „	53,333·3 „
	<u>150 „</u>	<u>160,000 „</u>

Example.—If the total quantity of air passing round the workings of a mine is 20,000 cub. ft. per minute when the size of the airway is 8 ft. × 5 ft. and 2500 ft. long; what quantity will circulate if the current is divided into 2, 3, and 4 equal splits?

First find the ventilating pressure required to circulate the original quantity through the given airway, by the formula

$$p = \frac{ksv^2}{a} = \frac{0.0217 \times (26 \times 2500 \times \left(\frac{20,000}{1000 \times 40}\right)^2}{40} = 8.8 \text{ lbs. per sq. ft.}$$

To find quantity with new area, $v = \sqrt{\frac{pa}{ks}} \times a_2$, but p , k and s are the same in each instance, therefore the quantity passing in two equal splits will be as $\sqrt{a_1} \times a_1 : \sqrt{a_2} \times a_2 :: Q : x$

$$\begin{aligned} \therefore x &= 20,000 \frac{\sqrt{80} \times 80}{\sqrt{40} \times 40} \\ &= 56,582 \text{ cub. ft. per minute.} \end{aligned}$$

The same result approximately may be arrived at by simply taking the number n_1 of splits, thus $\sqrt{1} \times 1 : \sqrt{2} \times 2 :: 20,000 : x$

$$\begin{aligned} \text{and} \quad x &= 20,000 \frac{2\sqrt{2}}{1\sqrt{1}} \\ &= 20,000 \times 1.4142 \\ &= 56,568 \text{ cub. ft.} \end{aligned}$$

$$\text{For three equal splits the quantity } x = 20,000 \frac{3\sqrt{3}}{1\sqrt{1}} = 103,920 \text{ cub. ft.}$$

$$\text{and for four equal splits the quantity } x = 20,000 \frac{4\sqrt{4}}{1\sqrt{1}} = 160,000 \text{ cub. ft.}$$

Unequal Splitting.—This expression is used when an air current is split up into a number of separate currents each of which traverses an airway of different dimensions and therefore meets with a varying amount of resistance.

If there are a number of splits of *unequal* area, subject to a common pressure, the quantities of air that will pass in each are in proportion to

$$\frac{\frac{1}{a}}{\left(\frac{1}{a}\right)^2 \times S} \text{ or } R = \sqrt{\frac{a^3}{S}}.$$

where

R = relative quantity of air passing into each airway.

s = total rubbing surface in sq. ft. of each airway.

a = area of each airway in sq. ft.

Example.—An airway 7 ft. $\times$ 9 ft. $\times$ 1200 ft., through which 80,000 cub. ft. per minute is passing, is divided into three unequal splits of the following dimensions: first split, 6 ft. $\times$ 7 ft. $\times$ 1200 ft.; second split, 6 ft. $\times$ 6 ft. $\times$ 900 ft.; third split, 6 ft. $\times$ 4 ft. $\times$ 840 ft. With the same total volume, what quantity will pass in each airway?

Let R_1 , R_2 , and R_3 denote the relative quantities going into each airway, then by the formula $R = \sqrt{\frac{a_2}{s}}$,

$$R_1 = \sqrt{\frac{(6 \times 7)^3}{(26 \times 1200)}} = \sqrt{\frac{74,088}{31,200}} = 1.54.$$

$$R_2 = \sqrt{\frac{(6 \times 6)^3}{(24 \times 900)}} = \sqrt{\frac{46,656}{21,600}} = 1.47.$$

$$R_3 = \sqrt{\frac{(6 \times 4)^3}{(20 \times 840)}} = \sqrt{\frac{13,824}{16,800}} = 0.90.$$

These relative quantities show that for every 1.54 cub. ft. going into the first split, 1.47 cub. ft. will be going into the second, and 0.90 cub. ft. going into the third split.

Let the total relative quantity be denoted by R_4 , then

$$\begin{aligned} R_4 &= R_1 + R_2 + R_3 \\ &= 1.54 + 1.47 + 0.90 \\ &= 3.91 \end{aligned}$$

The total volume x_1 going into the first split will be as

$$\begin{aligned} R_4 : R_1 &:: Q : x_1 \\ 3.91 : 1.54 &:: 80,000 : x_1 \end{aligned}$$

$$\therefore x_1 = \frac{80,000 \times 1.54}{3.91} = 31,508 \text{ cub. ft. per minute.}$$

For second split

$$3.91 : 1.47 :: 80,000 : x_2$$

$$\therefore x_2 = \frac{80,000 \times 1.47}{3.91} = 30,076.72 \text{ cub. ft. per minute.}$$

For third split

$$3.91 : 0.90 :: 80,000 : x_3$$

$$\therefore x_3 = \frac{80,000 \times 0.90}{3.91} = 18,416 \text{ cub. ft. per minute.}$$

$$31,508 + 30,076 + 18,416 = 80,000 \text{ cub. ft.}$$

The relative pressure or power required to pass the same quantity of air through airways of varying area and length may be found by the formula

$$S \left(\frac{1}{a} \right)^2 \text{ or } S \left(\frac{1}{a} \right) \text{ where}$$

A = area of roadways in sq. ft.
 a = „ „

S = Total rubbing surface in sq. ft.
 R = Relative quantity.

Instruments used at Collieries.—*Thermometer*.—A thermometer consists of a closed glass tube, with a capillary bore in the upper part, and a bulb below containing mercury or spirits of wine, and



FIG. 621.—Thermometers.

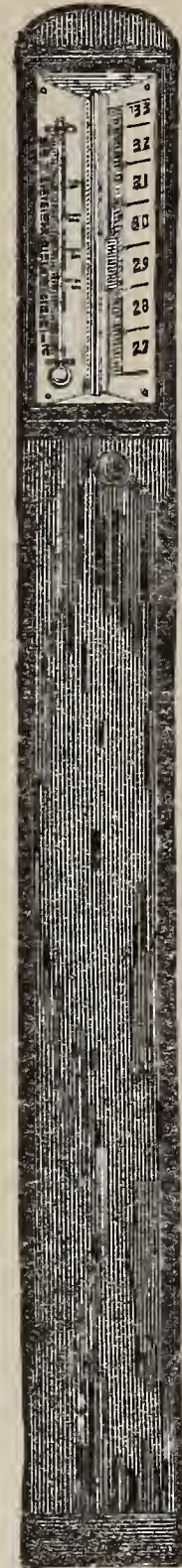


FIG. 622.—Barometer

provided with a graduated scale having two fixed points, viz., freezing point and boiling point. There are three different kinds of thermometric scale adopted for recording the temperature, viz., the Fahrenheit, Centigrade, and Réaumur.

Réaumur divided the space between the freezing and boiling points

into 80 equal parts, but there is no good reason why he should have preferred 80 over any other whole number. This scale is much used in Italy, Russia, and some parts of Germany. Celsius, a Swede, divided his thermometer scale into 100 equal parts between the boiling and freezing points: his scale is known as the Centigrade, and it is used in France and other countries of Europe, and is also being largely used in Britain among scientific men.

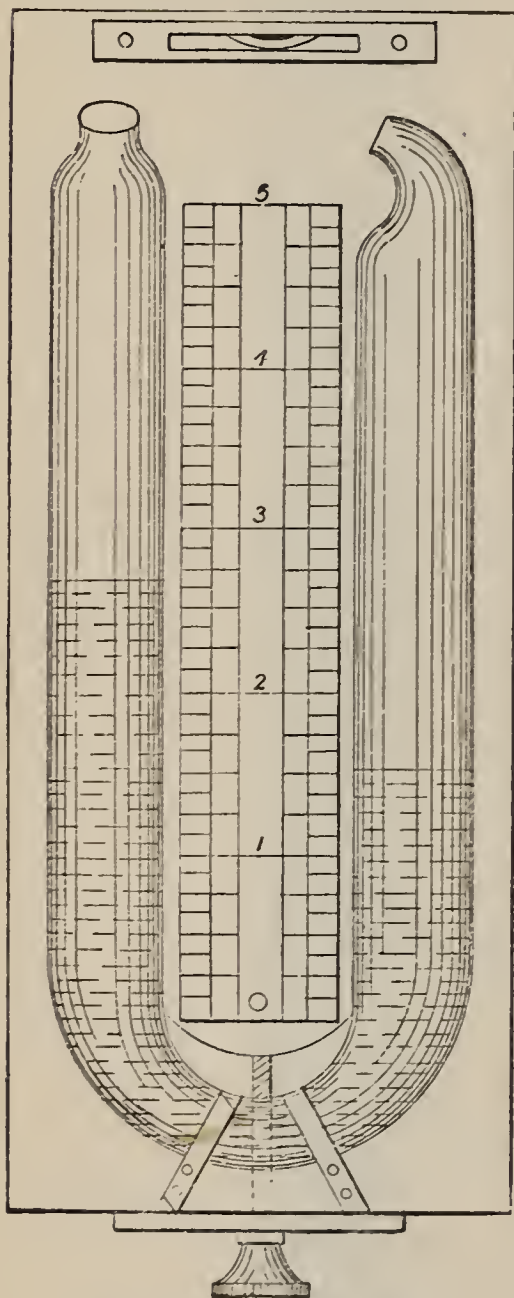


FIG. 623.—Water-gauge.

In Fahrenheit's scale the space between freezing and boiling points is divided into 180 equal parts, and he fixed his freezing point 32° above zero. This scale is the one most commonly used in Britain.

By the use of this instrument we are able to measure the temperature of the air in the workings, or note the difference in temperature between the air in the up-cast and down-cast shafts.

The Barometer is an instrument used for measuring the pressure of air. Its construction is as follows: A glass tube 36 in. long, closed at one end, is filled with mercury and reversed, the open end being temporarily closed until it is placed so that it shall project below the surface of a bath of mercury contained in a reservoir. The barometer has a scale fixed to it, and also a sliding vernier, by means of which it may be read to the one-hundredth part of an inch. Each inch on the scale is divided into tenths, and the divisions on the vernier are one less than the number on a corresponding length on the barometer scale.

A barometer is of great use at collieries, as it shows the changes in atmospheric pressure. With a low

barometer the gas issues more freely from the coal, and if there is any gas lying in the waste it will soon find its way into the working places, owing to the reduced pressure. It is important, therefore, to note the rise or fall of the mercury daily in connection with fiery collieries.

By the Coal Mines Act every mine must be provided with a barometer and a thermometer. The reading of the former should be noted at least once a day, especially by the firemen before going down to inspect the workings in the morning.

Water-Gauge.—A current of air exerts a certain pressure during its passage from one point to another, which pressure is usually measured by a water-gauge (fig. 623).

The construction of this instrument is very simple. It consists of a glass U-tube, of $\frac{1}{2}$ in. to 1 in. in diameter, to which a sliding scale divided into inches and tenths is attached. One of the ends is fitted with a tube at a right angle with the limb, which can pass *into* the intake through a door or stopping, while the other end is connected with the return current. Messrs John Davis & Son of Derby have an improved type of water-gauge, which is largely used. The tube, instead of being the usual U-shape, is composed of two separate pieces of straight boiler gauge glasses, sealed into a brass block; thus they can be readily repaired. In the lower brass block a tap is inserted with a small hole (see fig. 624), thus contracting the connection between the tubes and minimising the oscillation of the water column. The tap may be turned off, thus registering the height of the water column, and the water-gauge brought to a good light to read. The gauge is easily filled and refilled when required. If there is any difference in the pressure between the two currents the water level in the two limbs of the gauge will vary. As 1 cub. ft. of water weighs 62·5 lbs., a cub. in. will weigh ·036 lb. In a tube of 1 in. sectional area, a difference of level of 1 in. will represent a wind pressure equal to the weight of 1 cub. in., and, *pari passu*, to ·036 lb. per sq. in., or $\cdot 036 \times 144 = 5\cdot 2$ lbs. (approximately) per sq. ft.

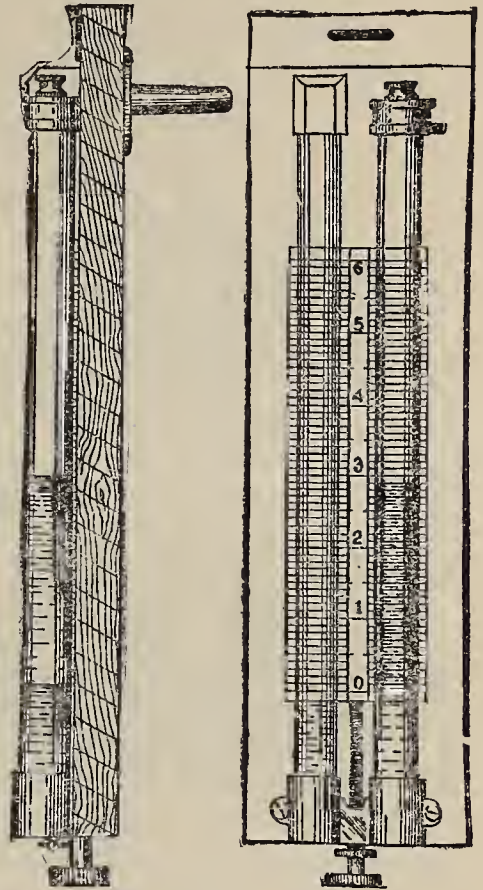
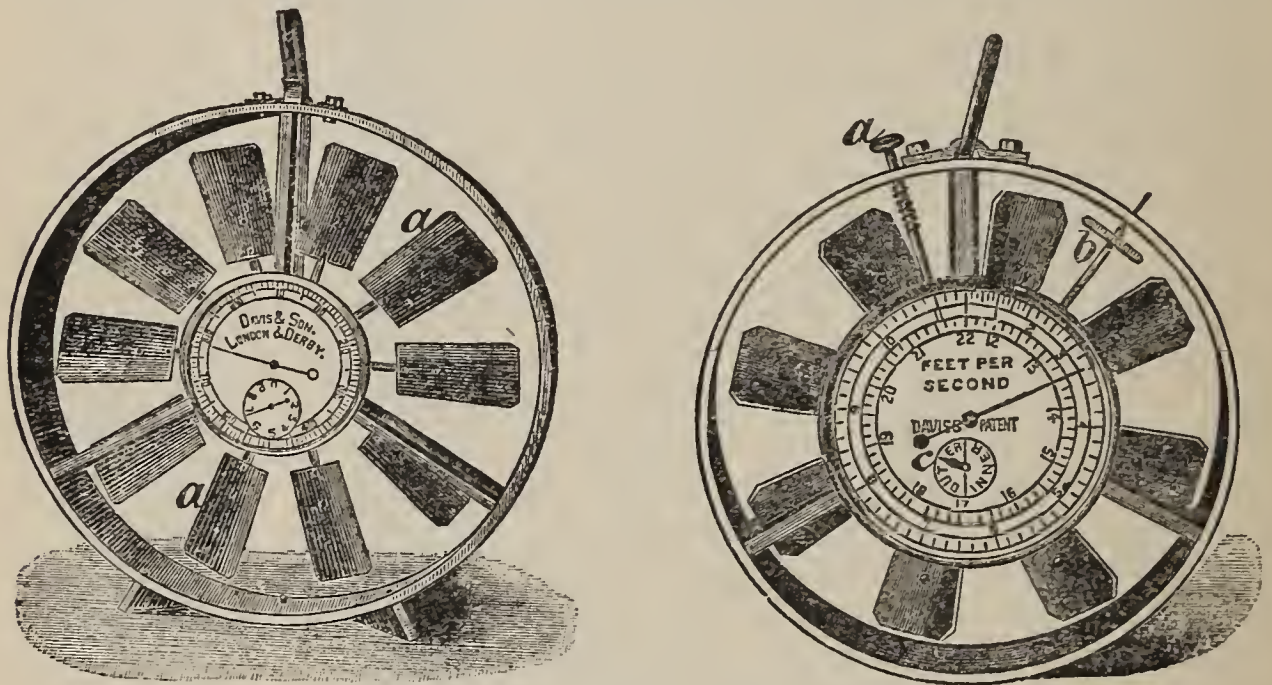


FIG. 624.—Improved water-gauge.

The water-gauge thus acts as a check (though not altogether a very reliable one) on the state of the underground airways. If the latter remain in the same condition for some time, and the ventilating power is neither increased nor decreased, then under ordinary circumstances the height of the water-gauge should not vary. If the difference in level is very much greater than usual, it will probably be due to the fact that the air is not taking its usual course, owing to some doors being open in the main airways, by which the air will be going direct from the down-cast to the up-cast without ventilating the workings. If the water-gauge is to be of any use at all, it ought to be placed as far back from the fan as possible, otherwise it will be impossible to get correct results, owing to the abnormal condition of the air close to the fan.

Anemometers.—To determine the velocity of the air passing in underground workings, it is the common practice to use the instrument called the anemometer. It is constructed somewhat like a small fan, with a number of blades or vanes fixed obliquely to the axis (see figs. 625, 626); when these vanes are rotated a clockwork gearing connected to the axis, which actuates the pointers on the dial of the instrument, records, by means of a scale, the velocity in feet at which the air is travelling. In determining the quantity of air passing, the pointers are first brought to zero, and the anemometer is held out at arm's length in the air passage for a measured period of time, and a reading is taken, the hundreds being read off the small pointer, and the odd feet read off the large one. The sectional area of the passage is carefully calculated, and the total



FIGS. 625 and 626.—Anemometer.

quantity of air passing can be ascertained by multiplying the velocity per minute by the area in sq. ft.

Coal-Dust and Methods of dealing with it.—That coal-dust is an important element in connection with explosions in underground workings seems to be fully recognised by all or nearly all connected with mining operations. It is many years since this fact was first pointed out and experiments made by two eminent authorities, Faraday and Lyell. It is over fifty years ago since these distinguished men first gave their opinion on this much-debated question, but it is only within recent years that much attention has been given to the subject. To Mr William Galloway and the late Sir Frederick Abel we are greatly indebted for the information we possess on the subject. These two authorities studied the question very closely for years, and after many careful and comprehensive experiments, came to the conclusion that “coal-dust is highly dangerous under certain conditions.” Others have entered the field of investigation,

and the results have placed beyond doubt that coal-dust, rather than fire-damp, plays the most important part in many colliery explosions.

There are two theories held by mining authorities regarding the action of coal-dust in colliery explosions, one being that coal-dust alone will cause an explosion without any fire-damp being present in the air, and the other, that before coal-dust becomes really dangerous a certain percentage of fire-damp must be present in the air.

We may here quote briefly the opinion of different authorities who were commissioned to make inquiry on the subject. MM. Mallard and Le Chatelier of the French Fire-damp Commission of 1882 rejected the theory that coal-dust alone would cause any serious danger, or that any colliery explosion of importance could be attributed, with any probability of authenticity, to the action of coal-dust.

The Prussian Fire-damp Commission in 1887 came to the conclusion that the presence of coal-dust in the complete absence of fire-damp gave rise generally to an elongation or propagation of the flame projected by a blown-out shot of limited extent, however far the deposits of dust may extend in the mine roads, but that there were certain descriptions of coal-dust which, if ignited by a blown-out shot, would not only continue to carry on the flame, even to distances much beyond the confines of the dust deposits, but would also, in the entire absence of fire-damp, give rise to explosive results which, in character and effects, were similar to those produced with some other dusts in air containing 7 per cent. of fire-damp.*

The Austrian Fire-damp Commission in 1891, after making a large number of experiments with different coal-dusts, found that, in the absence of fire-damp, nearly all kinds of coal-dust could be ignited by a $3\frac{1}{2}$ -oz. dynamite cartridge exploded in an unconfined space, while many dusts which were notoriously regarded as dangerous proved less inflammable than others which had been regarded as comparatively innocent, the fineness of the dust greatly increasing the liability to ignition.

The Royal Commission on Accidents in Mines, in their report of 1886, were satisfied that "a blown-out shot in working places *where highly inflammable coal-dust exists in great abundance*, may, *even in the entire absence of fire-damp*, possibly give rise to a violent explosion, or may, at any rate, be followed by the propagation of flame through very considerable areas, and even by the communication of flame to distant parts of the workings where explosive gas-mixtures or dust deposits in association with non-explosive gas-mixtures exist."

The Royal Commission on Explosions from coal-dust in mines issued a further report in 1894, in which they came to the following conclusions:—

- (1) The danger of explosion in a mine in which gas exists, even in very small quantities, is greatly increased by the presence of coal-dust.
- (2) A gas explosion in a fiery mine may be intensified and carried on indefinitely by coal-dust raised by the explosion itself.

* *Final Report of the English Commission on Accidents in Mines, 1886.*

- (3) Coal-dust alone, without the presence of any gas at all, may cause a dangerous explosion if ignited by a blown-out shot or other violent inflammation. To produce such a result, however, the conditions must be exceptional, and are only likely to occur on rare occasions.
- (4) Different dusts are inflammable, and consequently dangerous, in varying degrees, but it cannot be said with absolute certainty that any dust is entirely free from risk.
- (5) There appears to be no probability that a dangerous explosion of coal-dust alone could ever be produced in a mine by a naked light or ordinary flame.

Since these reports were issued, conclusive evidence has been forthcoming that an explosion may take place through the agency of coal-dust alone, as such an explosion, attended with loss of life, actually took place at Camerton Colliery in Somersetshire, where fire-damp was not known to have been present prior to the explosion. In this instance it was clearly proved that the explosion was caused by a blown-out shot fired by two men who were repairing on the main haulage road, which was dusty.

At the West Riding Colliery in Yorkshire an explosion took place which was proved, after careful inquiry, to have been caused by coal-dust alone, ignited by a blown-out shot.

Within the last five years extensive experiments have been carried out in Britain, America, France, and Austria, at specially constructed experimental stations. Elaborate and detailed reports have been issued by the governments of these countries, all of which go to prove conclusively that coal-dust alone is highly explosive and will cause disastrous explosions in mines in the entire absence of fire-damp. The British Government have now established an experimental station at Eskmeals in Cumberland, where further tests are being carried out with different kinds and qualities of coal-dust, and it will be some time yet before these are concluded and a final report issued. In the Report on the French Coal-dust Experiments, M. Taffanel, Director of the Experimental Station, in his general conclusions sums up lucidly the dangers arising from the presence of coal-dust in mines and the necessary precautions to be taken to avoid the danger of explosions. We may here give a brief summary of these conclusions. He states that "in considering what—in view of the experiments and the actual state of our knowledge—are the best means to employ in combating the dangers of coal-dust, the first thing is to recognise that prevention is better than cure. An ignition of coal-dust, even if only of limited character, is always a potential cause of numerous fatalities, either directly by asphyxiation or poisoning; whilst, on the other hand, there is also the risk of expanding into a general explosion that will be very difficult to arrest. Even though the methods described in the experiments foreshadow the probability of success, the serious nature of the catastrophes to be feared is such as to necessitate the superposition of precautionary methods, in order to reduce to a minimum the probability of the occurrence of an explosion.

In fact, the probability of an accident should be made the more remote in proportion to the presumptive gravity of such accident.

“Hence, among the most essential precautions against the dangers of coal-dust are those relating to the cause of ignition. These are, chiefly, precautions as to the danger of fire-damp—the ignition of which is one of the most dreaded causes of the ignition of coal-dust—and, secondly, precautions in connection with the use of explosives, in order to prevent the coal-dust from igniting by shot-firing. Consequently, the use of explosives in dusty mines should be restricted to such as are safe in the presence of dust—and also of fire-damp, if the latter be present. The shots must in all cases be well stemmed, so that they cannot blow out. There is no need to insist further on these points, which are well known, except to reserve the question of what constitutes a safety explosive [a very important and necessary reservation]—a point which is the subject of special investigation.

“A less important, but by no means negligible, risk is that of ignition by a flame or an electric arc, where a cloud of dust is accidentally formed in the vicinity of a coal fire or a tippler, or rising from the shunting of pit tubs. Here, however, the clouds must be far denser and larger in order to become a source of danger, than in those formed under ordinary conditions. Nevertheless, it is desirable to suppress the use of naked lights and arc lamps in places where clouds of coal-dust are liable to be formed. It is possible—though this has not been verified—that certain types of lamps with protected flame may produce ignition.” After giving a résumé of the preventive measures to be adopted, M. Taffanel states that “in the general application of preventive measures it is well to bear in mind that, on the one hand, attentive examination of each particular case alone will enable the more or less dangerous character of the workings to be recognised. The experiments afford important bases for this appreciation. Moreover, other conditions being equal, dry coal-dust will be the more dangerous in proportion as it is finer in grain. In galleries or working places where the dimensions of the finest particles reach 1 to 2 millimetres ($\frac{1}{25}$ th to $\frac{1}{12}$ th of an inch), no coal-dust explosion can be produced by the more ordinary initiating causes, though an explosion could be propagated there as the result of a violent initial explosion occurring in a more favourable environment as regards dust, or started by fire-damp or by the detonation of a store of explosives. Other conditions being equal, coal-dust rich in volatile matters is more dangerous than if the content of the latter be small. No explosion can be produced by the ordinary initial causes in a dust deposit where the content of volatile matters does not exceed 18 per cent., though it may be propagated there as the result of a violent initial explosion occurring over a portion of dust deposit richer in volatile matters, or started by fire-damp or by the detonation of a store of explosives. Coal-dust explosions developing, as the sequel of a powerful initial explosion, in a dust deposit

unfavourable to propagation, are naturally less violent and more easy to arrest than those propagated in a favourable deposit.

“The conclusions enunciated above will suffice to show all that can be deduced at present from the experiments already made in view of attaining greater safety in the working of mines.

“The most important questions have been elucidated, and we now know under what conditions a coal-dust explosion can be produced and propagated; we can appreciate the relatively dangerous character of coal-dust deposits; and we know how to prevent the production and arrest the progress of a coal-dust explosion.

“There are still a number of points of detail to be settled, including the investigation of special conditions, which had to be left out of consideration in order to attain the principal object more speedily; and it will also be necessary to ascertain whether the means of arrestation that succeed in a limited gallery of simple form have any prospect of being successful in more complicated or extensive cases—all of which points will be considered in further experiments at Liévin.”

Before an explosion can take place, certain conditions, which are fortunately of rare occurrence in mines, must be fulfilled.* There must be sufficient surplus power in the shot to stir up the dust and to produce sufficient compression of the air. The dust must be very fine and dry; the flame from the explosive must be one of very high temperature and considerable volume. Possibly the shape of the gallery, the relative temperature of the air, and the height of the barometer, may also conduce to modify or intensify the effects. From a consideration of the composition of coal-dust, it is evident that it is a very inflammable substance. Analyses of coal-dust from Ryhope Colliery showed on an average 21·17 per cent. of combustible gas. Dust from Brancepeth Colliery,† however, only gave 0·76 per cent. of combustible gas, and this latter dust was, notwithstanding the small percentage of combustible gas with which it was associated, very sensitive to ignition.

Coal-dust is prevalent to a greater or less extent in a large proportion of collieries, particularly in deep and dry ones. It is more frequent in deep mines, because the temperature of the air is higher in such mines than in shallow ones, and the air being warmer, the moisture in the atmosphere, as it enters the workings, is absorbed very rapidly, causing the air to become dry and favouring the formation of dust. Some seams, again, produce more dust than others, and the more friable the coal the more dust will accumulate.

Formation of Dust.—Coal-dust is most plentiful on main haulage roads, and is found settled on the roof, floor, sides, and timbers. The dust is formed, and accumulates in a variety of ways. Thus the ‘backs’ or cleavages of the coal are usually coated with a fine,

* *Trans. Fed. Inst. Min. Eng.*, vol. vii. p. 60.

† *Ibid.*, vol. vii. pp. 33–38.

friable, black substance, which becomes dust when the coal is broken up by the miner at the face, while the hewing and filling of the coal also contribute largely to the formation of dust. The breaking-off of pieces of coal from the sides of pillars in pillar and stall working, and the falling coal from tubs during haulage, and the subsequent pulverisation it is subjected to by the feet of men and horses, and by the wheels of the tubs passing along the roads at a rapid speed, and the descent of large quantities of very fine dust from the screens on the surface along with the air-current descending the down-cast shaft, also occasion the deposition of dust.

Methods of dealing with Coal-Dust.—Some means must be adopted to deal with and render less dangerous these accumulations of dust, especially if fire-damp is given off and shots are being fired in the mine. Coal-dust may be dealt with by different methods, such as:—

(a) By removing it altogether; (b) damping it with salt or other hygroscopic substances; (c) damping it with water; (d) treating with stone-dust; (e) neutral or arresting zones.

Removing the dust altogether is impracticable in most collieries, on account of the time, labour, and expense it would involve. The dust that adheres to the sides, roof, and timber is usually the most dangerous because of its fineness, and even sweeping with a brush would not wholly remove it. Clearing part of the roadway, if this can be thoroughly done, and keeping it clean and well watered, may have some effect in checking the course of the flame if an explosion should occur.

Damping the dust with salt is carried out to a considerable extent, and is found, in many cases, to give better results than watering, but is much more expensive. Salt water has also been used instead of pure water, and gives good results, as it keeps the dust in a moist condition longer, and also tends to form a deposit over the particles. Water containing clay has been tried in preference to pure water, as it has a binding or caking action somewhat similar to that of salt water, but is neither so easily prepared nor used.

Damping by water is the method that is commonly used in most collieries. The watering may be either by (1) water tanks provided with a pipe pierced with holes; (2) spraying produced with very small nozzles fixed to vertical stand piping, which is connected to a water pipe-line laid along one side of the roadway; (3) similar sprays in which compressed air is employed for the purpose of pounding or dividing the water into very fine particles; (4) by means of watering hose 30 to 40 ft. long, attached at one end of a short stand-pipe which connects to a water pipe-line lying along the side of the roadway. Sometimes when a watering-cart is used it is provided with a revolving brush, driven from the axles, which sprays the water all round, sprinkling the roof, floor, and sides, and is much more effective than the common watering-cart (see fig. 627). Automatic tanks, with pumps attached, and worked by cranks from the axle, have also

been used. Watering by means of pipes is, however, the method that is most in use, and is no doubt the most effective.

A simple and serviceable arrangement is to lead a line of pipes about $1\frac{1}{2}$ in. or 2 in. diameter along the side of the road, and have taps which can be connected to a rubber hose with a 'rose' set so as to allow the water to escape in a spray, placed at intervals of 40 or 50 ft. The rubber hose should be 20 or 25 ft. long, so that a length of 40 to 50 ft. may be watered each time it is connected to the water pipes. The supply of water is usually obtained by connecting the pipes to the pumping set in the shaft, or to some tank placed so as to afford the required pressure.

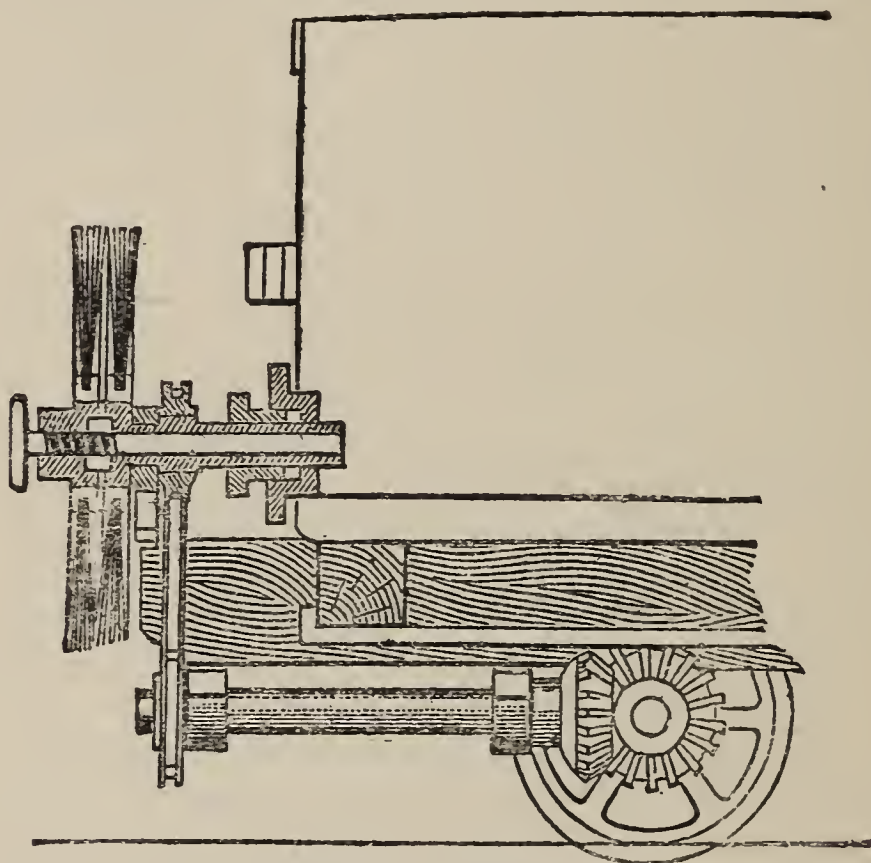


FIG. 627.—Water-cart.

In some collieries the water is allowed to issue from small jets in the form of very fine spray, at high pressure, in the intake air-currents, and is carried along to moisten the floor, roof, and sides. This method has the further advantage of keeping the air cool, and does not injure the floor so much as watering by a water-cart would do.

Water combined with compressed air has also been successfully employed for laying the dust, and is also effective in keeping the air cool. What is known as Martin and Turnbull's system, which is extensively used in the South Wales coal-field, is probably the best method of applying compressed air and water. Where compressed air is used for power purposes in the mine, the air in the spray-producer is taken from the compressed-air main by a $\frac{1}{2}$ -in. wrought-iron tube to the producer, which is generally fixed in the centre of the roadway. A water main is carried parallel with the

air main, and from it to the same producer another $\frac{1}{2}$ -in. tube conveys the water. Immediately before entering the spray-producer the air and water pipes are united. The air passes through a conical nozzle, whilst the water issues through a similar orifice around the conical water nozzle, where they are united in one stream.

Fig. 628 shows the arrangement of this appliance in elevation.

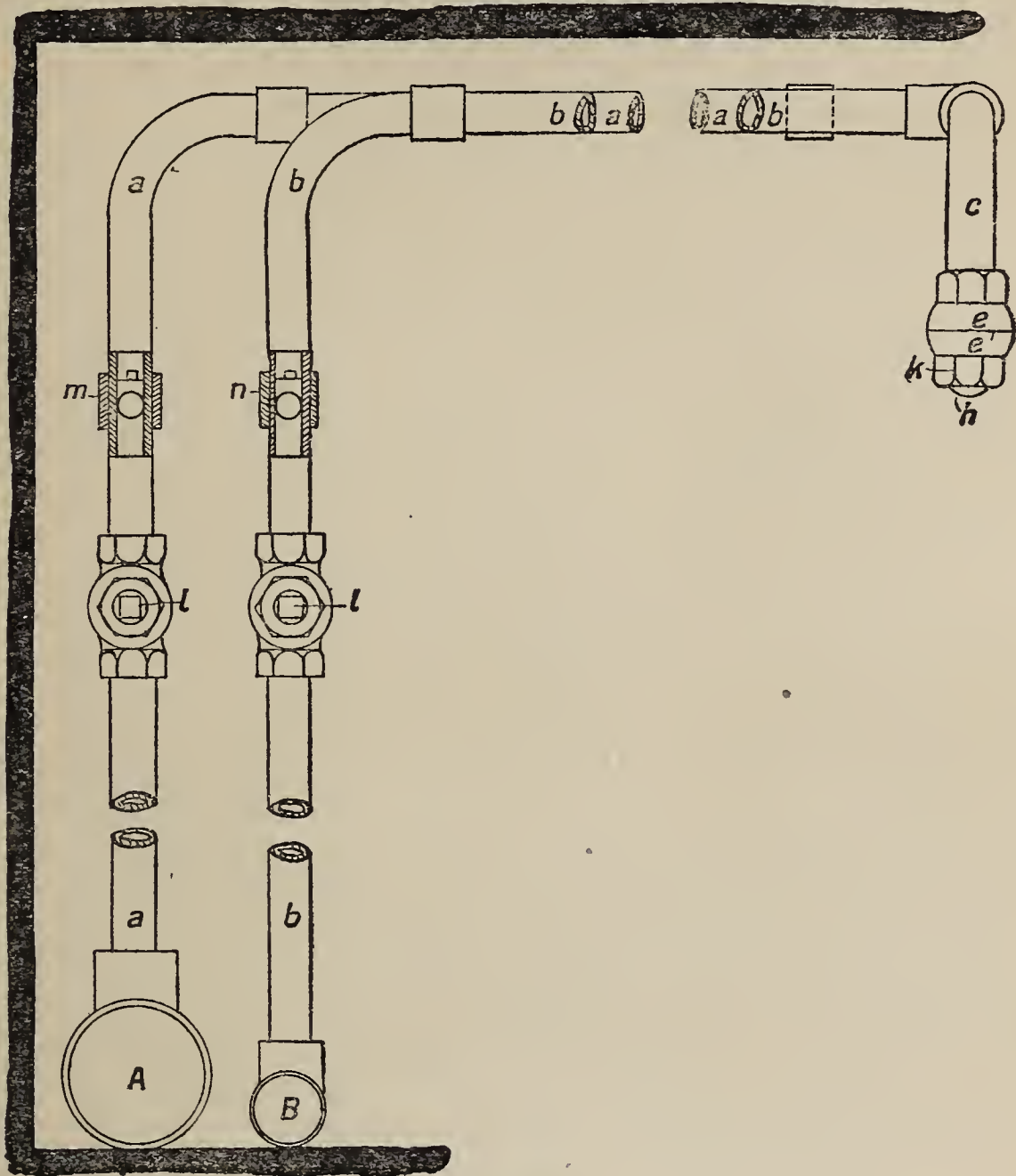


FIG. 628.

The compressed-air main A, and the water main pipe B, are laid parallel with one another, preferably at one side of the road. At any desired intervals in these mains, branch pipes *a* and *b* are carried up towards the roof. By these pipes the compressed air and water are carried into the delivery pipe *c*, into which the air pipe *a* extends in the form of a nozzle, terminating beyond the point of junction of the water pipe *b*, with the coupling *d*, as shown in figure.

The air and water are discharged into the mine through a terminal

instrument or nozzle, so constructed as to cause the issuing compressed air and water to form a fine spray or cloud of vapour. Another convenient form of spray-producer is composed of two cup-shaped members, of which one is inverted on the other. The cup is connected by screw-threads to the end of the delivery pipe, into which the compressed air and water are received from the pipes. The air and water pass by the orifices into the cavity formed by the cups. Extending downwards from the top cup is a spindle, the lower end of which is furnished with screw-threads. The lower cup fits loosely over the lower end of the spindle, and a washer is interposed between the cup and a shoulder formed on the spindle. By a nut working on the screwed part of the spindle, the cups are held towards each other, and the degree of fineness of the spray is governed by the extent to which the lower cup is screwed towards the upper one. The degree of fineness of the spray escaping between the rims of the cups is capable of the nicest adjustment, and, when required, the issue of the spray may be made to cease by forcing the lower cup sufficiently tight against the upper one. With this form of spray-producer a sufficiently fine spray may be obtained when water alone is used. Another form of spray-producer consists of a pipe, the end of which is flattened so as to present a narrow slit or orifice for the issue of the compressed air and water. The pressure of the water is in excess of that of the compressed air, and the supply of each is controlled by taps situated in the branch pipes. In the event of a cessation of the supply of compressed air, the water is prevented by a valve from passing into the main pipe; and similarly, should an accident happen to the water main, the compressed air is prevented by a valve from passing into it. By this system the extreme fineness of the spray is such that it is carried by the ventilating current in the mine for long distances, and effectually cools the air in the mine and damps the dust lurking in crevices and behind the timber, as well as on the roof, floor, and sides, without unnecessarily wetting the roads, which so often causes creep in the floor.

From investigations made in Germany on the subject of watering coal-dust, it would seem that the mere spraying of the dust, while it may improve the climatic conditions of the underground workings by cooling the atmosphere, would not greatly diminish the risk of fire-damp explosions. Samples of coal-dust taken, after watering, at Maybach Colliery, showed 2.05 to 3.65 per cent. of water, but the researches of the Prussian Fire-damp Commission have demonstrated that the explosive properties of coal-dust are not destroyed until it has taken up 50 per cent. of its weight in water. From this it would seem that, to prevent coal-dust explosions entirely, nothing short of systematic and thorough watering will be effective. Watering the dust is not a very satisfactory way of treating it, nor is it always a preventative of explosion. For this system to be effective the whole

of the dust would require to be drenched with sufficient water to form a liquid mud, and this is practically an impossibility. Besides, such a method would have a most disastrous effect on the roof and sides, especially where the strata is of a soft nature. Those best able to judge are coming more and more to the opinion that watering coal-dust is no real remedy or precaution against explosions. As M. Taffanel remarks: "The great drawback of systematic watering is that it must be repeated at very short intervals of time, owing to the rapid evaporation. The quantity of water indicated by the experiments should be present at all times, even just previous to the next watering. It is not, however, easy to ascertain what quantity of water remains in contact with the coal-dust, and in order to determine the requisite quantity of water it is necessary to sample the dust, a delicate and difficult operation. The most reliable method would be to remove all dust from the walls, and to keep such as is on the floor in a constant state of mud; but this is not easy to do, nor is it free from certain serious inconveniences." And again: "In the case of watering, one must have in contact with the dust about four times more water (by weight) than there is dust, that is to say, two or three times as much as would be necessary, theoretically, to form a liquid mud if the mixture were perfect. Whether such intense humidification could be easily realised at all times, and with the desired certainty, is a question that practical trial alone can decide."

Treating with Stone-Dust.—The application of stone-dust to accumulations of coal-dust to render it non-explosive has within recent years been brought into use, and is regarded as a more practical method of dealing with the danger than watering. The usual practice is to scatter fine ground stone- or shale-dust in some quantity amongst the coal-dust to render the latter non-inflammable. But here, again, the great difficulty is the inability to deal with the fine and dangerous accumulations of dust which gather on the sides and on the timbers. The whole question of dealing with these fine accumulations of coal-dust is probably one of the most difficult problems that the mining engineer has to face, in dealing with fiery and dusty mines. Extensive and careful experiments are at present being carried on with the stone-dust method, but whether it will effect a real solution of the question remains to be proved.

Arresting or Dustless Zones.—Once a coal-dust explosion has been initiated it becomes very difficult, almost impossible, to arrest its progress throughout the whole mine. The terrible results of the Courrieres and Hulton explosions demonstrated this very clearly. The question of arresting an explosion that has already become serious has therefore to be faced, and a remedy, if possible, found.

It was with this object in view that the method of arresting or dustless zones has been tried. A zone of this description simply means a length of roadway which has been specially treated, either by wholly removing the dust and keeping the whole perimeter of

the roadway in a perpetually wet or moist condition, or by treating the zone with stone-dust or a combination of stone-dust and water. To quote again M. Taffanel, he states that "as regards schistification (stone-dust treatment) the doubt is no less serious. With 75 per cent. of slate-dust, violent initial explosions developed over only 75 metres (254·25 ft.) continue to propagate themselves, even though the length of the arresting zone be increased to 150 metres (164 yds.) In the instances where favourable results were obtained the arresting zone contained nothing but pure slate-dust—free from any combustible matter—a condition that cannot be realised in practice. The question, however, arises whether in practice it would be possible to maintain a similar arresting zone in such a high condition of schistification, or whether, having attained that condition, by previously removing the coal-dust, there is not the risk of the proportion being quickly altered—especially on the surface of the stone-dust deposits, that is to say, in the portion of the deposit that is most easily stirred up—by the continuous deposition of fresh dust, far richer in combustible matter."

Concentrated schistification and concentrated watering, consisting of arrangements which may be described as arresting barriers of water or incombustible dust, were devised and tried at the Liévin testing station for the purpose of arresting violent explosions, instead of the ordinary watering and stone-dust treatment, which did not furnish satisfactory results. The object in view was attained by this concentrated system even under conditions far more stringent than in the previous experiments, since the length of the initial explosion was increased from 75 to 150 metres, whilst the deposit of coal-dust extended throughout the whole length of the gallery, *i.e.* 230 metres (251 yds.). It was also found that these arresting barriers succeeded equally in the case of medium or slow explosions. The concentrated system consists in placing a number of tilting troughs, containing water or stone-dust, in the arresting barrier. These troughs are fixed in such a way that they will discharge the water and stone-dust which they contain immediately an explosion takes place, but before the flame reaches the barrier. The troughs should be concentrated within a length of about 10 metres (11 yds.) in order to ensure the extinction of violent explosions. In the experiments at Liévin the arresting barrier proved successful with 90 litres (20 gallons) of water per square metre ($10\frac{3}{4}$ sq. ft.) of sectional area of the gallery. M. Taffanel says that "prolonged experiments on a larger scale are necessary to decide the question whether the arresting barriers are really more efficacious than ordinary watered or schistified zones." The arresting barriers or dustless zones which have been installed in some collieries in England usually consist of lengths of roadway in which brick or concrete arching has been erected to enable the coal-dust to be more readily removed, and the length of arching made thoroughly wet or treated with stone-dust. The

writer is not aware whether any of these dustless zones have ever had a practical test to prove their worth as a preventative. The difficulty will be to fix a proper length to be capable of arresting the most violent explosion.

The following problems are given to illustrate further the application of some of the formulae given in the foregoing chapter on ventilation :—

Example. — Two airways, one 10 ft. $\times$ 5 ft. $\times$ 300 fms. long, and the other 4 ft. $\times$ 6 ft. $\times$ 500 fms. long, circulate a total quantity of 30,000 cub. ft. per minute. What quantity of air would, at a pressure of 1 lb. per sq. ft. at the down-cast shaft, pass into each under the same conditions?

Let the 10 ft. $\times$ 5 ft. airway be designated x and the other y .

In the case of x

$$PA = KSv^2$$

$$1 \times 50 = .01 \times 30 \times 1800 \times v^2$$

$$1 = 108 v^2$$

$$\therefore v^2 = \frac{1}{108}, \text{ and } v = \sqrt{.00925} = .390$$

$$\text{Now } V = .390 \times 1000 = 390 \text{ ft. per minute}$$

$$\text{And } Q = V \times A = 390 \times 50 = 19,500 \text{ cub. ft. per minute}$$

(Quantity of air circulating in x).

$$PA = \frac{y}{KSv^2}$$

$$1 \times 24 = .01 \times 20 \times 3000 \times v^2$$

$$1 = 25 v^2$$

$$\therefore v^2 = \frac{1}{25} \text{ and } v = \sqrt{.04} = .200$$

$$V = .200 \times 1000 = 200 \text{ ft. per minute}$$

$$Q = V \times A = 200 \times 24 = 4800 \text{ cub. ft. per minute}$$

(Quantity circulating in y).

$$\therefore x + y = 19500 + 4800 = 24300 \text{ cub. ft.}$$

circulating with a pressure of 1 lb.

But the total quantity circulating in both airways was 30,000 cub. ft. per minute.

$$\therefore (\text{by proportion}) 24,300 : 30,000 :: 4800 : y$$

$$\text{and } y = \frac{30,000 \times 4800}{24,300} = 5925.92 \text{ cub. ft. per minute,}$$

$$\text{and } 30,000 - 5925.92 = 24,074.08 \text{ cub. ft. per minute} = \text{quantity passed by } x.$$

Example.—If two airways of the same area pass a total quantity of air equal to 10,000 cub. ft. per minute, subject to the same pressure in each case, the resistance in the airways being in the proportion 5 to 1, what proportion would each airway pass respectively?

This problem may be worked out in same way as the preceding one, or by the formula represented.

$$R = \sqrt{\frac{A^3}{S}}, \text{ where } R \text{ is the relative quantity going into each.}$$

Let the two airways be represented by x and y , and let A equal 1 in both cases.

$$\text{Then } R = \sqrt{\frac{A^3}{S}} = \sqrt{\frac{1^3}{1}} = 1 = \text{relative quantity in } x,$$

$$\text{and } R = \sqrt{\frac{1^3}{5}} = .447 = \text{relative quantity in } y.$$

The actual quantities passing into x and y will therefore be found by proportion,

$$\text{thus } 1.447 : .447 :: 10000 : y = \frac{.447 \times 10,000}{1.447} = 3089.15 \text{ cub. ft. per minute,}$$

$$\text{and } x = 10,000 - 3089.15 = 6910.85 \text{ cub. ft. per minute.}$$

Example.—Find the total quantity of air per minute passing in an airway 10 ft. $\times$ 7 ft. $\times$ 2000 ft., at a pressure of 8.5 lbs. per sq. ft.

$$PA = KSv^2$$

$$\text{Here } P = 8.5 \text{ lbs. ; } A = 70 \text{ sq. ft. , } \text{Per} = 34 \text{ , } S = 34 \times 2000 \text{ ; } v^2 = \left(\frac{Q}{1000 \times A} \right)$$

$$8.5 \times 70 = .01 \times 34 \times 2000 \times v^2$$

$$595 = 680v^2$$

$$\therefore v^2 = \frac{595}{680} \text{ ; and } v = \sqrt{.8750} = .9354,$$

and Q the total quantity = $.9354 \times 1000 \times 70 = 65,478$ cub. ft. per minute.

Example.—If the quantity of air passing round a mine is 10,000 cub. ft. per minute before splitting, when the size of airway is 6 ft. $\times$ 5 ft. $\times$ 1200 ft. long, what quantity will circulate if the current is split respectively into two, three, and four equal parts, the pressure and other conditions remaining the same?

Here the pressure and rubbing surface will be the same in each case, therefore the quantity will vary as $\sqrt{A_3}$ or $Qn = \sqrt{n} \times n \times Q$.

$$(1) Qn_1 = \sqrt{2} \times 2 \times 10,000 = 28,280 \text{ cub. ft. in two equal splits.}$$

$$(2) Qn_2 = \sqrt{3} \times 3 \times 10,000 = 51,930 \text{ cub. ft. in three equal splits.}$$

$$(3) Qn_3 = \sqrt{4} \times 4 \times 10,000 = 80,000 \text{ cub. ft. in four equal splits.}$$

Example.—A down-cast shaft, 14 ft. diameter and at a temperature of 50° F., passes 121,000 cub. ft. per minute. What size of shaft would the up-cast require to be, if the velocity of air current in both shafts is to be equal, supposing the temperature in the up-cast is 100° F.?

$$\text{Velocity in down-cast} = \frac{Q}{A} = \frac{121,000}{14^2 \times .7854} = 786.1 \text{ ft. per minute.}$$

$$\text{Increase of volume in up-cast} = \frac{Q \times t}{459} = \frac{121,000 \times 50}{459} = 13,180.82 \text{ cub. ft.}$$

$$\therefore \text{total volume in up-cast} = 121,000 + 13,180.82 = 134,180.82 \text{ cub. ft.}$$

$$\text{and area of up-cast} = \frac{Q}{V} = \frac{134,180.82}{786.1} = 170.69 \text{ sq. ft.}$$

$$\therefore \text{diameter of up-cast} = \sqrt{\frac{170.69}{.7854}} = 14.74 \text{ ft.}$$

Example.—Two shafts, each 15 ft. $\times$ 5 ft. $\times$ 100 fms. deep, are connected by a drift 11 ft. $\times$ 5 ft. $\times$ 400 yds. long, the quantity of air passing being 30,000 cub. ft. per minute. Find the quantity that would pass if another drift were added 11 ft. $\times$ 5 ft. $\times$ 400 yds. long, (1) with same pressure, (2) with same horse-power?

(1) The pressure required will be the same for each shaft.

$$\text{Now } P \times A = K \times S \times v^2$$

$$\therefore P \times 15 \times 5 = .01 \times 40 \times 600 \times \left(\frac{30,000}{75 \times 1000} \right)^2$$

$$\therefore P = .01 \times 8 \times 40 \times \left(\frac{2}{5} \right)^2 = .51 \text{ lb. per sq. ft.}$$

and the pressure in each shaft = $.51 \times 2 = 1.02$ lbs.

$$(2) \text{ Find pressure in drift. } P \times 55 = .01 \times 32 \times 1200 \times \left(\frac{30,000}{55 \times 1000} \right)^2$$

$$P \times 11 \times 121 = .01 \times 32 \times 240 \times 36$$

$$\therefore P = \frac{.01 \times 32 \times 240 \times 36}{11 \times 121} = 2.07 \text{ lbs. per sq. ft.}$$

If another drift of the same size were added, the velocity would be halved, and therefore would only require $\frac{1}{4}$ the pressure and $\frac{2.07}{4} = .517$ lb. to pass 15,000 cub. ft. in each airway.

Pressure in shaft and drift before splitting $= 2.07 \times 1.02 = 3.09$ lbs.

„ after adding an additional airway $= 1.02 \times .517 = 1.537$ lbs.

Now quantity $\propto \sqrt{\text{pressure}}$. $\therefore \sqrt{1.537} : \sqrt{3.09} :: 30,000 \text{ cub. ft.}$
 $\therefore 1.23 : 1.75 :: 30,000 : x$

and $x = \frac{30,000 \times 1.75}{1.23} = 42,482.92$ cub. ft. per minute (when the second drift is added).

H.P. before adding airway $= \frac{30,000 \times 3.09}{33,000} = 2.8$

„ after „ „ $= \frac{42,482.92 \times 1.537}{33,000} = 1.97.$

Example.—If the difference in temperature between the up-cast and down-cast shafts is increased four times, how is the pressure and quantity altered?

(1) The pressure varies as the difference of temperature between the up-cast and down-cast, so if the latter increases four times, the pressure will also be increased four times.

(2) The quantity varies as $\sqrt{\text{difference of temperature between the two shafts}}$. Therefore increasing the temperature four times increases the quantity by the $\sqrt{4} = \text{twice}$.

CHAPTER XIV.

SAFETY LAMPS AND RESCUE APPARATUS.

It is now over ninety years since Sir Humphry Davy crowned his long and patient researches by the invention of his safety lamp, an event which marked a new epoch in coal-mining. Before Davy's time the miner had to rely on very insecure methods for detecting or for working in the presence of fire-damp. At the beginning of the nineteenth century Spedding's steel mill was the only apparatus which enabled him to continue his work in the presence of small accumulations of gas. By this machine, which was soon demonstrated to be unsafe, and which was the cause of numberless explosions, the miner was enabled to work by the faint light of an intermittent spark resulting from the contact of a piece of flint with a revolving steel disc.

Since the introduction of the Davy lamp, large numbers of other safety lamps have been patented, but all are on practically the same principle as Sir Humphry Davy's, although some are, of course, a great improvement on the original, both as regards safety and lighting power.

Definition.—"Safety lamps are contrivances by which a light, surrounded by an explosive mixture of fire-damp and air, may be maintained in lamps without communicating flame to the outside atmosphere." As at present constructed, they depend upon the fact that flame, when brought in contact with wire gauze of certain degrees of fineness, cannot pass through it, owing to the rapidity with which the heat is conducted away, so that it cannot be communicated to the outside atmosphere.

Sir Humphry Davy first demonstrated this by some experiments he made with metallic tubes. He found that it was easy enough to effect an explosion of fire-damp and air in a wide vessel, but that it was impossible to effect it in a narrow metallic tube.

Metallic tubes of $\frac{1}{8}$ of an inch in diameter and $1\frac{1}{2}$ in. long prevented an explosion, and this phenomenon, according to Davy, probably depended "upon the heat lost during the explosion in contact with so great a cooling surface, which brings the temperature of the first

portions exploded below that required for the firing of the other portions ; and it has been already shown that the fire-damp requires a very strong heat for its inflammation. Mixture of the gas with air I found, likewise, would not explode in metallic canals or troughs, when their diameter was $\frac{1}{4}$ of an inch., and their depth considerable in proportion to their diameter, nor could explosions be made to pass through such canals. Explosions, likewise, I found would not pass through very fine wire sieves or wire gauze." Now wire gauze is nothing more than a series of small tubes, having very small diameters,

and of very short lengths. The wire gauze mostly used for safety lamps has 784 apertures to the sq. in.

Davy Lamp.—This lamp, as originally constructed, consisted of a small cylindrical vessel *b* for holding the wick and oil, provided at the bottom with a pricker *f* for trimming the former, and surmounted by a cylinder of wire gauze *a*, made double at the top, and supported by small iron rods *c*, terminating in the cover *d*, to which is attached a handle. The gauze cylinder is about $1\frac{1}{2}$ in. diameter, and 7 in. long, with wires about $\frac{1}{50}$ of an inch in diameter, having, as already stated, about 784 openings per sq. in., through which air enters to keep the flame burning freely. If a certain percentage of fire-damp enters along with the air, the mixture will ignite and fill the space inside the gauze with flame ; but as soon as this flame comes into contact with the wire gauze, it is immediately cooled down, and cannot pass through the opening unless it is allowed to burn until the gauze becomes heated to a

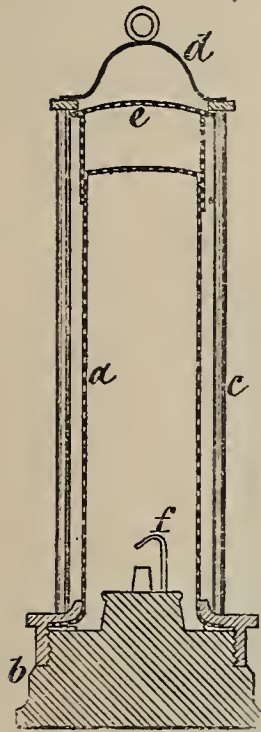


FIG. 629.—Davy Lamp.

certain temperature, when the flame can pass through to the outside.

An explosion may be brought about by a 'Davy' lamp in several ways, such as :—

By allowing gas to burn inside until the gauze becomes red hot.

By allowing a strong current of air to blow against the lamp, thus forcing the flame through the gauze, which occurs when the air attains a velocity of about 5 ft. per second.

By a sudden jerk or shock to the lamp, or by a shock due to heavy blasting operations.

By the miner carelessly damaging his lamp or opening it in the presence of an explosive mixture.

The great disadvantage of the 'Davy' lamp is the very poor light that it gives, $\frac{1}{15}$ to $\frac{1}{30}$ of a candle power only.

The 'Davy' lamp in its original form is little used now, being unsafe in most collieries, where the air now travels at such high velocities round the workings.

Clanny Lamp.—This lamp is similar to the 'Davy' lamp in con-

struction, but has a glass cylinder *a* instead of the lower portion of the gauze, which enables it to give a much better light, and to be more easily carried in an air current; but, on the other hand, it causes an explosion more readily, owing to the area of gauze cylinder being smaller than in a 'Davy.' It is also unsafe in a very strong current of air, as it will readily pass the flame when the air is travelling at a velocity of 6 or 7 ft. per second. This lamp is not now used unless it has an additional protection in the shape of an iron shield surrounding the outside of the gauze.

Stephenson Lamp.—The essential points of dissimilarity between a 'Davy' lamp and a 'Stephenson' lamp are that, whereas in the former the flame is simply surrounded by a wire gauze through

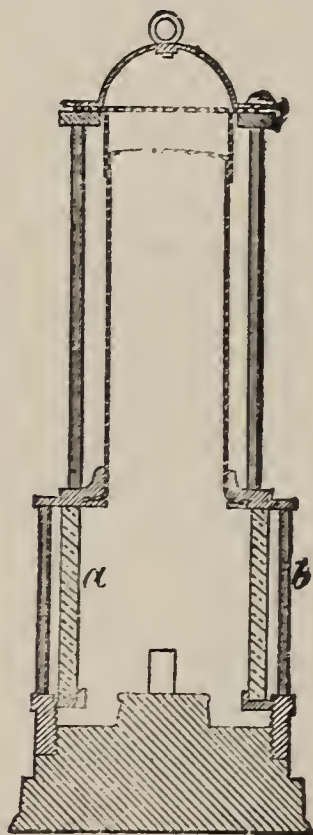


FIG. 630.—Clanny Lamp.

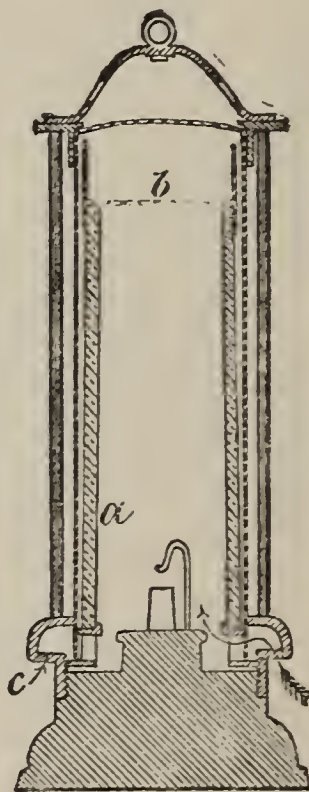


FIG. 631.—Stephenson Lamp.

which the air passes, in the latter the flame is surrounded and burns within a glass cylinder *a*, covered at the top with a perforated copper cap *b*, and is fed by air passing through perforations in a metal ring *c* at the bottom. The lamp has in addition the wire gauze of the 'Davy,' and if the glass happens to get broken it still remains safe.

Like the 'Davy,' the 'Stephenson' lamp has a very small illuminating power, and is very readily extinguished, but is very much safer than the former in air currents, as it will not 'pass' the flame until the velocity of the air reaches 8 to 10 ft. per second.

This lamp is now little, if at all, used in fiery mines.

Marsaut Lamp.—The 'Marsaut' lamp, which was the invention of a well-known French mining engineer, differs very little from the 'Clanny' lamp. Instead of having a single gauze like the

'Clanny,' it is provided with two or three gauzes fitted into the inside of each other, which tends greatly to increase the safety of the lamp.

As used in Britain it is made with two gauzes only, with the addition of an iron shield as a further protection, and in this form it is a very safe kind of lamp for use in fiery mines.

The Marsaut lamp is largely used both in England and Scotland, and has much to recommend it, as it does not pass the flame until a very high velocity of air current is reached, and it has the further advantage of not being so easily extinguished as some other forms of safety lamps. For oncast men especially, such as roadsman, pony-drivers, etc., the Marsaut is to be recommended, as the nature of work those persons are engaged in varies greatly from that of

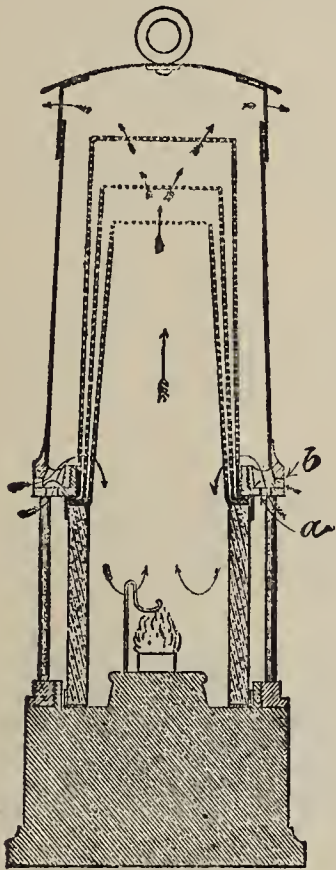


FIG. 632.—Marsaut Lamp.

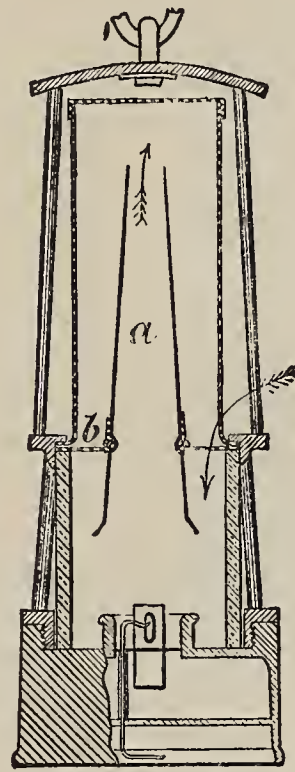


FIG. 633.—Mueseler Lamp.

a miner working at the coal-face, where a lamp is used under more favourable circumstances, being in most cases kept perfectly still or at most moved about only over small areas. In collieries where there is only a small amount of fire-damp given off, this lamp is often used without the outer shield, and with only two gauzes, which makes it much better suited for underground requirements, as the amount of light given out is much increased by the freer supply of air that the flame receives, and the latter is not so easily extinguished as when the shield is used.

Mueseler Lamp.—The Mueseler lamp is constructed somewhat like the Marsaut. It has the glass cylinder round the flame in the same way, but instead of having two or three gauzes it has a single gauze only, with a conical-shaped metal chimney *a* (fig. 633)

fitted inside, immediately above the glass cylinder, to which it is fixed by a ring of gauze *b*. The air to feed the flame passes first through the outer gauze, then through the gauze cap between the metal chimney and the glass to the flame, and the products of combustion pass up the metal chimney through the gauze cylinder and into the atmosphere. The metal chimney has thus a double purpose to serve, viz., to create a strong upward draught and to insure the inlet air being drawn down close to the glass to keep it cool. When fire-damp is suddenly ignited in this lamp the resulting gases, principally CO_2 , fill up the conical chimney and speedily extinguish the flame.

The great drawback to this lamp is the readiness with which the light is extinguished, it being very sensitive to the least shock or jerk, while, if held slightly out of the perpendicular, the light at once goes out through the supply of air being cut off. Nevertheless, it is extensively used both in Britain and in other European countries, and is both good and safe where there is plenty of ventilation.

The Royal Commissioners on Accidents in Mines objected to this lamp on the following grounds:—

- (1) It is very easily put out.
- (2) The glass is easily broken by a blow, by the flame playing on it, or by cold water coming in contact with the hot glass.
- (3) There are difficulties in getting tight joints where the metal ring and glass cylinder meet.
- (4) Difficulties arise from combustion with a tendency to smoke the glass, thus lowering the illuminating power.

These objections might also be urged against nearly all the safety lamps at present in use.

Hepplewhite-Gray Lamp.—In this lamp and its newer modifications, the construction differs somewhat from any other safety lamp; the differences consisting chiefly of the manner the air is admitted to feed the flame, and also in the shape of the glass cylinder surrounding the latter (fig. 634).

The standards for supporting the lamp, instead of being solid, as they are in other lamps, are made of tubes down which the air passes to an annular chamber, situated immediately over the oil vessel, and protected by wire gauze.

In the form in which this lamp is now made, there are only three inlet tubes instead of four as formerly, one of the tubes being considerably broader than the others and acting as a deflector.

The glass surrounding the flame, instead of being cylindrical in shape, is made in the form of a cone, and is very much longer than the glass of an ordinary safety lamp. Immediately above the glass is a gauze, also of conical shape, and outside that a cone of metal, the whole forming a very strong compact lamp. In addition it is fitted with a shut-off arrangement, so that the air can be admitted either at the top or bottom of the tubes. In the new form a shield-plate *a a*, which is part of the hood, projects over and completely covers

the top of the inlet tubes, and prevents the lamp from being extinguished as readily as the older type. The air passes down the tubes, while the products of combustion pass upwards and out into the atmosphere by two horizontal rows of holes *b b*, in the hood *c*. To facilitate the cleaning of the lamp, the ring securing the glass is screwed on to a vertical plate *d*, which forms the air inlet. This arrangement enables the whole of the inside to be quickly and easily removed when the lower gauze ring is unscrewed. It is claimed for this lamp that it gives a much superior light to any other form, owing to the shape of the glass, which allows the rays of light to be projected in all directions, thus permitting the roof to be examined with ease without tilting the lamp. It is also claimed for it that very small

quantities of gas can be readily detected by its means. The author has had little practical experience of this lamp, but some time ago some practical firemen and overmen, who were attending a class to which he was lecturing, made, under his directions, a number of tests with the Gray lamp, and at the same time with Mueseler and Marsaut lamps; and reported that the former could be used to detect small quantities of

gas much more easily and quickly than either of the others. The only objections to the use of the Gray lamp are its weight and expensiveness, but there can be little doubt that it is a very superior lamp for use underground. It is very safe in every respect and can withstand almost any current of air met with in collieries, having been tested with velocities up to 100 ft. per second without passing the flame.

Wolf Safety Lamp.—This safety lamp, constructed to burn benzine, is very largely used on the Continent. It is somewhat similar to an ordinary Marsaut lamp, with two gauzes, and is provided with a corrugated shield, provided with apertures, which is said to give it additional safety and allows of its burning more freely.

It is also provided with an 'igniter' for relighting it if extin-

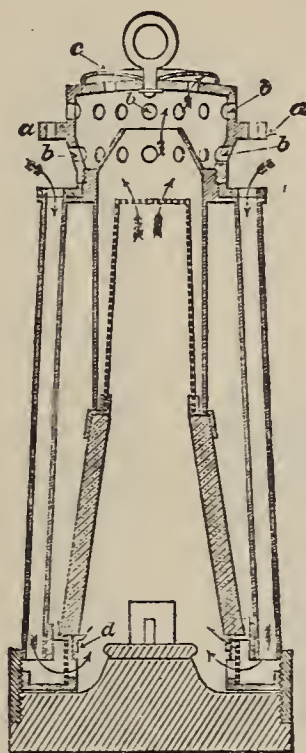


Fig. 634.—Hepplewhite-Gray Lamp.

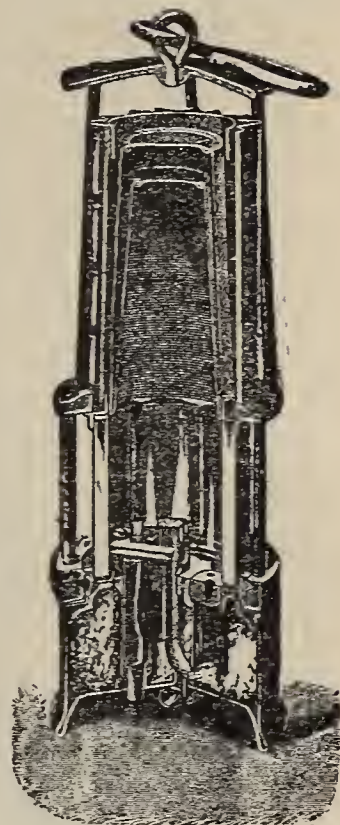


Fig. 635.—Wolf Lamp.

guished, without rendering it necessary to unscrew the bottom of the lamp, which is a great advantage. The fastening is secured by a magnetic lock, and cannot be opened without the aid of a powerful

magnet. It is said to give a much better light than an ordinary safety lamp burning oil, and can withstand strong air currents, and detect small percentages of gas, while the cost of fuel is low compared with other lamps.

Wolf-Dahlmann Lamp.—This may be said to be an improved Wolf lamp. In it the air is conducted from above the flame through a gauze ring $1\frac{1}{2}$ ins. in height, which is covered by a movable brass cylinder. The products of combustion are carried off by an inner brass cylinder, at the top of which is fixed a wider cylinder (see fig. 636), the latter being capped with wire gauze. Inside the inner brass cylinder is a gauze chimney, so arranged that it can be taken out for the purposes of cleaning. By means of this simple arrangement the products of combustion rapidly pass off, while fresh air enters from all sides through the gauze to the flame without becoming mixed with the products of combustion. In this manner a good circulation results, which causes the flame to burn steadily and brightly, while it is easy to light the lamp while locked by means of the igniter which is placed within it.

* *Evan Evan's Lamp.*—This is a bonneted Clanny lamp, with the bonnet, which extends from the flange above the glass to the dome

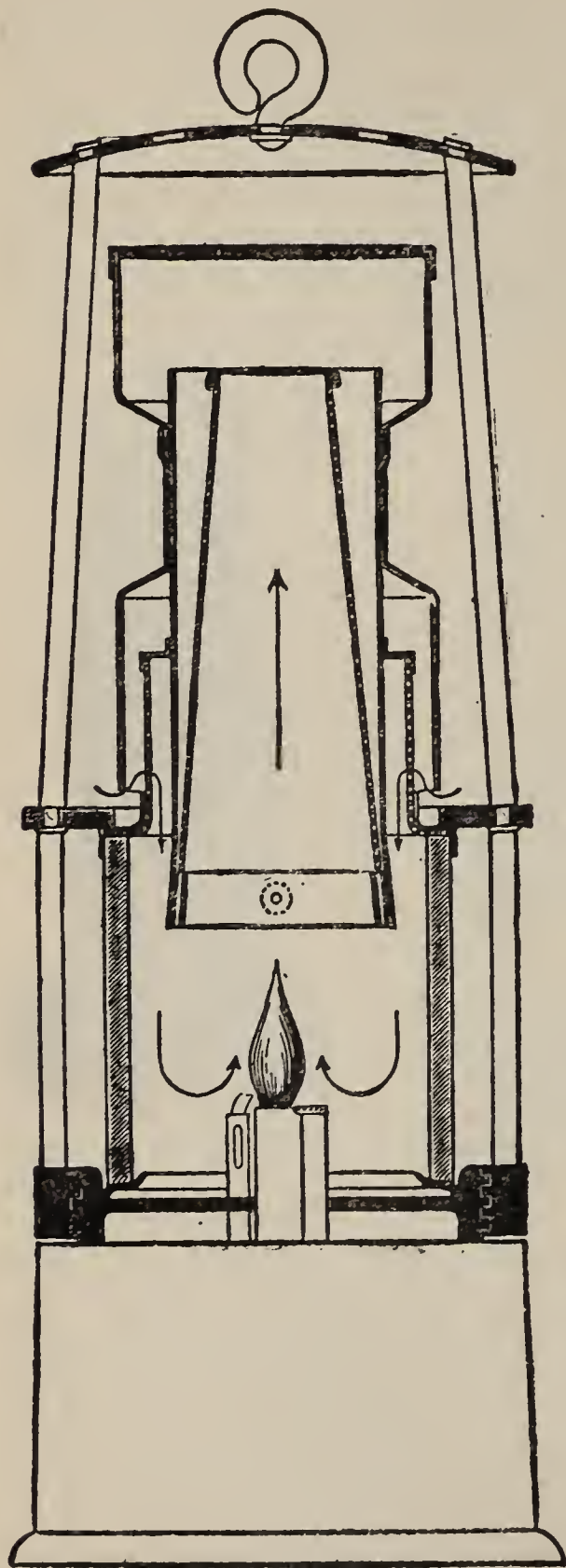


FIG. 636.—Wolf-Dahlmann Lamp.

of the lamp case, fixed permanently. The air is admitted through a series of holes in the horizontal flange above the glass. The products

* *Report of Royal Commission on Accidents in Mines*, p. 206.

of combustion escape through a series of holes in the top of the bonnet (see fig. 637). This lamp is also provided with an automatic arrangement for closing it in the event of gas becoming ignited inside the gauze, the main features of this arrangement being as follows:—Within the bonnet and surrounding the gauze are two cylinders of the same height closely fitting one another. The inner cylinder is open at the top and is perforated near the bottom with fifteen holes, each $\frac{1}{4}$ in. in diameter. The outer cylinder, which is closed at the top, is perforated near the top, with a similar series of holes. A rod slides through a tube and is maintained in position by a loop of thread close to the gauze and stretched between two hooks. If this loop gets burned through, by gas burning inside, the rod is

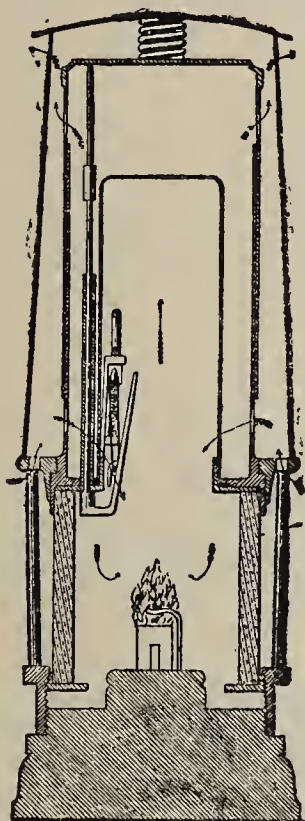


FIG. 637.—Evan Evan's Lamp.

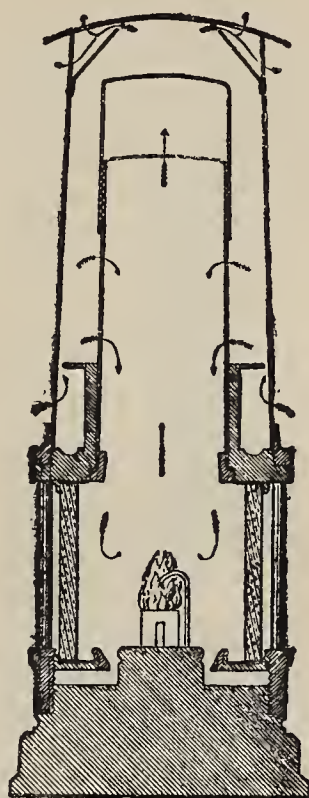


Fig. 638.—Evan Thomas Lamp.

no longer held in position, and a strong spiral spring between the top of the outer cylinder and the dome of the lamp pushes the former down over the inner cylinder, and thus closes both series of holes simultaneously, and extinguishes the flame in a few seconds.

Evan Thomas Lamp.—This was one of the lamps which was tested by the Mines Commission, and reported upon as giving very satisfactory results. In its principles and construction it is an improved form of the Clanny lamp. At the bottom of the gauze cylinder is a close fitting brass ring or tube about 1 in. high, the top of the ring terminating in a horizontal flange, which extends to within about $\frac{1}{16}$ of an inch of the outside shield or bonnet. Through this small space the inlet air is admitted to feed the flame, the products of combustion passing out at the top of the bonnet.

The Evan Thomas lamp is simple in construction, gives a fairly good light, and is safe in currents of air travelling with a velocity of over 50 ft. per second. It has also the advantage of not being very readily extinguished by a sudden jerk or by being held on the slant. In an explosive atmosphere, however, it soon becomes extinguished.

The following table of experiments with safety lamps in currents of air of different velocities is interesting, as it shows the behaviour of different lamps and the ratio of safety under these conditions.

Name of Lamp.	Percentage of Gas present in air.	Velocity of Air in ft. per minute.	Duration of Experiment.	Results.
Davy lamp, . . .	7 per cent.	600	10 seconds.	Explosion.
„ . . .	7 „	400	60 „	„
„ . . .	7 „	370	180 „	No explosion.
Davy (shielded), .	7 „	800	60 „	„
„ (in can), . . .	7 „	1500	60 „	„
Stephenson lamp safe up to 800 ft. per minute.				
Clanny „ „	„	600	„	„
Mueseler lamp, .	8 per cent.	2888	5 „	Lamp went out.
„ „ . . .	13 „	2888	75 „	Continued to burn.
„ „ . . .	13 „	600	6 „	Explosion.

The Marsaut lamp gave the same results as the Mueseler, but it was found that if the current of air was reversed while being admitted to the flame, an explosion occurred in 5 seconds. The number of safety lamps that have been patented and placed on the market is so large that it would take a large volume to describe fully them all. One writer enumerates over seventy, and that by no means exhausts the list.

Safety lamps have now to be of an approved type (see Coal Mines Act, 1911, section 33), and have to undergo certain specified tests at the Government testing station at Eskmeals in Cumberland. If a lamp successfully passes the tests, it is approved for use in mines.

Deflector Lamp.—This lamp, which has passed the Home Office test and is made by Messrs Richard Johnson, Clapham & Morris, Ltd., of Manchester, is so constructed that when the shield is fitted in position it forms an upper and a lower chamber. Near the centre of the lamp a deflector ring is fitted with a baffle ring below, there being an air space of 0·855 sq. in. below the deflector ring, and a similar air space of 0·83 sq. in. at the intake of the middle collar of the baffle ring. This construction of the air passages enables the lamp to burn in a vitiated atmosphere, and to be safe in velocities up to 4000 ft. per minute, without passing the flame. It is constructed to burn either paraffin or mineral colza oils, and at the end of ten hours is said to give 0·65 candle power. The lamp is fitted with a winding burner, for the purpose of lowering the flame when testing for fire-damp, the burner being lowered or raised by means of a small milled-headed screw in the bottom of the lamp. This is an

improvement on the old type of lamp in which the flame has to be manipulated by a pricker, which is so apt to extinguish the light when a test is about to be made. With this winding burner, combined with the construction of the upper portion of the lamp, the percentage of fire-damp present in the air can be ascertained very accurately. The lamp can be fitted with either a lead or magnetic lock, and also for electrical ignition if required. An important feature of the deflector lamp is that it is so constructed that it is impossible to fit it up without both gauzes being in position. The arrangement for obtaining this can also be fitted on to the bonneted Clanny lamp with a single gauze.

Patterson Lamp.—The Patterson lamp is made in various types, five of which types have been approved by the Home Office for use in mines.

Type 'A 1' and Type 'A 3.'

Each of these lamps, the general design of which is shown in fig. 640, is a double-gauze, flame, oil lamp, with an air-feed through vertical holes in the bonnet ring. Each consists of the following essential parts:—

1. Bonnet or shield of steel, with riveted seam and with a separate securely riveted crown, or of seamless steel, the bonnet and crown being in one piece. Furnished with outlet holes immediately below the crown. Bonnet ring of brass, steel, or iron, riveted to the bonnet, screw-threaded to fit the pillar ring, and provided with twenty-four vertical air-inlet holes of total area not greater than 1.2 sq. in.

2. Pillar ring of brass, steel, or iron, screw-threaded to take the bonnet ring, and locked thereto by means of a sliding pillar kept in position by the oil vessel. Pillars of steel or iron, five or more (one sliding), so arranged that a straight line touching the exterior part of consecutive pillars does not touch the glass. Bottom or lock ring of brass, steel, or iron, provided with a fixed seating for the glass, and having, when fitted with a magnetic lock, two or more ratchet teeth to engage with the bolt of the lock; provided or not with the patent electric ignition device

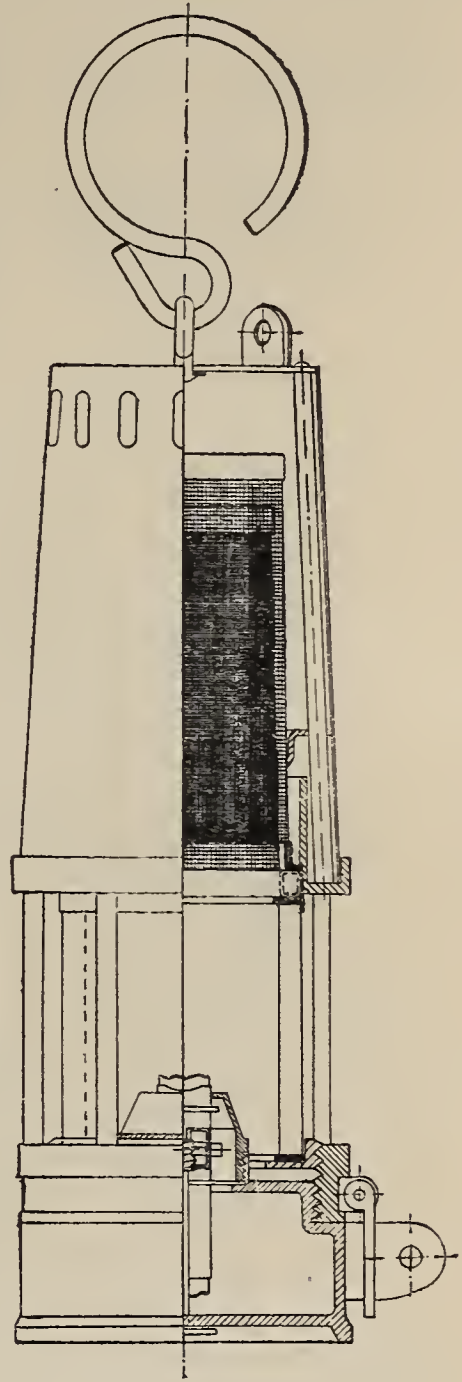


FIG. 639.—Deflector Lamp.

shown in fig. 640, so fitted as not to cause the lamp to be dangerous in an explosive atmosphere.

3. Gauzes of not less than 28 S.W.G. steel wire, 784 meshes to the square inch, with secure flame-tight double-lap seams, or butt seams with folded edge secured on the inside by a folded metal strip; each gauze formed to fit flanges of the outside and inside base rings, and so secured to the same by punch indentations or by riveting as to make strong and flame-tight joints; the gauzes being so constructed

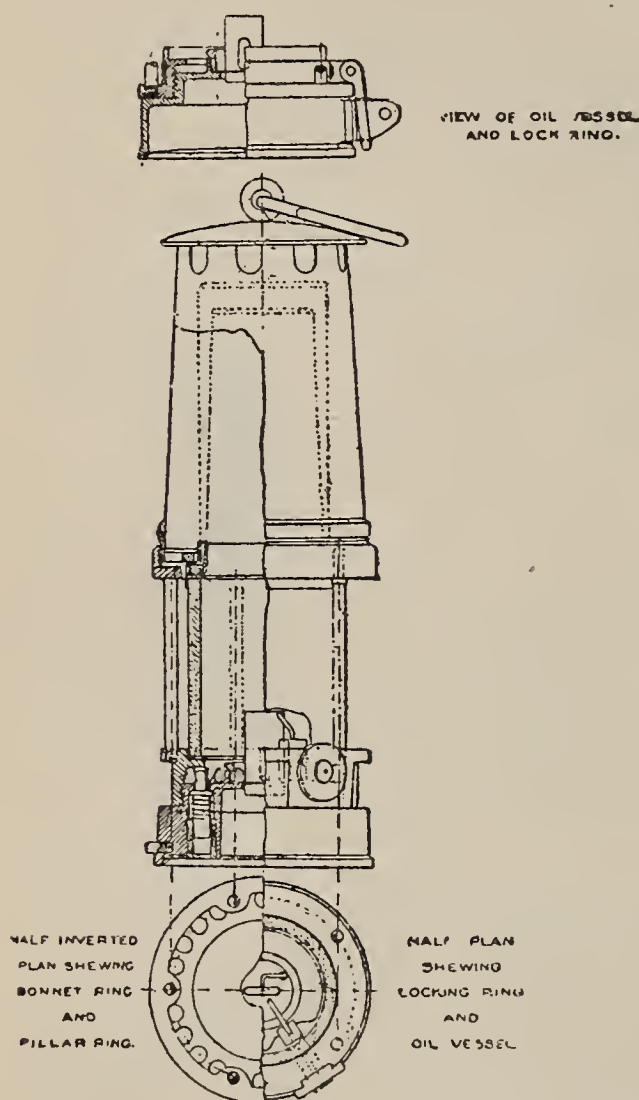


FIG. 640.—Patterson Lamp.

as to form the seating for the glass necessary to hold it firmly in position, thereby preventing the lamp from being put together without them. The lamp may be fitted with a Mueseler tube and horizontal gauze in place of the inner gauze; the horizontal gauze forming a secure and flame-tight joint with the Mueseler tube. Also the lamp may be fitted with a single gauze, of the dimensions of the outer gauze given above (with or without a gauze cap).

4. The glass furnished with top and bottom asbestos washers to ensure flame-tight joints with the gauzes and lock-ring seating.

5. Oil vessel of brass, steel, or iron, of sufficient capacity to provide the required light for the required time; fitted with a flat $\frac{1}{2}$ -in. burner; provided or not with a winding wick adjuster, and so fitted as not to

cause the lamp to be dangerous in an explosive atmosphere.

6. Locking devices:—In the 'A 1' lamp, one or other of the following may be used: (1) A vertical iron bolt (held in position by a spiral spring) working within a closed or open-ended brass tube, as shown in fig. 639, and so constructed that the bolt can be withdrawn only by means of a powerful electro-magnet. (2) An efficient lead-rivet lock, with a horizontal or vertical lug or hinge securely attached to the lock ring or to a pillar, and a horizontal or vertical lug or staple securely soldered to the bottom.

Fire-damp Indicators.—Within the last few years a large number of instruments for detecting small percentages of fire-damp have

been brought before the mining public, but very few of them are of much practical use to the miner, most of them being too complicated in design, too sensitive to be handled freely, and too expensive.

Hydrogen Indicator.—It has long been known that the pale hot flame of hydrogen gas is very sensitive to the presence of fire-damp, even when the latter is present in very small quantities. Professor Clowes has invented a lamp in which the hydrogen flame is used for the detection of small percentages of CH_4 . The accompanying figure (641) shows this apparatus as used in conjunction with an ordinary Heppelwhite-Gray safety lamp. A small cylinder C, containing hydrogen at very high pressure, is attached to the lamp, and at the bottom of the cylinder a tube B, of small diameter, is con-

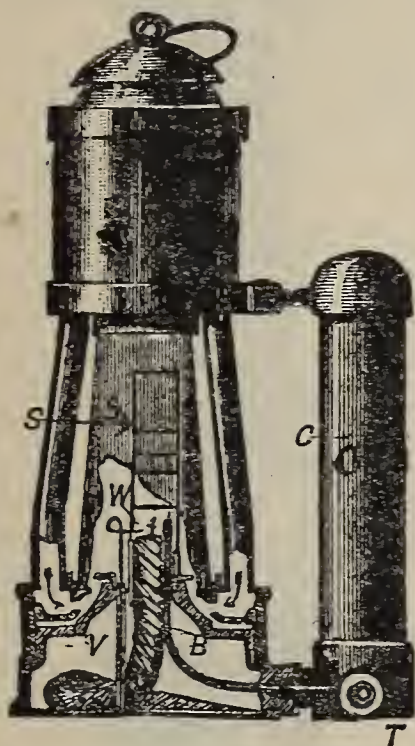


FIG. 641.—Hydrogen Indicator.

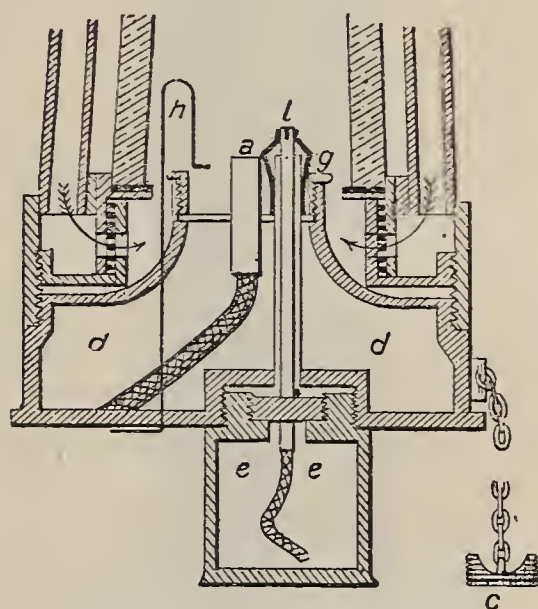


FIG. 642.—Stokes' Indicator.

nected with the interior of the safety lamp, the top of the tube being just about on a level with the burning wick W. When a test is to be made, hydrogen is admitted through the small tube B, by opening a tap T, with a key. The flame of the ordinary wick immediately lights the hydrogen, the jet of which can be regulated to any required size of flame. The ordinary wick is then drawn down by means of the pricker, and the large flame being extinguished, the indicator is ready to make a test with the hydrogen flame alone. A small ladder-like scale S is fixed inside beside the flame, to measure the percentage of gas found, each of the steps on the scale being a definite value. The lamp is guaranteed to measure as little as 0.25 per cent. of fire-damp in air. The small hydrogen cylinder can be detached and carried in the pocket when not required for testing, and the safety lamp can then be used in the ordinary way.

Stokes' Indicator.—In this lamp 'absolute alcohol' is employed

to produce a flame for detecting the presence of small quantities of fire-damp. The indicator may be used with any ordinary safety lamp, and is a very simple arrangement, shown in fig. 642.

A small vessel *ee*, having a thin tube and wick, screws into an opening at the bottom of the safety lamp, the top of this tube reaching to the top of the wick *a*, where it can be lighted; a slit *i*, at the top of the alcohol tube gives the standard flame for testing. When a test is about to be made, the brass plug *c*, which fits the opening



FIG. 643.—Pieler Lamp.

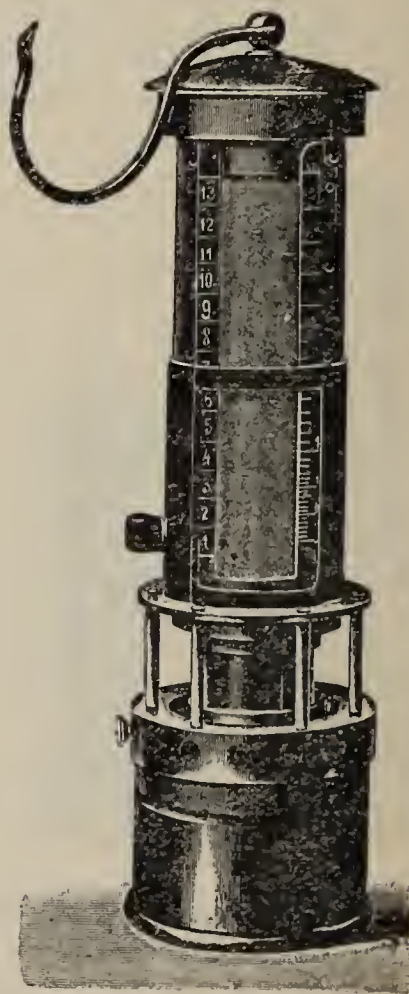


FIG. 644.—Chesneau Lamp.

at the bottom of the lamp, is unscrewed, and the tube of the indicator inserted. In a few seconds the heat will cause the alcohol to ascend and ignite at the oil flame. The oil wick is then drawn down by the pricker *h*, and the test proceeded with. After it is completed the oil wick can be raised and re-lighted at the alcohol flame. The alcohol vessel is then unscrewed, the plug *c* put in, and the lamp again becomes an ordinary safety lamp. It is said to be capable of indicating as little as $\frac{1}{2}$ per cent. of CH_4 in air.

Pieler Lamp.—The Pieler indicator may be described as a large Davy lamp constructed to burn pure alcohol with a special wick. The air which supplies the flame is admitted by a tube, protected by superposed discs of gauze, which pass vertically through the vessel

containing the alcohol. Immediately above the burner and surrounding the flame, is a short, conical chimney, open at both ends, and before making tests, the flame should be adjusted in pure air, so that it comes exactly to the top of the chimney. When burning in a mixture of fire-damp and air the flame shows a much more conspicuous cap or 'halo' than can be produced by the flame of an ordinary safety lamp burning mineral or vegetable oil. A scale is fixed in front of the lamp for measuring the different percentages according to the height of the flame. The apparatus is said to be capable of detecting the presence of $\frac{1}{2}$ per cent. of fire-damp in air. While there is no doubt that it is a most sensitive gas-detector, it is in its present form practically useless for the ordinary usage to which safety lamps are subjected when testing for gas underground. It is too sensitive, too easily extinguished, and requires to be very carefully handled, as the vapour given off from the burning liquid is itself highly explosive when mixed with air. The lamp as now constructed is fitted with a shield and is much safer, in currents of air, than the original type.

Chesneau Lamp.—This apparatus is also constructed to burn alcohol, and is somewhat like the Pieler indicator. It is constructed of a brass reservoir for the alcohol, surmounted by a circular crown for the admission of air (which can be regulated) through double gauzes. Resting on the crown and surrounding the wick-tube is a solid cylinder of sheet-iron, which serves as a screen. Above this screen and resting on it, is an iron wire gauze $5\frac{1}{2}$ in. high. The gauze is surrounded by a sheet-iron shield furnished with an observation window of mica, on which is engraved a scale for measuring the percentages of fire-damp. The shield is fitted at its base with an annular diaphragm, which closely surrounds the base of the gauze, so that the exterior does not impinge directly on the gauze.

The interior of the reservoir contains a small piece of cotton wool under the wick-tube, to prevent the rapid escape of alcohol if the lamp is overturned, while if the apparatus is laid horizontally it is at once extinguished. It is said that caps can be observed when as little as 0.1 to 0.2 per cent. of fire-damp is present in the air, and that the cap becomes quite marked when the percentages reach 0.5.

Like the 'Pieler' lamp, it is best suited for making observations in a still atmosphere, which is not the ordinary condition of underground workings. In the 'Chesneau' lamp, dew forms on the sheet of mica, by the condensation of the aqueous vapour resulting from the condensation of the alcohol, aided by the cold external air impinging against it, and this prevents the observer from making accurate observations.

Various other devices have been invented for the detection of small percentages of fire-damp in the air. One of the first of these devices is that known as the Cunynhame-Cadman. This device consists of a piece of asbestos sheeting steeped in a strong solution

of carbonate of soda. This asbestos sheet is secured in a suitable holder in such a manner that it may be moved into the lamp flame whilst the flame is still comparatively large. When the asbestos sheet is moved into the flame a slight fuzzy yellow or orange halo appears round the flame towards the upper part of the asbestos, this halo being from the oil. If fire-damp is present a yellowish conical cap will appear above the halo, the length and distinctness of the cap varying according to the percentage of fire-damp present. It is said that $\frac{1}{2}$ per cent. of fire-damp can be detected with this device.

Another fire-damp detecting device is that known as the Briggs' loop. In this appliance a small copper or silver loop is fitted in the bottom of the lamp, so fixed that it can be easily turned into the flame when a test is about to be made. When the loop is turned into the flame the temperature of the flame is reduced, and it is claimed that by this means small percentages of fire-damp can be detected much more readily than with an ordinary safety lamp. In the hands of an expert experimenter these devices may detect small percentages of gas, but for the ordinary workman they are not very satisfactory, and they have not had any extended application.

Electric Lamps.—With the advent of electricity for illuminating purposes, much was hoped for in the way of lighting underground workings, but so far, with the exception of its use for lighting pit-bottoms and main roads, comparatively little progress has been made. A number of electric lamps have within recent years been introduced, and in several collieries in England and Scotland they have been adopted for the miners working at the coal face. With the more stringent regulations of the Coal Mines Act, an extension of this method of illumination is likely to take place in the near future. The majority of the electric lamps which have been introduced have been too large and unwieldy, burned for too short periods, were liable to go suddenly wrong, and were too expensive for colliery work. An even greater bar to their usefulness was that they were nearly all constructed on the 'wet battery' principle, which is not suitable for underground work, where lamps are often pretty roughly handled.

Another serious fault against these electric lamps is that they are unable to show the presence of fire-damp in the air of the mine.

A good portable safety lamp is that known as the 'Gray-Sussman' patent electric lamp. The improved lamp is made in three sizes, giving 1, $1\frac{1}{2}$, and 2 candle power, and weighing 3, $3\frac{3}{4}$, and $4\frac{1}{4}$ lbs. respectively; each size has a burning capacity of from 11 to 12 hours. The bulb, which is a 2-volt metal filament, is mounted on a base plate, between two conical reflector domes, the lower dome containing a switch which, when fixed, is quite gas-tight. The domes are surrounded by a strong glass cylinder, having a thickness of about $\frac{1}{4}$ in., the whole being secured to the base plate by means of

an aluminium top plate and four standards. The switch is operated from the outside by means of a milled-head screw on the right-hand side of the lamp. The base plate is hinged to the battery case by a strong brass butt, and is sealed by a hasp and staple with the usual lead rivet. The battery case is made of either strong tinned steel or cast aluminium.

On the under side of the lid two spring contacts are fitted, which engage with two corresponding lugs or battery terminals when the lid is closed. The battery (which is of the semi-dry type) is square,

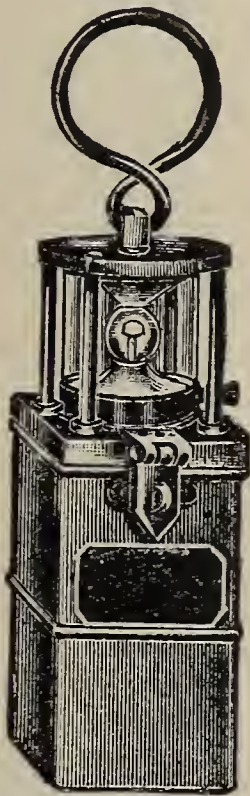


FIG. 645.—Gray-Sussman Electric Lamp.

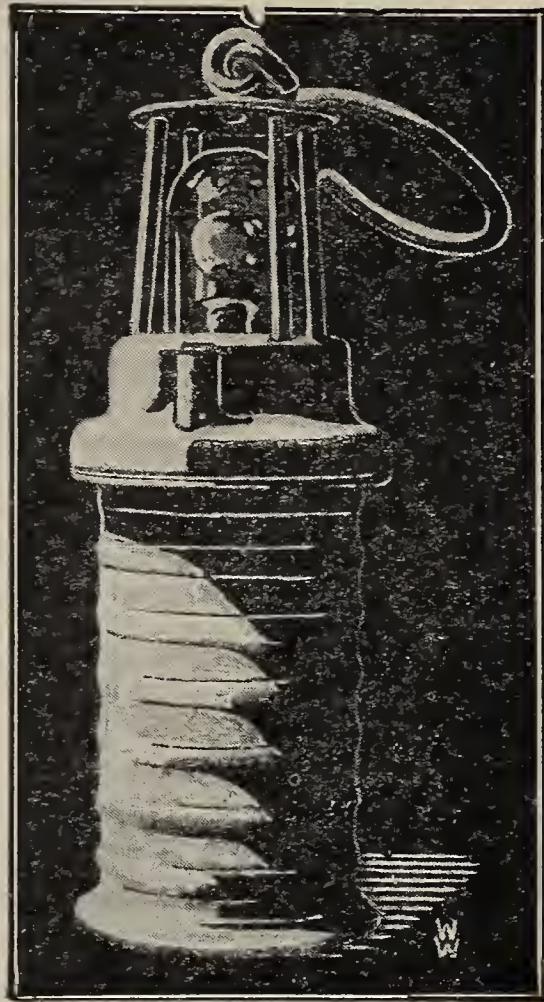


FIG. 646.—'Ceag' Miner's Electric Lamp.

the case being made of stout celluloid. The acid is absorbed by a special composition, so that there is no free liquid to spill.

These lamps are an approved success, upwards of 7000 having been supplied to various collieries in England.

The 'Ceag' Lamp.—This lamp was awarded the first prize in the international competition for the best electric lamp for miners.

The 'Ceag' lamp consists of three parts, viz., the accumulator, the bottom part and the top part, and weighs a little over 4 lbs.

The accumulator consists of a strong celluloid casing and two lead electrodes. The latter are screwed into the lid of the casing, and are of circular shape, by which means warping is avoided, while a considerable capacity of resistance is attained.

The lid is detachable, so that the electrodes can be exchanged easily.

An acid-proof closing arrangement on the battery permits free egress to the gases without allowing the liquid to escape.

On both pole ends are two detachable contact pins on spiral springs which form the contact for the current to the glow lamp. This arrangement prevents the dangerous oxidising of the contacts, as these can be cleaned after using by simply putting them into warm water.

The accumulator is separated by an intermediate space from the steel casing of the lamp, and is thus protected against damage by rough usage or external force.

The bottom part of the lamp is a round stout casing of heavily tinned drawn steel plate strengthened by steel ribs pressed into it. The casing is cylindrical in shape, of somewhat conical form. On the bottom is a small furrow, which keeps the accumulator in its position, but does not interfere with its prompt removal.

The top part of the lamp is fastened with a bayonet joint on to the bottom casing, and can therefore quickly be detached in a most simple manner. The lamp is locked with a magnetic lock, which only can be opened by a strong magnet. The connection between the loose contact and the glow lamp is effected through a brass segment, which is placed in insulating material, rendering it possible to switch the light on and off by turning the top part while the lamp is locked. The glow lamp is a 1·5 candle power metal filament lamp, and it burns after the charging—which only takes from 4 to 5 hours—continually during 16 hours. The bulb is protected by a thick glass dome and four iron bars, and is placed between spiral springs. By this arrangement, not only is economy of bulb got, but also safety against fire-damp, as the circuit is broken automatically when the dome glass is shattered.

Careful calculations made during several years at a number of mines which are equipped with 'Ceag' lamp cabins, show that the upkeep of these is not higher than that of oil lamps. In pits where new modern 'Ceag' lamp cabins have been erected according to a new system, distinct benefits have accrued from the consequent economy of labour and essential simplicity of the upkeep. The 'Ceag' lamp has been in actual practical use for a number of years, and it may be mentioned that there is now a modern 'Ceag' lamp cabin for 3000 lamps at the Bullcroft Main Colliery near Doncaster.

The 'Fors' Miner's Electric Lamp.—This electric lamp is one of those awarded £50 by the Home Office judges in the recent competition. The lamp is of a simple and strong construction, the battery case being made of a hard aluminium alloy. The lantern is hinged to this case, and forms the cover on top of the battery; the cage of the lantern is furnished with steel pillars and glass cylinders $\frac{1}{4}$ in. thickness, such as is used in ordinary miners' safety lamps. A

single-cell 'Fors' patent accumulator, giving 2 volts 0·9 ampères and 10 ampère hours, supplies the electric energy, and is put in or taken out in a few seconds. A projection on the side of the cell and a corresponding slot in the top of the battery case enables this to be done, so that the cell registers in its correct position for making the electrical contacts. There are no wire connections to undo, both the lamp and the switch contacts being made with specially designed patent taper pin and spring clip sockets, and the pins readily find

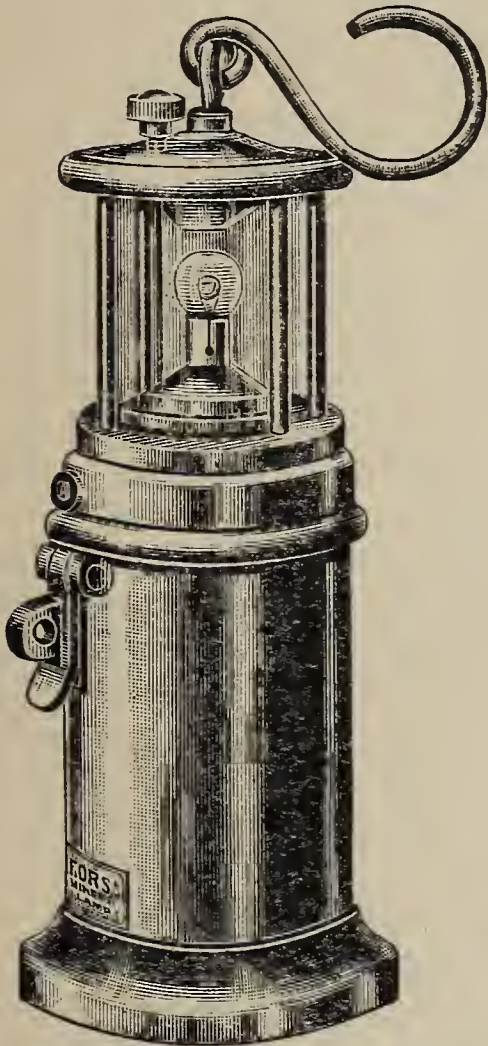


FIG. 647.—'Fors' Miner's Electric Lamp.

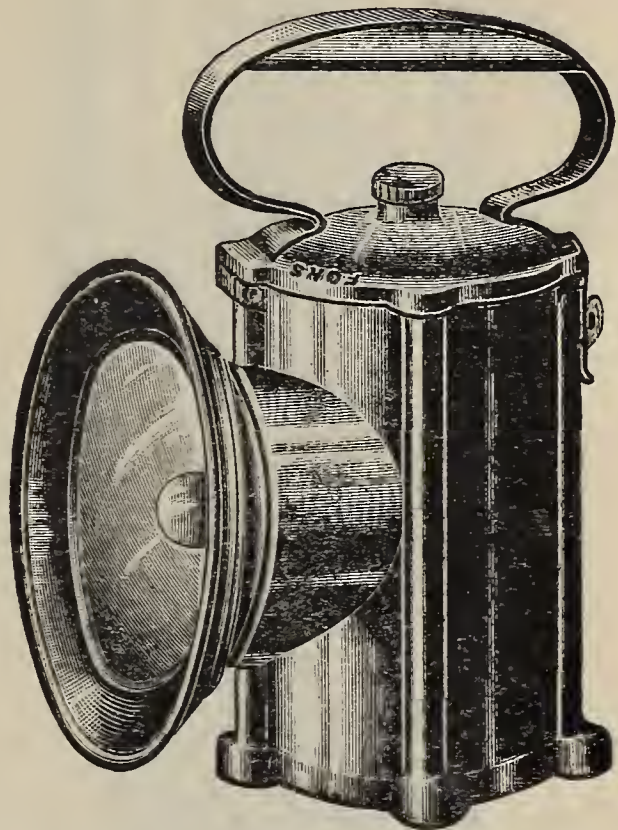


FIG. 648.—'Fors' Portable Electric Hand Lamp.

their position in the socket and are self-cleaning, whilst the sockets are detachable.

In the 'Fors' accumulator the positive electrode is cylindrical in shape, and the negative electrode concentric surrounding the positive, with a calcined asbestos porous pot separator between; the electrodes cannot buckle or short-circuit, and, as there is no local action, they do not sulphate. An important feature of this battery is that it retains its charge almost indefinitely when not in use, and as the cells are all made to a standard size, a fully charged cell can be put quickly into the lamp to replace a used-up cell.

The 'Fors' lamp weighs 3 lbs. 8 ozs., the same weight as an

ordinary miner's oil safety lamp, the light is 2 candle power effective reflected light for 10 hours. The light can be turned on or off by the milled-head button switch shown on the top of the lantern. All contacts are made inside the lamp case, and are hermetically sealed

from the outer air, so there is no possibility of communicating the spark on making the contacts.

The cells are all fitted with a patent unspillable vent plug, and the lamp can be turned upside down without spilling the solution; this enables the terminals and contacts to be kept quite clean.

A special feature of the spring clip sockets and taper pin contacts is that a number of cells can be quickly connected up for re-charging by simply inserting contact pins into the sockets of the cells placed in a row in position under them; this saves much time and trouble.

The 'Fors' portable electric hand lamp, type 403 H, shown in fig. 648, is a very handy lamp for inspectors, and gives 2 candle power effective reflected light for 10 hours, and weighs about $3\frac{1}{2}$ lbs. The 'Fors' battery used in this lamp is the same as used in the other lamp.

Some material improvements in the lamp have been introduced. In the new lamp the hinges connecting the lantern to the battery case are replaced by a form of bayonet and slot joint, which has the advantage that rotation of the members, after the removal of the plug seal, automatically disconnects the circuit and extinguishes the lamp. Means have also been provided for removing the glass chimney and bulb for the pur-



FIG. 649.—The 'Stach' Miner's Electric Lamp.

poses of cleaning or examination more rapidly than in the type shown, without the necessity of removing nuts, as in the earlier lamp, whilst in several other particulars the construction of the lamp has been improved.

*The 'Stach' Miner's Electric Lamp.**—The 'Stach' mining lamp,

* *The Colliery Guardian*, 30th August 1912.

which was awarded a prize by the Home Office, is of very strong construction and well adapted to withstand the rough usage necessarily obtaining in mining work.

As will be seen from the illustration, it consists externally of a wrought-iron welded case, carried by an iron hoop, so arranged that the lamp is capable of being used in any position. To the top of this hoop is attached a spiked iron hook, for the purpose of suspending the lamp during work. On the top of the iron case there is

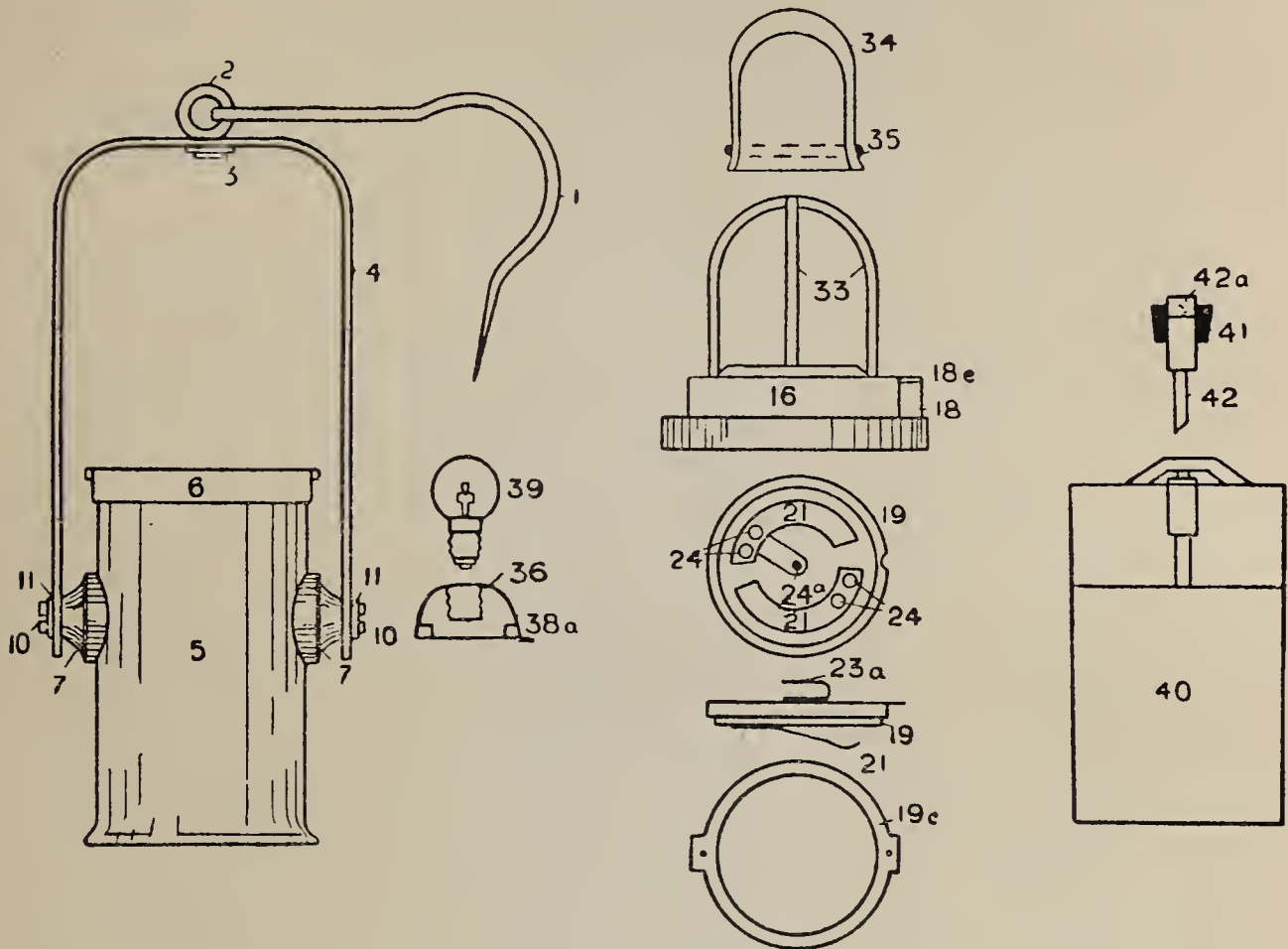


FIG. 650.—Details of Lamp.

- 1, carrying hook; 2, eye with rivet; 3, washer for same; 4, hoop; 5, pot; 6, fitting ring; 7, side springs; 10, side screws; 11, washers for same; 16, upper ring; 18, magnetic lock; 18e, covering plate for magnetic lock; 19, drilled ebonite disc; 19c, bayonet ring; 21, contact springs; 23a, contact springs for glowlamp; 24, screws for contact springs; 24a, lamp contact screw; 33, bent brass rods for guard; 34, patent bell glass; 35, round rubber fitting ring for same; 36, lampholder; 38a, nickel reflector; 39, patent glowlamp; 40, accumulator; 41, rubber plug; 42, gas outlet cartridge.

fitted a brass ring rendered water-tight by means of a rubber seating. The ring and the case are arranged to fit together, and are locked by means of an iron pin worked by a spring. This spring can only be withdrawn by means of a powerful magnet. On the upper part of the lamp is a stout brass wire guard, protecting the glass globe and the glow lamp. The glass globe is of considerable thickness, and is set upon a rubber ring, which also renders this part of the lamp water-tight and gas-tight.

The metal filament lamp is of the Aegma type, 2 candle power,

2 volt, and is of the screw pattern. It has no tip, but is sealed off so that the bulb is quite smooth. Between the leading-in wires there is placed a little wall of glass, so that even if the lamp is destroyed by breakage of the bulb and filament, the leading-in wires cannot be brought together so as to cause a short circuit and a dangerous spark.

The method of lighting the lamp is to turn the upper part in a clockwise direction relatively to the lower part. This causes the contact springs to slide over the terminals of the accumulator, thus ensuring a good rubbing contact with a wiping action. Switching off is performed by reversing this operation. There is no perceptible spark at opening or closing the circuit, and lengthy experiments have been carried out which have proved that the ignition of inflammable gas, either inside or outside the lamp, is impossible.

The cell consists of a 2-volt, round, semi-dry, lead accumulator, in a celluloid case. It is so constructed that it not only always fits down into the proper position in the iron case, but it cannot be put on to the charging board in the reverse direction. This is effected by means of a celluloid projection fixed eccentrically on the bottom of the cell, which also prevents any rotary motion in the battery case, as a corresponding recess is formed in the latter. The electrodes are wrapped in glass wool, which absorbs some two-thirds of the sulphuric acid used as electrolyte. The plates are so prepared that they are able to stand rapid discharge, and they can also be left for many months in a discharged condition without being damaged. In consequence, they may be charged and discharged over 200 times before their useful life is exhausted. The special stopper is provided to prevent the escape of acid, and to render harmless any gas evolved during discharge. This stopper is so arranged that it is impossible to spill the acid, even if the cell is inverted, and should any very small drops of acid come through with the gas, they are taken up by a small and easily replaced piece of absorbent material. The terminals consist of substantial lead blocks properly marked 'positive' and 'negative,' and the celluloid case is hermetically sealed, with the exception of the special stopper above described.

The lamp used gives 2 candle power, and one charge will last for 14 hours. A concave nickel or aluminium reflector can be provided if desired, so as to project the light in any direction, or to shield the eyes during use.

*The 'Bohres' Lamp.**—This lamp is made in two types, the one to throw the light upwards, as shown in fig. 651, the other to throw it downwards. The lamp case is made of tinned sheet steel, and is rectangular in section with a round foot and top, being painted with an acid and rust-resisting composition. A magnetic or screw lock is provided at the locking ring. The incandescent bulb is protected by a strong air-tight glass globe. Contact is formed by turning the

* *Iron and Coal Trades Review.*

upper part of the lamp above the lower, whereby two broad spring plates are slid over the contact bridges of the accumulator.

The single-cell accumulator, enclosed in a celluloid case, fits loosely into the lamp body, and can be readily removed and replaced. The leads to the contact pieces are bedded in and are acid proof. The filling opening is fitted with a three-chambered screw-closing device, allowing any gas developed to escape, but preventing any acid from

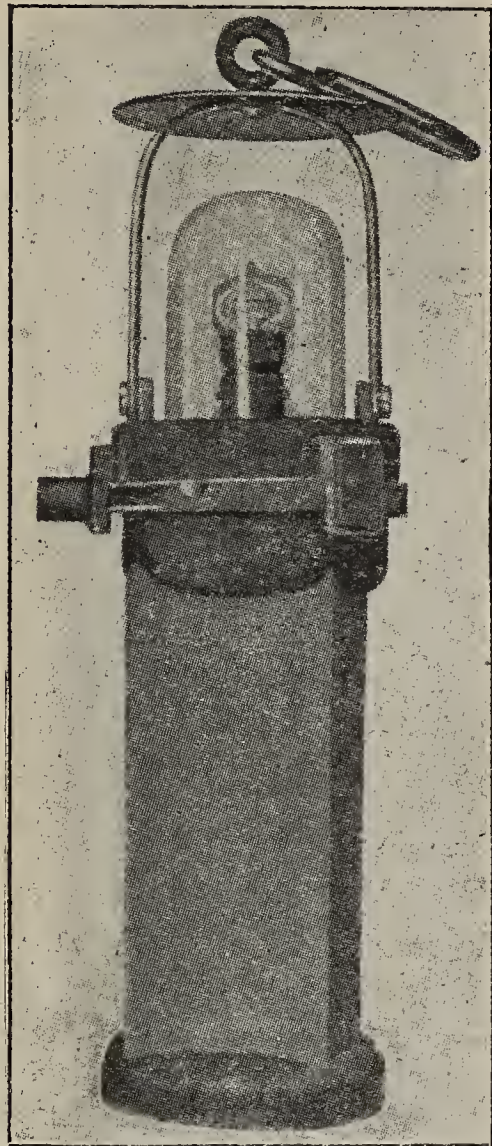


FIG. 651.—'Bohres' Electric Safety Lamp.

flowing out when the lamp is held in a slanting position. The accumulator feeds the glow lamp, giving $1\frac{1}{2}$ to 2 candle power for 12 hours. The accumulator is connected up to a charging device by plugs. The weight of the lamp ready for use is 5 lbs., and the height is $10\frac{1}{2}$ in.

The 'Wolf' Alkaline Lamp.—This lamp, which has been recently introduced, is made in two sizes. The smaller size gives $1\frac{1}{2}$ candle power for 16 to 18 hours, or 2 candle power for 12 to 14 hours. The larger size gives 2 candle power for 20 to 22 hours, or 4 candle power for 10 to 12 hours. It is provided with an alkaline accumulator, which gives a higher illuminating power and longer duration of

light than a lead cell. The latter type of cell has also the disadvantages of having a very limited life, frequently requires repairs, and needs very careful attention. It has the further disadvantage that it requires about twice the weight and proportionately larger

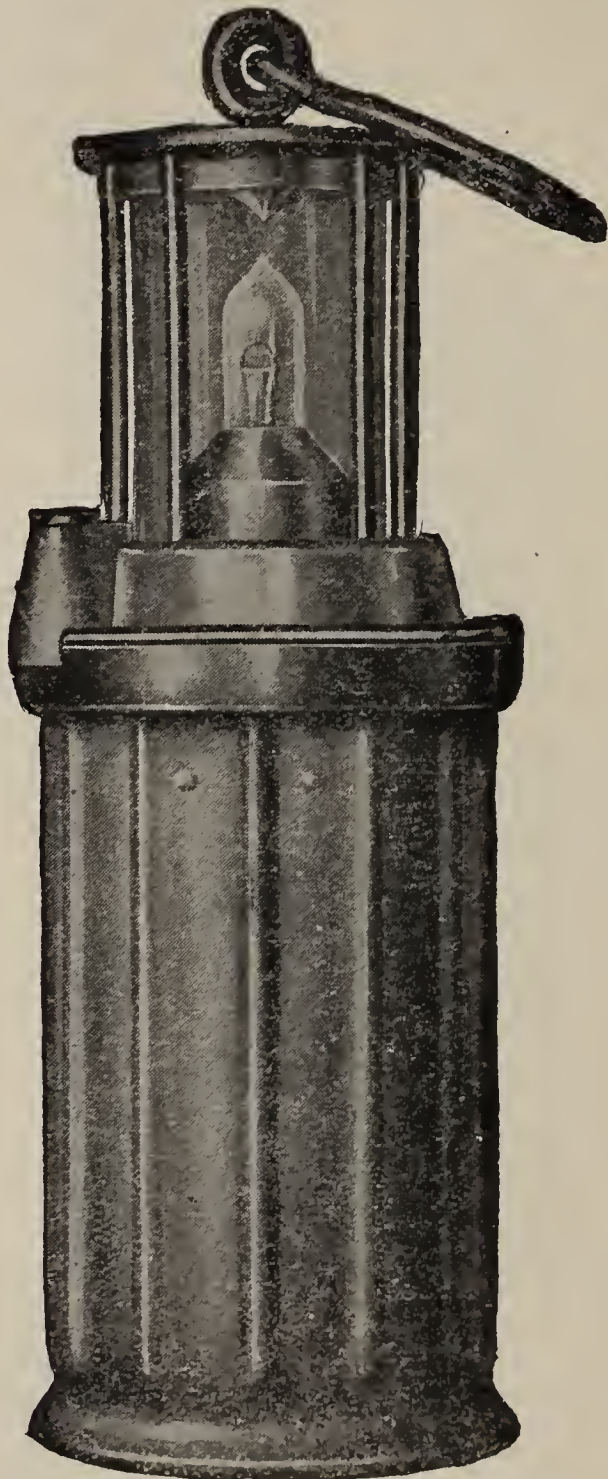


FIG. 652.—'Wolf' Alkaline Safety Lamp.

dimensions to give the same candle power as a lamp fitted with an alkaline accumulator. The shell or containing case of the 'Wolf' alkaline lamp is made of pressed steel, and is reinforced by a number of vertical ribs. A celluloid anti-concussion ring, fitted with rubber studs, is placed in the base of the shell to prevent the transmission of mechanical shocks to the battery and working parts. Sufficient

clearance is provided between the battery and the shell, in order to obviate the difficulty of removing cells, and to prevent damage to

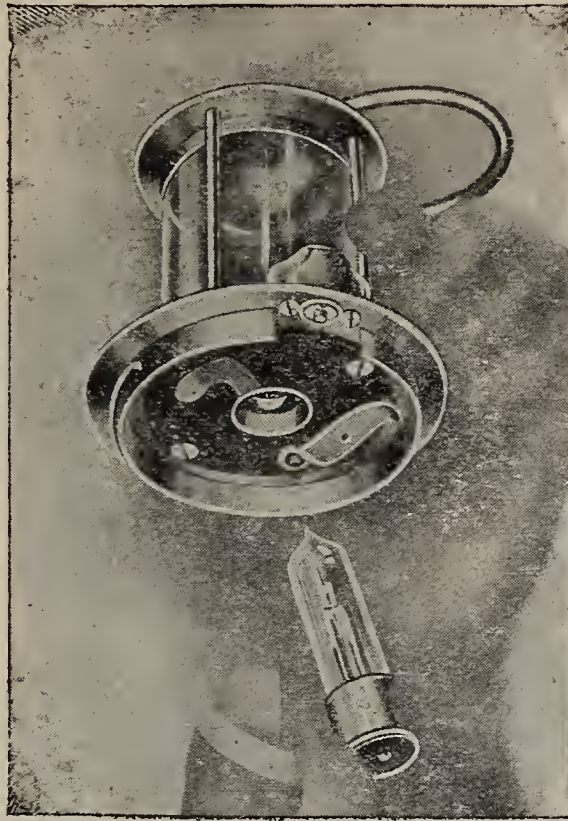


FIG. 653.—Inserting Bulb in a 'Wolf' Lamp.

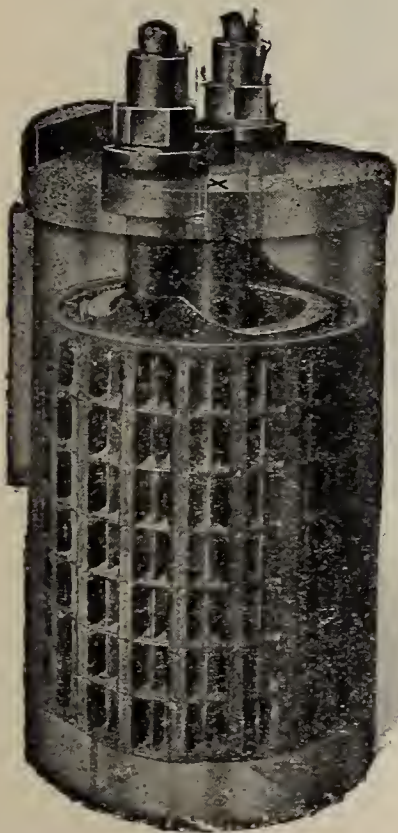


FIG. 654.—'Wolf' Lead Cell.

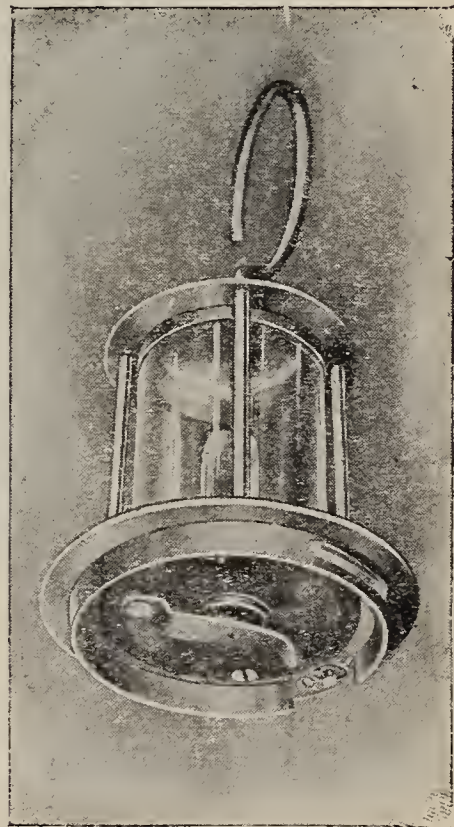


FIG. 655.—Lamp with Bulb Inserted.

them, if the shell is struck by a tool or otherwise damaged in the pit. The lamp is magnetically locked, the lock being easily replaceable. A special device is provided for ensuring safety when switching on or off. The device consists of a double contact between each pole, with the space between the same bridged across by a small



FIG. 656.—'Wolf' Rescue Lamp.

high resistance coil which is embedded in the insulating base. This forms a shunt across the contacts resistance which prevents sparking on rupture of the circuit, and it also takes the first rush of current from the cell on making contact, and thus lengthens the life of the lamp. A lamp to give 2 candle power for 10 to 12 hours weighs $4\frac{1}{2}$ lbs., and is 10 in. in height.

Hailwood's Electric Miner's Lamp.—This lamp primarily consists of two main portions of cylindrical form. The bottom cup-like portion contains a circular accumulator having flat plates, the paste of which has been inserted under great pressure. The lamp bottom is provided with a solid cast ledge R (fig. 657) having two slots L G formed therein.

The upper part of the lamp is passed down and over the bottom, lock catches L C on the upper part entering the slots L G, a slight rotation (bayonet joint-wise) to the left causes these catches to pass

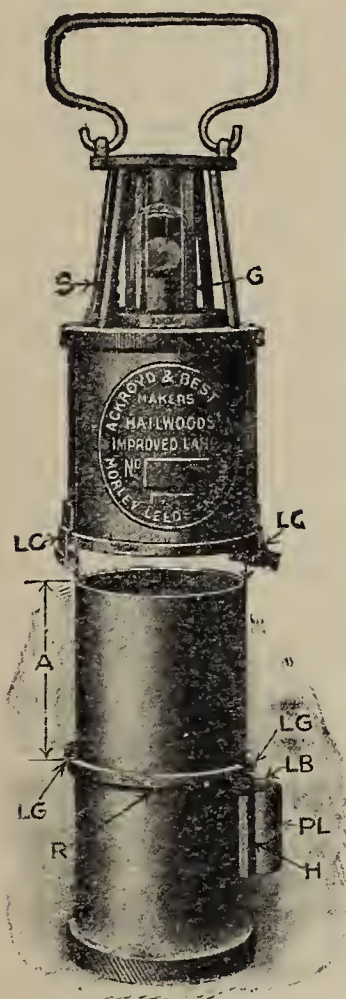


FIG. 657.

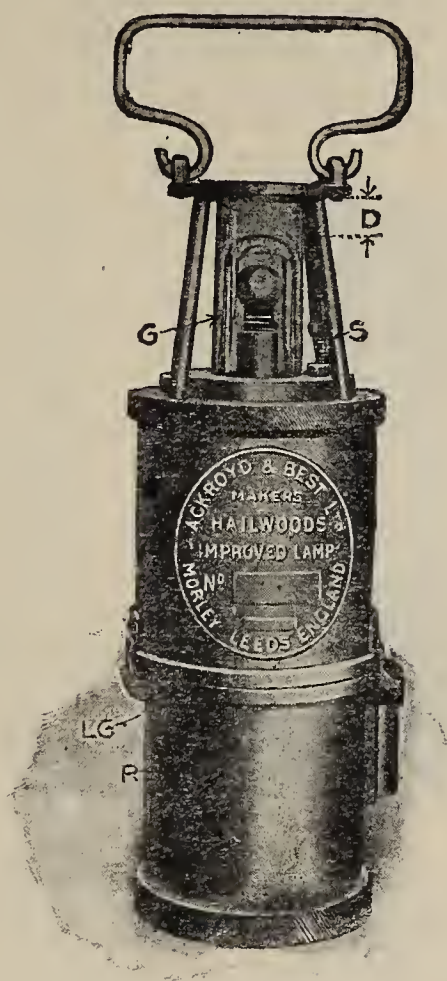


FIG. 658.

under solid rim R, the lock bolt L B engaging on teeth on the underside of one of the catches L C, suitable guards being provided on the plate L C to screen the lock and prevent same being tampered with. The catches L C are formed so as to anchor behind another solid rim on the lamp top, so that there is no fear of the top and bottom severing and the formation of a spark by the breaking of the electric contact, as may so easily happen with lamps which have hinged or similar joints.

The upper part of the lamp overlaps the bottom part at the part marked A (fig. 657) to the extent of $2\frac{5}{8}$ in., and a clearance of $\frac{1}{32}$ in. is left between the walls of the upper and lower case, which, while allowing acid fumes to ventilate and get clear of the lamp, *will not*

allow flame to pass out alive, as the flame in passing over $2\frac{5}{8}$ in. of cold metal has the heat abstracted from it and cannot possibly ignite gas on the outside of the lamp.

The electric bulb is contained in a glass G, which is of small outer diameter and has very thick walls, and has the formation of a top $\frac{1}{2}$ in. thick at the part D (fig. 658), the top of the glass being flat, forming a strong bed against the crown plate of the lamp, an aperture being formed in the crown plate to allow rays of light to pass through the thick upper end of the glass.

The accumulator is fitted with a special arrangement of vents, which, whilst permitting the escape of the fumes from the interior of the accumulator, will not allow sulphuric-acid to run out.



FIG. 659.—Section showing Hailwood's Patent Magnetic Lock.

The lock P L is of the pneumatic type, and formed of brass and aluminium, to resist corrosion from the sulphuric acid or acid fumes. The pneumatic lock can be arranged to open with an air pressure of from 100 lbs. to 250 lbs. per square inch. The compressed air enters the lock through an aperture H, passes along a tortuous channel, and depresses the lock bolt L B, freeing it from the lock catch L C, when the top may be rotated until the catches L C are in line with slots L G, and the top then lifted off.

After the lamp has been closed, the light may be turned on by the screwing down of the screwed switch S, the lower part of the switch pressing on to an accumulator contact ring.

Hailwood's Improved Safety Lamp.—

Fig. 659 illustrates this lamp, in which part of the vessel and bush have been cut away so as to show the interior of the Hailwood patent lock. The glass-holding ring *a* is provided on the underside with ratchet teeth into which engages an iron lock bolt *b*, which is held in the up position by means of a strong spring *d* resting on a slidable iron guard plug *e*, the latter resting on a solid shoulder formed in a recess in the lamp bottom. This guard plug effectively prevents any tampering with the lock. Anyone pressing the guard upwards compresses the spring *d*, and causes it to hold up the lock bolt *b* even more effectively, making it impossible to tamper with the lock by means of a permanent magnet. The depression of the electro-magnet pedal drawing down the guard plug, and with it lock bolt *b*, thereby releasing same from the teeth of the glass ring, when the lamp top may be unscrewed from the oil vessel. To lock the lamp, the act of screwing on the vessel will cause the lock bolt *b* to click over the ratchet teeth of the glass ring

until the oil vessel has been screwed home, when such vessel cannot be again removed until opened by the electro-magnet.

Construction of Safety Lamps.—In all the improvements and modifications that have been made in safety lamps of late years, the tendency has been in one direction, viz., to render them as safe as possible in currents of air travelling at high velocities. While this object is one to be commended, a great deal more than this is required of a good safety lamp, and if some of the inventors would turn their attention to combining safety with good illuminating power and simplicity of construction, they would render good service to those who are compelled to use such lamps. A very large number of safety lamps, which give a good light when on the surface, or in the main airways underground, are of little or no use under the ordinary conditions met with in mines, either owing to the small amount of light which they give, or the sensitiveness which shows itself by the ease with which the light becomes extinguished. Until a safety lamp that will give a light equal to at least one candle power can be placed in the miner's hands, it cannot be said that the limit of improvement has been reached.

Lighting Power of Lamps.—The lighting power of lamps is a very variable quantity, and differs very much under different conditions. To obtain good results there must be a good burning oil, a fairly large wick and burner, and a glass so made that it will diffuse the light in every direction. With safety lamps, under ordinary conditions, it takes about three Mueseler's or Marsaut's to yield one candle power.

Some tests were made some years ago with various safety lamps at the Hamilton Gasworks, and the following results were obtained :—

Name of Lamp.	Number of Lamps equal to one Standard Candle.	Oil Burned.
Naked light	0·76	Paraffin wax.
„	0·60	Sweet oil.
Davy lamp	6·17	„
Mueseler (Belgian)	2·51	„
„ (protector)	2·02	Naphtha spirits.
Williamson lamp	1·54	Sweet oil.

Mr T. A. Saint, B.Sc.,* has made a number of interesting tests on the lighting efficiency of safety lamps. The tests were made with a 'Flicker' photometer and a standard candle. Five different blends of oils were used, for the various tests, of the following composition :—

* *Trans. Inst. Min. Engs.*, vol. xlv. p. 327.

COMPOSITION.

	Sp. Gr.	Mineral.	Vegetable.
		Per cent.	Per cent.
1	0·820	100	0
2	0·841	70	30
3	0·844	50	50
4	0·865	30	70
5	0·889	0	100

The following are the results of some of these tests :—

TESTS WITH NO. 1 OIL.

Type of Lamp.	Size of Wick.	Average Candle Power per Ten Hours.	Cost per 1000 Lamps per Candle Power per Hour
			s. d.
Marsaut . . .	$\frac{3}{8}$ -inch flat	0·734	0 11½
Small Donald . . .	„	0·455	1 10
Mueseler . . .	$\frac{3}{8}$ -inch round	0·480	1 2½
Davy . . .	„	0·490	1 6
Stephenson . . .	„	0·415	1 4
Clanny . . .	„	0·751	0 11½
Scottish Davy . . .	$\frac{1}{2}$ -inch flat	0·527	1 5½
Deflector . . .	flat	0·950	0 11½
Routledge and Johnson .	$\frac{3}{8}$ -inch flat	0·467	1 7½

TESTS WITH NO. 3 OIL.

Type of Lamp.	Size of Wick.	Average Candle Power per Ten Hours.	Cost per 1000 Lamps per Candle Power per Hour.
			s. d.
Marsaut . . .	$\frac{3}{8}$ -inch flat	0·702	2 1½
Small Donald . . .	„	0·481	3 7
Davy . . .	$\frac{3}{8}$ -inch round	0·440	3 7½
Clanny . . .	„	0·751	2 2½

TESTS WITH BENZINE, SP. GR. 0·765.

Type of Lamp.	Size of Wick.	Average Candle Power per Ten Hours.	Cost of 1000 Lamps per Candle Power per Hour.	
			s.	d.
Wolf	$\frac{1}{2}$ -inch flat	1·018	1	4 $\frac{1}{2}$
Protector	$\frac{3}{8}$ -inch round	0·695	2	1 $\frac{1}{2}$

No. 1 oil is very mobile, and is not very suitable for burning in safety lamps, as the lamp after burning for an hour or so becomes very hot. No. 3 oil is a mixture of half mineral and half vegetable, and this tends to greatly decrease the heat.

These tests are only of value as a comparison between different lamps when new and clean and tested in a room on the surface. In the actual working conditions underground none of the lamps tested will give anything near the above lighting power.

Dr T. Lister Llewellyn,* who has made a large number of tests with safety lamps, gives the following results :—

Type of Lamp.	Candle Power.
Cambrian fireman's (new type)	0·6
Ackroyd and Best	0·4
Patterson	0·42
Wolf	0·8
Pope electric	0·6
Float „	0·9

Dr Llewellyn states that “the oil safety lamp at the coal face rarely gives more than one-third of a candle power when clean, and much less when dirty.”

The following table is given by Professor Lupton,† showing the light of different safety lamps compared with that of a standard candle :

Name of Safety Lamp.	Number of lamps required to equal one standard sperm candle, the illuminating object being on a level with the flame.	Number of lamps required to equal one sperm candle, allowing for the shade cast by lamp cover, bottom, and pillar, the candle and the lamp being each in a cylinder of white tracing paper 2 ft. high and 8 in. diameter.
Davy lamp, good flame	13 (average)	26 (average)
Protector shielded Marsaut . .	3 „	...
„ „ „	„	...
flame average height „ . . .	4 $\frac{1}{4}$ „	...
Deflector lamp, large flat flame	3 „	...
„ „ moderate clear	5 „	...
Clifford lamp, very good flame	1 $\frac{9}{10}$ „	...
„ „ moderate flame	2 $\frac{1}{2}$ „	5 (average)
Ashworth lamp	3 $\frac{1}{2}$ „	11 $\frac{1}{2}$ „
Stephenson lamp, good flame	7 $\frac{1}{2}$ „	23 „
Tallow candle, good average flame	1 $\frac{3}{4}$ „	...

* *Ibid.*, vol. xlv. p. 274. † Lupton's *Mining*, p. 288.

The light of safety lamps will depend a good deal on the sort of oil used and the state of the ventilation ; light oils, such as petroleum and colzaline, giving a much better and clearer light than heavy oils, such as rape or seal oil.

If light oils are used, they require to be very carefully handled, and no naked lights brought near the oil receiver, as they give off inflammable gases at comparatively low temperatures, and are apt to catch fire very readily and do much damage. The writer has known at least half a dozen cases where lamp-rooms have been set on fire and destroyed through careless handling of these light oils.

The Royal Commission on Accidents in Mines recommended a mixture of vegetable or animal oil with about one-half its volume of a petroleum oil of safe flashing point, as giving the best results as an illuminant for safety lamps. It has been found in practice that a 75 per cent. mixture of mineral-colza oil burns very well, and gives

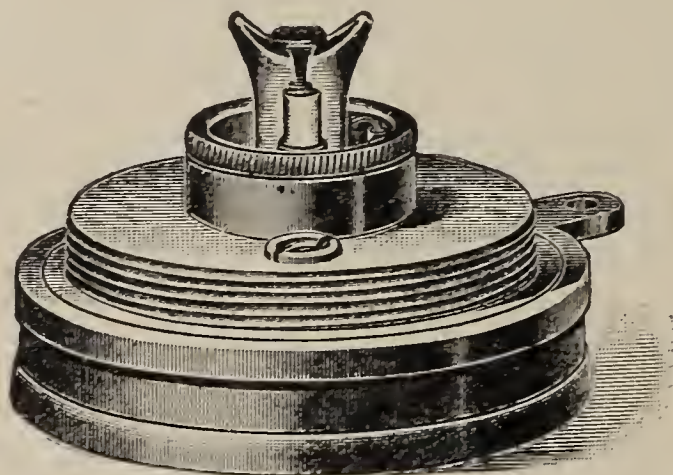


FIG. 660.

a high illuminating efficiency. The greatest objection to heavy vegetable oils is that the wick very frequently requires trimming, a crust forming on the top of the wick after the lamp has burned for an hour or two in the pit. With a light spirit oil this is avoided.

Form of Burner.—The illuminating power of a safety lamp depends not only on the oil used, but also on the form of burner and shape of wick employed. The two most common types of burners are the flat form, as used in the Clanny lamp for burning a heavy oil, and the round shape, as in the Mueseler for burning a light oil.

Some years ago a new type of burner was introduced and has been now largely adopted for safety lamps. It is what is known as the bifold burner, and is usually made of porcelain (see fig. 660). This shape of burner allows the wick to spread at the top, with the result that an increased amount of light is given, sometimes amounting to about one-third greater than the ordinary flat or round burner.

Cost of Upkeep of Lamps.—The average cost of safety lamps—if used with light oils, which are cheaper than the heavy vegetable oils, —including both oil and wick, is about a halfpenny per lamp per

shift. Thus Davy lamps cost about 0·375d. per day for oil alone ; Mueseler lamps about 0·295d. ; while the cost of naked lights may be estimated at 1·25d. per diem.

The following is a detailed statement of the cost of safety lamps per annum, as ascertained at the Clyde Collieries, Hamilton, although the writer thinks that some of the items, such as gauzes and glasses, are underestimated :—

	s.	d.
Cost of oil (300 days) per lamp, ...	7	4½
Repairs, ,, ...	0	4½
Gauzes and glasses (300 days) per lamp, ...	0	2¾
Lampmen's wages, ,, ...	9	3½
Total cost	17	3¼

In a paper read before the Chesterfield and Midland Counties Institute of Mining Engineers, the late Mr A. H. Stokes gives the average cost of safety lamps for four large collieries, each using over 1000 lamps, as 0·736d. per lamp per day, made up as follows : oil 0·162d. ; wick 0·0016d. ; cleaning 0·414d. ; repairs and renewals 0·079d. ; interest on capital 0·08d.

Locking Contrivances.—Nothing conduces more to the safety of a lamp, or of those using it, than an efficient method of locking ; for

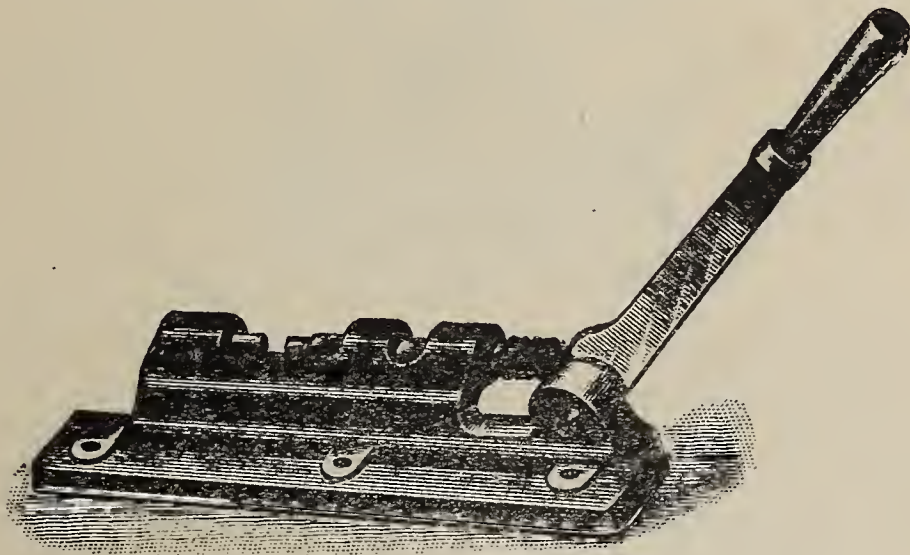


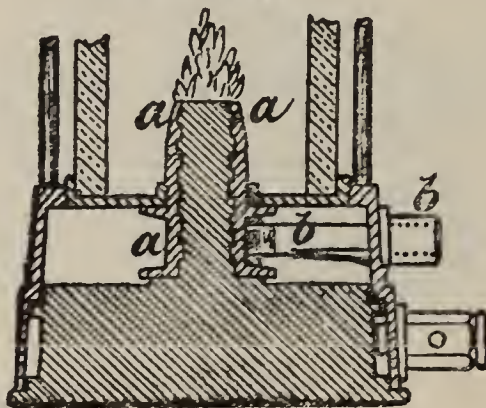
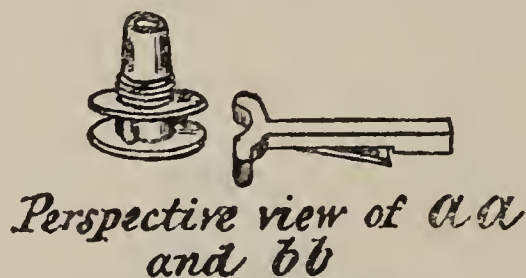
FIG. 661.—Machine for Locking Safety Lamps.

if the cause of many explosions could be accurately ascertained, it would be found that not a few of them were caused by miners themselves surreptitiously opening their lamps ; indeed, in many explosions, this has been clearly ascertained to have caused the disaster. It is a notorious fact that few men are more careless of their own safety than miners, who have been known to open their lamps at the face, even in the neighbourhood of large quantities of fire-damp.

In the old method of locking lamps, a padlock was often used, and could be opened easily by a duplicate or skeleton key. Another

lock which was largely used, and is still used to some extent, was a small screw bolt, either with a square or tapered head, which was turned until the body of the lamp and the bottom were fastened together, by means of the bolt pinching the bottom part. This lock was of little use, as any workman with an old nail filed to the proper size could open his lamp by unscrewing the lock. The simplest and best method of locking a lamp is by using a riveted lead plug, connecting the body of the lamp with the oil-vessel. The lead plug should be firmly riveted, and each end stamped with letters or marks, varied from day to day.

Without this precaution it has been found that the rivets can be removed and replaced in such a way as to render detection extremely



FIGS. 662 and 663.—'Protector' Lock.

difficult. Fig. 661 shows a handy machine for locking safety lamps with lead rivets which is in use at many collieries.

Magnetic locks are also used, in which the lamp can only be opened by the aid of a powerful magnet. Of this class, Wolf's magnetic lock is the simplest and most satisfactory.

'Protector' Lock.—Many lamps are now fitted with the 'Protector' arrangement for securing the oil-vessel to the top part of the lamp. In addition to this arrangement the lamp is also locked in the usual way.

The apparatus will be understood from figs. 662, 663. The wick-tube has a screw thread upon it throughout its whole length, and on this is screwed a 'thimble' *aa*, provided at its lower end with a flange, on the outer end of which a screw-thread is also cut. By the latter screw the oil-cup is attached to the lamp, and when this has been done, the flanged thimble is fastened in position by a bolt *b*, provided with a spring; the bolt cannot, therefore, be withdrawn until the oil-cup is removed. The thimble is screwed on to the wick-tube

before the wick is lighted. The oil-vessel can, of course, be readily removed, but only by making the end of the wick-tube traverse the closely-fitting thimble above mentioned for a distance of 1 to 1.5 in., and during this process the diminished flame of the naphtha spirit is certain to be extinguished. With this arrangement it is practically impossible to open the lamp without extinguishing the flame, unless the lock bolt gets broken or the spring attached gets out of order, which does not readily happen with fair treatment.

As already stated, the oil-vessel is fastened in the ordinary method by riveting with a lead plug, which gives additional security, and also permits the oil-vessel to be partially unscrewed for the purpose of regulating the flame, but prevents it from being altogether withdrawn.

Testing Lamps.—All safety lamps, before being taken into the mine, require to be tested at the lamp station on the surface, or at a station at the bottom of the shaft, to see that they are in a safe condition for using, and also to ascertain that the parts are properly fitted together, particularly the glass and the connecting parts at the top and bottom of it, for it is absolutely essential to safety that no opening should be left at the junction of the glass with the brass rings for an explosive mixture to enter the lamp and ignite at the flame.

The testing of the lamps is often done in a very cursory way at many collieries, owing, no doubt, to the lamp attendant not being aware of the true value of such testing.

The simplest method of testing safety lamps is to take a brass tube of small diameter, somewhat like a blow-pipe, and to blow through it with the mouth so that a current of air impinges against the glass all round the top and bottom edges. If the glass is not properly fitted, it will be detected by the current of air getting in at the edges and deflecting the flame or extinguishing it altogether.

Sometimes the lamps are tested in a more scientific manner by being inserted in an inflammable mixture to see if they are really 'safe,' this method being much better than the blow-pipe test.

* A testing apparatus for safety lamps is shown in fig. 664.

A and B are two cylinders which fit into one another. The cylinder A is filled with water to about three-fourths of its capacity, and the cylinder B, the bottom of which is open, contains a valve at the top of it. By simple pressure of the thumb, the valve can be opened, whereupon cylinder B is made to rise to three-fourths of its height, and become filled with air. At the bottom of the cylinder A is a pipe, which, as it extends above the surface of the water, receives the air that cylinder B forces into the reservoir C, where the gases are generated. C is a reservoir, the inside of which is divided by several partitions, made of corrugated iron, and filled with some absorbent material such as cotton wool. The glass cylinder D is filled with benzine immediately before the apparatus is required for

* The author is indebted to the makers, Messrs Friemann & Wolf, for the diagram and description of this apparatus.

use. If the cock E be opened, the benzine will flow into the reservoir C, in which it is obliged to follow the windings of the sheet iron, and in this manner offers a large surface to the air, the latter being thus impregnated with benzine and transformed into gas.

The gas then passes through the cock F into the testing cylinder G, which is made of tin-plate, and fitted with a pane of glass. Inside this testing cylinder there is a spiral pipe, the inside of which is drilled with small holes which serve for the admission of air. By means of a metal sliding valve the quantity of air entering may be regulated. The lighted lamp is placed within the spiral pipe in the

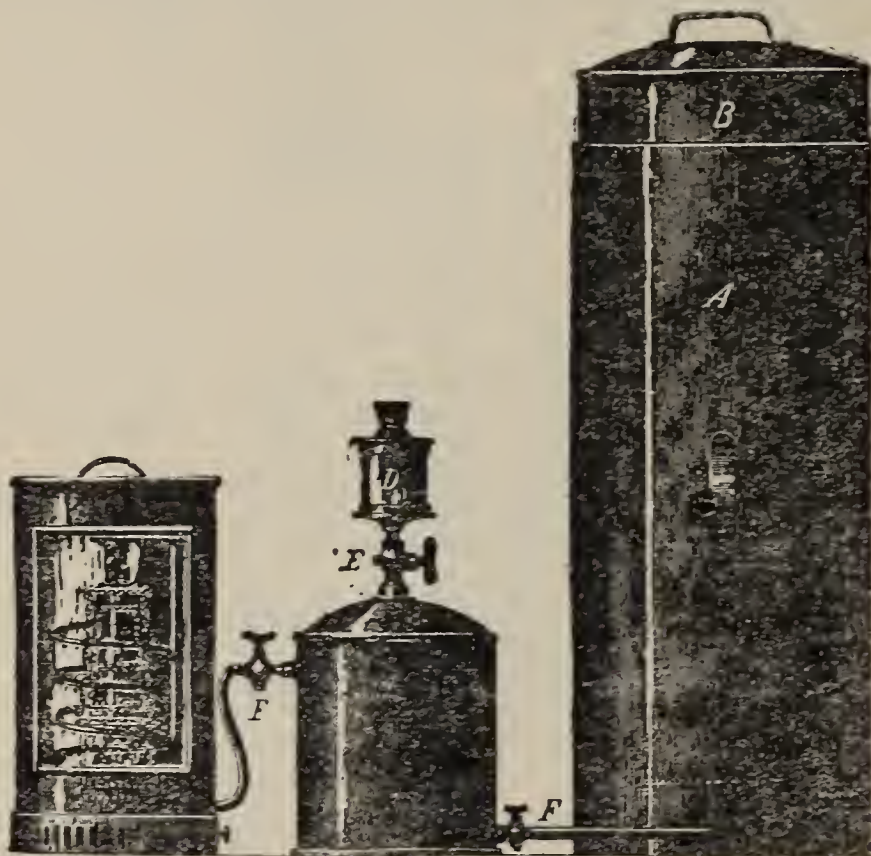


FIG. 664. —Testing Apparatus for Safety Lamps.

cylinder G, and the cock F is opened. The gas issues through the fine holes in the pipe and streams on to the lamp.

If the latter be defective, the generated gas will be ignited in the cylinder G, but the flame of a well-fitted lamp will be extinguished in the presence of too large a quantity of this gas, which will produce the same effect upon the flame as fire-damp does.

To make more elaborate tests of safety lamps in explosive gases, a larger and more costly apparatus may be employed. The apparatus will be understood from fig. 665. It consists of a tank surrounded by water like an ordinary gasholder. Connected to this tank are two pipes C and D, of 2 in. diameter. The pipe C is connected to the mouth of a long wooden box B, about 26 ft. long, 10½ in. high, and 4¾ in. wide, constructed of boards 1½ in. thick, carefully

jointed together, with all cracks or seams completely closed. On the top of this box, and at distances of $2\frac{1}{2}$ ft. apart, are openings which are closed with covers H H, which are easily displaced by any explosion of gas which may occur in the box, and permit the exploded gas to escape.

To produce changes in the direction of the current of mixed gases from the horizontal, upward or downward, according as it may be desired that the gas should impinge upon the burning lamp, arrangements have been made to adjust the lamps inside to different levels (see fig. 665).

In order to make the necessary observations on the conduct of the lamps during the test, a part of the wooden box is inclosed by a wooden partition I, in such a way as to entirely darken it, and enable the observation of every change in the flame of the safety lamp to be made. The somewhat dangerous observation of the safety lamp

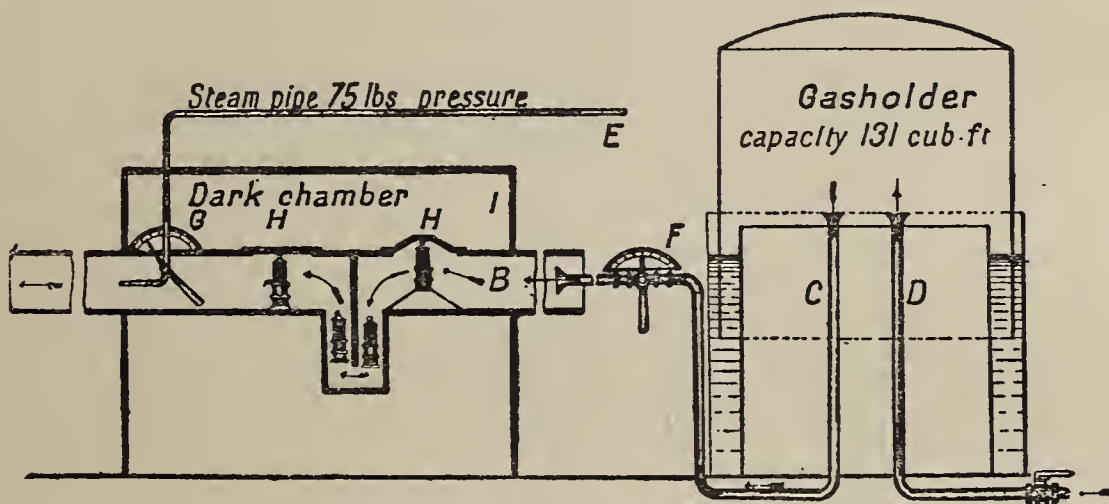


FIG. 665.—Special Testing Apparatus.

during a test is made through a glass plate $\frac{1}{8}$ in. thick, which is set in the door through which the lamp is placed in the box.

The velocity of the air current or gas mixture, which can be increased up to 59 ft. per second, is produced by means of a steam nozzle E, the steam having a pressure of 75 lbs. per sq. in. enters the nozzle through a valve to which an arm K is attached, by means of which it can be adjusted so as to produce the desired velocity. In order to adjust and indicate the velocity an anemometer is used, and a scale constructed which indicates the velocity by means of a pointer attached to the arm.

The gas is conducted to the apparatus by the pipe D into the gas tank, to which is attached a meter for measuring accurately the quantity of gas entering the holder.

By weighting the gas tank, the gas will be forced down the pipe C into the wooden box, the inflow being regulated and adjusted by a nozzle having a pointer and scale F attached, to show the *percentage* of the gas mixture.

The end of the pipe C, where it enters the box, terminates in a funnel-shaped opening which is covered with a gauze so as to prevent the flame, in case of an explosion caused by a lamp, from entering the gas pipe and communicating with the gas-holder.

This apparatus can only be used when a supply of coal gas is available.

Trimming and Cleaning Lamps.—Probably nothing is more important as regards safety lamps than proper cleaning, trimming and fitting together of the various parts of which they are composed, for this part of the work bears directly upon the safety of the whole colliery.

While the object to be aimed at is to get a good safe lamp, giving out a good light, simplicity of construction is also to be greatly desired, and this is wanting in many lamps brought before the public. When it is considered that at some collieries from 500 to 1000 safety lamps have to be cleaned, trimmed, and fitted together every shift, the necessity of simplicity in construction is at once apparent, because, if a lamp is complicated and consists of a great many parts, there is a great probability that, in preparing a large number for use, some may be imperfectly put together, and however safe a lamp may be when in perfect order, it may become most unsafe under these conditions. The principal parts of a safety lamp which are liable to get out of order, or may be improperly fitted together, are the gauzes, the glass chimney, and the joints between the latter and the metal. The glasses used for safety lamps should be of the best quality, ground smooth, and parallel at the edges; glasses with chipped or rough edges should be discarded, for it is almost impossible to get tight joints where they come in contact with the metal rings. In the latter, 'washers' of asbestos millboard should be introduced, which will give a good bearing surface to the edges of the glass. Washers of leather or india-rubber are not to be recommended, as they are perishable and shift their positions if they become strongly heated, which often occurs. The gauzes should also be carefully examined daily, and thoroughly cleaned so as to free them from any oil or coal dust which may have clogged the meshes while underground. At the majority of collieries the gauzes are cleaned by hand, and where the lamps are not very dirty this does well enough, if good brushes of the proper size and quality are used, but in collieries where the workings are muddy and the dirt adheres firmly to the gauze, the lamps are more difficult to clean, as the dirt adheres strongly to the gauze and cannot easily be removed. When this is the case, the lighting power of the lamp will become very inefficient, as the necessary air for proper combustion is unable to reach the flame.

Machines either worked by hand, steam, or electric power are now used for cleaning the gauzes and glasses. By this means the gauzes are cleaned more thoroughly, and are not subjected to such rough treat

ment as when cleaned by hand, there being also an economy in labour, an experienced man being able to clean 300 to 400 gauzes per hour.

Filling and Lighting Lamps.—As already stated, the utmost care should be exercised in charging safety lamps with oil, especially if the oil used is what is termed a 'light' oil, such as petroleum, naphtha or benzine, either of which is of a highly inflammable nature, especially when the temperature reaches a certain point. These oils, as a rule, have a low 'flash' point, and when they reach this point they give off a dangerously combustible vapour which will readily explode if brought in contact with a light. At collieries where these oils are used, the lamp-room should be so arranged that the lamps can be filled in a separate apartment to that in which they are lighted before distribution to the men. It will also increase the security against fire or explosion if the filling tank containing the oil is situated in a special vault outside the lamp-room, and a pipe led from it to where the lamps are filled.

Dripping pans should always be provided below each filling tap to receive any excess of oil. The dripping pans should be emptied at frequent intervals, as it is often from the overflowing or upsetting of one of these that accidents occur.

Tanks which work automatically are sometimes used for filling the lamps. Such an apparatus consists of an iron tank which holds 10 to 15 gallons of oil, and by means of a three-way cock placed at the bottom of the tank, the small quantity of naphtha necessary for the filling of a lamp is drawn from the tank into the glass reservoir, and by the turning of the cock is entirely shut off from that in the tank. When the sponge in the lamp will absorb no more oil, the outflow from the reservoir stops automatically. By this arrangement safety and economy are secured.

Note.—For students who desire more detailed information as to safety lamps, the author would recommend *The Report of the Royal Commission on Accidents in Mines*, 1886, where over one hundred such lamps are figured and described.

See "The Safety Lamps Order," August 26, 1913, in which all the approved types of safety lamps are described and illustrated.

EXTRACT FROM STATUTORY RULES AND ORDERS 1913, No. 886.

LIST OF APPROVED SAFETY LAMPS.

PART I.

Flame Safety Lamps Approved for General Use.

Manufacturer.	Name of Lamp.	Page.	Plate.
1. Ackroyd & Best, Ltd.	Hailwood, Nos. 01 and 01 S	4	1
	Hailwood, No. 2	5	2
2. Best's Safety Lamps, Ltd.	Best's Excelsior, No. 1	6	3
3. Joseph Cooke & Son	J.C.B., Nos. 19 and 22	7	4
4. Cremer Lamp and Engineering Co., Ltd.	Cremer, No. 11	9	5
5. John Davis & Son (Derby), Ltd.	No. 1, Davis-Marsic	10	6
	Davis-Marstry		
	Davis-Marstryn		
	No. 2, Davis-Kirkby		
	No. 1 B., Davis - Marsty-Loose.	12	7
	No. 5, Davis-Thornley	12	8
	No. 8, Davis-Thornburry		
6. Richard Johnson, Clapham & Morris, Ltd.	Nos. 1 and 2	14	9
	Nos. 4 and 4 A Deflector	15	10
	No. 5 J.C.M.	15	11
7. James Laidler & Sons	Nos. 1, 1 A and 2	16	12
8. John Mills & Sons	"J.M.S." No. 1	17	13
9. J. H. Naylor	N.		
	Bifold Burner Marsaut	19	14
	Marsaut "A" and "C"		
	Marsaut "B" and "D"		
10. Patterson & Co.	Types A 1 and A 3	21	16
	Type B 1	23	17
11. Protector Lamp and Lighting Co., Ltd.	Prestwich Patent Protector	24	18
	No. 176, Oil	25	19
12. J. H. Rothwell & Co.	"A"	27	20
	"C"	28	21
	"B," "D," and "E"	29	22
13. W. E. Teale & Co., Ltd.	Protector, No. 1	30	23
	Protector, No. 2	31	24
	Standard Bonneted Marsaut, No. 4	33	25
	Protector No. 3 and Standard Deflector Marsaut No. 5.	34	26
14. E. Thomas & Williams, Ltd.	Cambrian, Nos. 1 and 3	35	27 & 28
	Cambrian Deflector, No. 15	36	29

PART II.

Flame Safety Lamps Approved for Use by Officials only.

Manufacturer.	Name of Lamp.	Page.	Plate.
1. Ackroyd & Best, Ltd. .	Hailwood, No. 01 A . . .	37	30
	Hailwood, Nos. 06 and F 1	37	
2. Cremer Lamp and Engineering Co., Ltd.	No. 11 A	38	
3. John Davis & Son (Derby), Ltd.	No. 11, Davis-Beacatorh . .	38	31
	No. 3, Davis-Boss		
	No. 6, Davis-Alumthorn . .	40	32
	No. 7, Davis-Diabl	40	
	No. 3 C, Davis-Bossgy . .		
4. Richard Johnson, Clapham & Morris, Ltd.	No. 3 and No. 6	40	
5. James Laidler & Sons . .	No. 2 A	41	33
6. John Mills & Sons	"J.M.S." No. 2	41	
	N.	42	
	"J.M.S." No. 3		
	N.	43	
7. J. H. Naylor	Bifold Burner Marsaut (<i>for officials</i>).	43	34
	Marsaut "A 1" and "B 2" . .		
	Marsaut "E"		
8. Patterson & Co.	Types A 2, A 4, and B 2 . .	44	
	Type D 1	44	
9. Protector Lamp and Lighting Co., Ltd.	"Prestwich Patent Protector, A."	45	
10. W. E. Teale & Co., Ltd. .	Standard Bonneted Marsaut, No. 4 A.	45	35
11. E. Thomas & Williams, Ltd.	Cambrian, No. 1 A	46	
	Cambrian Improved Fireman's, No. 21.	46	

PART III.

Electric Safety Lamps Approved for General Use.

1. "Ceag" Electric Safety Lamp Co.	"Ceag"	47	37
2. W. E. Gray	Gray - Sussmann, Nos. 3 and 4.	48	38
3. Oldham & Son	Oldham	49	39
4. Wolf Safety Lamp Co. . . .	Wolf Alkaline	49	40
	Wolf Lead		

RESCUE APPARATUS.

“Rescue apparatus have been designed for the purpose of enabling men by its use to live under irrespirable atmospheric conditions. The portable rescue apparatus is a breathing apparatus enabling the wearer to penetrate irrespirable gases without being supplied with air from any outside source.”* This definition succinctly sums up the use of rescue apparatus in mines. The necessity for the use of such apparatus may arise when an explosion of fire-damp or an outbreak of fire takes place in the underground workings of a mine. It is well known that when an explosion of fire-damp or an underground fire takes place, the atmosphere speedily becomes irrespirable through the presence of the noxious gases which have been generated. The so-called after-damp resulting from an explosion of fire-damp is composed chiefly of carbon dioxide and carbon monoxide, and the products of combustion from an underground fire are practically the same, but the proportion of carbon monoxide is usually much greater in the latter. We have shown how very dangerous small proportions of CO in the air are to those who may be forced to breathe it, and how fatal the results are to human life. We have already referred to Dr Haldane’s estimate of 70 to 80 per cent. of the lives lost in colliery explosions being due to carbon monoxide poisoning, and all connected with mining know how disastrous to human life an underground fire may be from the same cause. Recently, an underground fire has taken place at Cadder Colliery, near Glasgow, resulting in the loss of twenty-two lives. The rescue apparatus is meant to enable men to enter the mine workings as soon after an explosion of fire-damp has taken place or an underground fire has been discovered, as the circumstances will allow, with the view to rescue those who may still be alive but entombed in the workings. While this is the laudable and humanitarian object of the use of rescue apparatus, it is to be regretted that up till now such apparatus has been the means of saving very few lives, in fact, some lives have been lost through the use of rescue apparatus in mines. At the same time, the apparatus has been found useful in exploration work, and in the extinguishing of fires in the workings; and for such operations, if for nothing else, rescue apparatus may prove to be very serviceable. The construction and use of rescue apparatus is not an idea of recent date, as scientific men have worked at the problem for many years. In 1846 Sir Henry T. De la Beche and Dr Lyon Playfair, in a report to Viscount Canning, stated “that after a coal mine explosion a sufficient quantity of oxygen remains to support the respiration of those who survive its effects were it not for the presence of carbon dioxide”; and they suggested “that a mixture

* *The Use of Oxygen Breathing or Rescue Apparatus*, by Arthur T. Winborn, p. 8.

of Glauber salts and lime would prove useful to those who were trying to rescue the sufferers after an explosion, such a mixture to be placed in a coarse bag and applied to the mouth."

Professor T. Schwann, professor of physiology at the University of Liège, constructed an apparatus in 1854, and in a paper published by him twenty-two years later, he explained that his breathing apparatus depended upon the principle that the same volume of air can be continuously employed for respiration, upon the condition that it is freed from carbon dioxide, and that the oxygen consumed in the act of breathing is replaced by fresh oxygen. In 1878 Professor Albert Habets of Liège stated that "the apparatus of Professor Schwann is the starting-point of a great improvement which will allow of rescue operations being carried out in dangerous atmospheres."

The well-known Fleuss apparatus followed in the main the same idea as Professor Schwann, and it was exhibited in Britain in 1880, and in the following year it was used at the Seaham Colliery explosion for rescue work. Very little progress was made in the use of such apparatus—at least in this country—until 1899, when Mr (now Dr) W. E. Garforth suggested that an experimental gallery should be erected to test the different forms of apparatus that might be introduced to coal-owners, and to train men in the use of them. This experimental gallery, made like the damaged roadways of a mine after an explosion, was erected at Altofts Colliery in Yorkshire, and was the first of its kind in Britain or in Europe. A full description of this gallery will be found in the *Transactions of the Institution of Mining Engineers*, vol. xxii. p. 169. During the year 1901 a number of tests were carried out at the Altofts gallery with the view of practically testing the best forms of live-saving apparatus known at that time, viz., the pneumatophore, the Mayer helmet, and Giersberg apparatus. The tests were far from satisfactory, as the various apparatus could not be worn without much discomfort and distress to the wearers. It was therefore decided that further testing should be discontinued with such imperfect apparatus, and to await the introduction of a more reliable appliance which could be used without causing so much discomfort, and which would allow of more useful work being done. Meantime, engineers, principally in Germany, were working to improve the construction of rescue apparatus, chiefly in the supply and control of the oxygen supply under high pressure to the wearer. The lamentable explosion at the Courrières mines on 10th March 1906, by which over 1100 men and boys lost their lives, gave a great impetus to the employment of rescue apparatus. On the day following the explosion, a rescue party from Westphalia went to Courrières with rescue appliances, and for fifteen days assisted in the exploration of the workings and the recovery of the bodies, and also assisting to cope with the fire which had broken out in sections of the mine. The

apparatus used in this instance was that known as the 'Shamrock,' which is now known as the improved 'Meco' apparatus. In this, which was the first practical test of rescue apparatus on a large scale in actual underground work, one of the German rescue party lost his life while wearing the apparatus, showing the danger there is entailed upon those who undertake such work, and showing how fallible such appliances may be at a critical moment. A number of fatal accidents of a somewhat similar kind have taken place in British collieries to members of rescue parties who were well trained to the use of the apparatus, which shows how imperfect these rescue appliances are in their present construction.

The British Royal Commission on Mines (1908), in their final report after exhaustive experiments, recommended the use of breathing appliances for rescue work, and in 1910 an Act was passed by Parliament embodying the recommendations of the Commission. This Act was repealed by the Coal Mines Act, 1911, which came into force on 1st July 1912, and on 2nd April 1912 an Order was issued by the Home Office making the use of rescue apparatus compulsory in all mines producing coal, but not to apply to mines employing less than 100 persons, if certain specified conditions were complied with. This Order makes it compulsory for rescue brigades to be organised and maintained as soon as practicable on the following scale: where the number of persons employed underground is 250 or less, 1 brigade; over 250 but not exceeding 700, 2 brigades; over 700 but not exceeding 1000, 3 brigades; over 1000, 4 brigades. Each brigade is to consist of not less than five mine employees, certified to be medically fit, and a majority to be trained in first-aid and hold an ambulance certificate. One of each brigade is to be selected as a captain. The Order also defines the course of training the persons forming the brigades have to undergo. Two suits of portable breathing apparatus are to be provided for each 250 men employed underground, and, in addition, every mine which maintains a rescue brigade must provide and maintain two or more small birds or mice for testing for carbon monoxide; two electric hand lamps for each brigade, capable of giving light for four hours; one oxygen reviving apparatus; safety lamp for each member of the brigade, and a St John's Ambulance Association box or similar box. Rules are to be adopted at every mine for the conduct and guidance of persons employed in rescue work, and the brigades are to be thoroughly instructed in the rules. The Order allows the provision of central rescue stations which may hold and provide the breathing apparatus for all collieries situated within an area having not more than ten miles radius. Many mining engineers think that this Order was premature in view of the imperfect construction of the breathing appliances which were available, and as stated some time ago by Professor John Cadman of Birmingham University, that "although legislation to enforce the universal

application [of rescue apparatus] was inevitable—and indeed desirable—it seems unfortunate that a little more time was not given to permit of a more thorough investigation of the subject.” The Home Office has evidently recognised this fact, as they have now (August 1913) intimated that a technical committee will be ap-

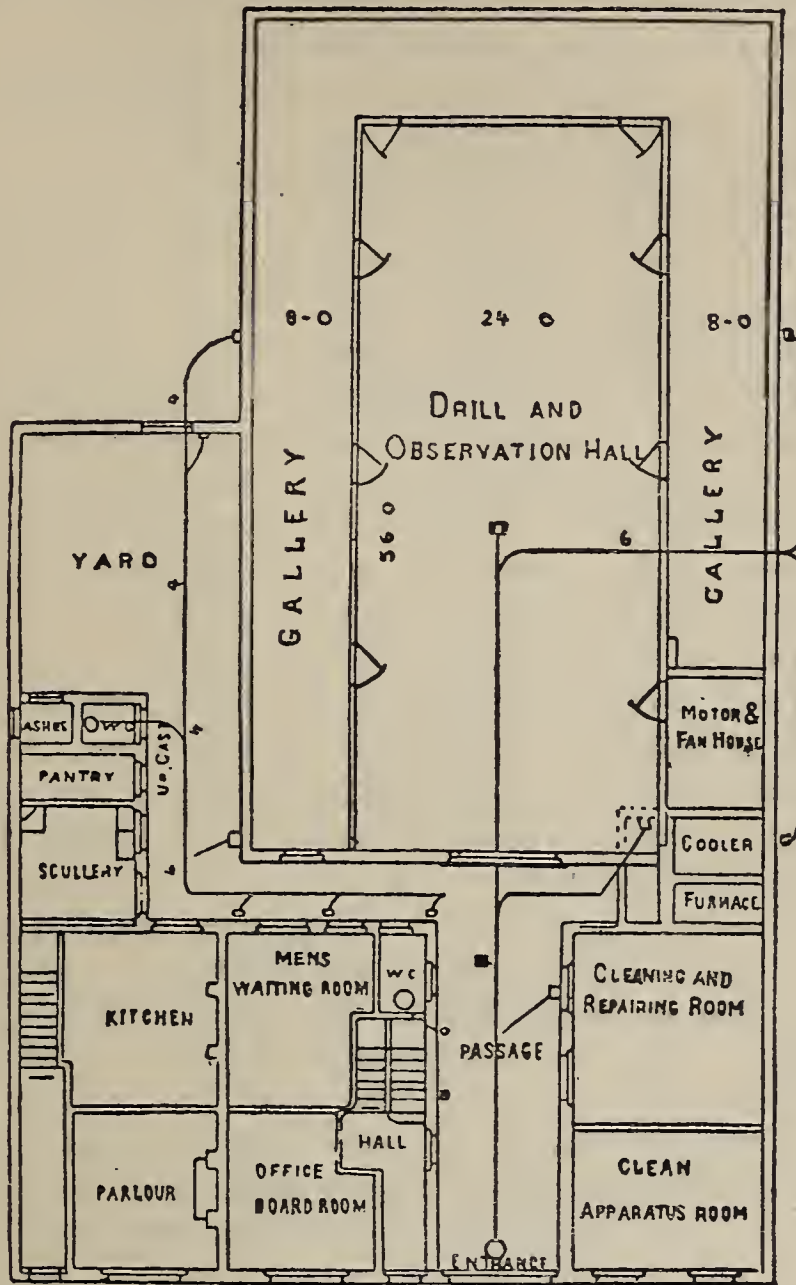


FIG. 666.—Ground Plan of Howe Bridge Rescue Station, Lancashire.

pointed to investigate and report as to the best type of rescue apparatus for use in mines. Dr W. E. Garforth* defines what he considers should be aimed at in the construction of such apparatus. He states that: “The apparatus should be so designed that it is light in weight, and not irksome to carry, even in thin seams; it should be so simple in construction that there may be no doubt of obtaining from it a full and automatic supply of oxygen combined with nitrogen, or regenerated air, and a complete absorption of the

* *Trans. Inst. Min. Engs.*, vol. xxxi. p. 625.

vitiated air from the lungs: and the wearer of it should be fully protected from the surrounding noxious atmosphere by a pneumatic joint connecting the apparatus with the mouth. The apparatus should be so constructed as to leave both hands free, to rescue or to afford relief to sufferers. It should protect the eyes, but still allow them to penetrate a dimly lighted and heated roadway, and should leave the ears open to every cry for help or to noise caused by a fall of roof. It should shield the head from projecting stones or timber, and should not overburden the body by excessive bulkiness, but leave the wearer capable of considerable exertion whilst working in, or surrounded by, a poisonous and moist atmosphere of high temperature. At the same time, it should enable the explorer to carry an electric lamp in his belt, an arrangement which permits both hands to be at liberty. The whole device should leave, as far as possible, the mind of the explorer free to think about the best means to be adopted, so as to overcome unexpected difficulties."

These are very comprehensive and exacting requirements, and we know of no rescue appliance at present in use that would comply with them. The most of the present types of apparatus are complicated in construction, clumsy and heavy, and not very suitable for the underground workings of a mine, especially the conditions that would be met with after a severe explosion of fire-damp, where many of the roadways are frequently wrecked and rendered almost impassable.

Doubtless improvements will be made in the construction of these appliances, and that they will be made more suitable for the conditions to be met with in mine rescue work.

'Proto' Breathing Apparatus.—This was formerly the Fleuss apparatus, but has been developed and improved under the Fleuss-Davis patents, and is manufactured by Messrs Siebe, Gorman & Co., Ltd., of London. It is used at the central rescue stations at Howe Bridge, near Manchester, and at Stoke-on-Trent. It is of the non-injector type, the circulation of air being brought about by the action of the breathing upon two mica respiration valves leading to the mouth (L and M, fig. 667). The alkali used as an absorbent for getting rid of the carbon dioxide, when exhaling, is caustic soda instead of potash, the advantage claimed for the soda being that it remains practically hard, dissolving away from the surface only, whereas potash, when exposed to moisture, becomes a pasty mass which reduces the area of its absorptive surface. The caustic soda is carried loose in a flexible breathing bag (D, fig. 667) placed on the chest, and not in metal cases on the back as in other types of apparatus. This enables a man to lie flat upon the bag and still breathe freely, and it is also of advantage when crawling through low or obstructed parts of a roadway. The portion of the apparatus which is suspended from the back consists of the oxygen cylinders (B B, fig. 667), to which is attached a reducing valve C for regulating the supply of oxygen from the cylinders. If the reducing valve C fails

to act or the breathing bag becomes deflated, an emergency valve I is provided which, on being opened, inflates the bag again and allows breathing to go on. To show the pressure of the oxygen in the cylinders a pressure gauge P is provided, and rests in a pocket provided upon the front of the breathing bag. This gauge is connected

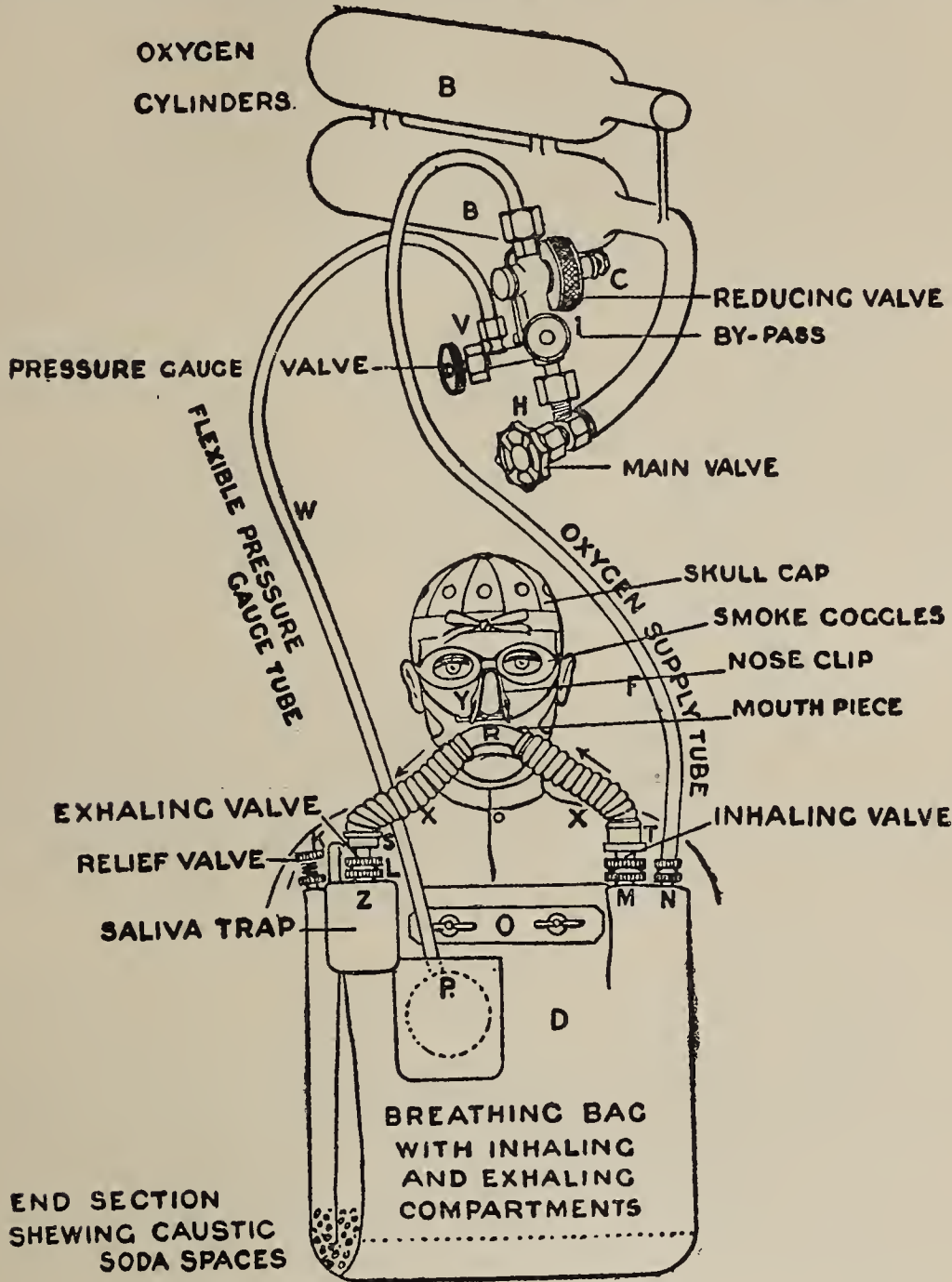


FIG. 667.—Diagrammatic View of 'Proto' Rescue Apparatus.

to the cylinders by flexible tubing capable of withstanding a pressure of 1764 lbs. per square inch, and the pressure can be tested when required by opening the valve V which allows the oxygen to flow immediately to the pressure gauge.

The breathing bag is furnished with a small relief valve K for the escape of any superfluous air, and also with a saliva trap Z. The relief valve does not work automatically, which is a disadvantage, but has to be manipulated by the wearer when necessary. The

breathing bag—which is made of vulcanised india-rubber, and has an outer protecting covering of canvas—is divided into two compartments, inhaling and exhaling. These compartments are arranged so as to ensure the passage of the expired air through the caustic soda, which is distributed in both compartments equally. The mouth-piece R is made of rubber, and is used in conjunction with a light canvas cap furnished with straps which prevents the mouthpiece from being inadvertently knocked from the mouth.

The goggles are used only in cases where a man has to work in smoke or in gases which affect the eyes. They should be adjusted beforehand to fit the eyes accurately; wetting the rubber cushion with water will help to ensure an air-tight joint. Before putting on the goggles the glasses should be wetted inside to prevent fogging.

The whole apparatus is supported upon a broad belt which is strapped round the body with three straps and buckles. There is also a leather strap attached to the belt, which is to be passed through a pair of leather loops upon the body side of the bag to hold it in position when stooping, crawling, etc. The bag is also hung by a pair of shoulder braces with adjustable straps and buckles at the back to suit the height of the wearer. The total weight of the apparatus when fully charged is 32 lbs.

'Meco' Apparatus.—The *'Meco'* apparatus (figs. 668–671), which is made by the Mining Engineering Co., Sheffield, is of the injector type, and can be worn with a helmet, a mouth-breathing piece, or a mask, as in the *'Weg'* apparatus. It consists of the usual two back and front portions, the back portion comprising the oxygen cylinders, automat, regenerator or purifier, and cooler; whilst the front portion contains the breathing bags and flexible pipe connections to the mouthpiece or helmet.

The breathing bags, which are used to equalise the varying demand during breathing, are fitted on the chest, and are protected on the outside from being cut or torn by means of a protecting cover, which is attached with straps and swivels to the shoulder straps and the back carriage, so that although the breathing bag is held in position on the chest, it is sufficiently free to allow of expansion. The same apparatus can be used for mouth and helmet breathing. If it is to be used for mouth breathing, the unions (113*a* and 118*a*), which are fitted to the hoses (113 and 118), are screwed into the mouthjet (114), on to which the mouthpiece is fitted (115), see fig. 671. This mouthjet is divided into two compartments by means of a metal strip in the middle. Through one compartment of the mouthjet the pure oxygenated air is brought direct into the wearer's mouth, and the other compartment serves to suck away the exhaled air, so that the re-inhalation of the exhaled air is absolutely impossible. The rubber mouthpiece (115) is attached to the mouthjet in a very simple manner by simply being stretched on to it: by this means a firm fitting of the mouthpiece is accomplished, and it allows the mouth-

piece to be quickly changed after use, so that it can be cleansed and disinfected. To prevent the inhalation of poisonous gases through the nose, a spring screw noseclip (117) is fitted round the mouthjet so that it is impossible for it to be either loosened or knocked off. The mouthjet and noseclip are held in position by means of straps attached to a skull-cap. This attachment prevents these parts from falling out or being pulled out of the mouth. The cap also takes the weight of the mouthpiece off the lips and gums.

As already stated, the same apparatus can be used with a helmet instead of mouthpiece. The helmet (129) (fig. 668) is screwed on to the unions (113a and 118a) between the inhalation and exhalation tubes (113 and 118) in a similar way to the mouthjet. The fresh air enters the helmet in close proximity to the mouth, and is therefore brought directly in contact with the mouth and nose. The warm exhaled air is sucked away at the top of the helmet, so that an automatic and unbroken circulation takes place at the top of the helmet, and a re-inhalation of exhaled air is prevented.

The face washer is fitted with a pneumatic inner lining (150), see figs. 668 and 670. It is necessary to pay particular attention to the pneumatic fittings, and to see that they are in good condition. All that it is necessary to do is to remove the clip ring (131) and a small tube (135).

A small rubber ball (151), which is used for pumping up the pneumatic lining, is fixed on the outside of the helmet, and for head protection; the helmet is fitted with an adjustable leather cap (132). This cap ensures a comfortable fitting of the helmet, and is held in position by a pulling-up strap (136), which keeps the helmet in position. The padded bars (137 and 138) (fig. 669) press against the forehead and chin respectively, so that the helmet is prevented from slipping or being knocked off. The back of the head and the neck of the wearer are covered with a leather protection shield (139), which is a continuation of the leather cap (132).

The observation window (140) is protected from blows or pressure by means of cross bars (141), which are also utilised to open and shut the window. To remove any condensed moisture, etc., which may form on the window, a cleaning pad (142) is fitted. This is rotated by means of knob (143) and so cleans the windows. Where required, the window can be made of mica instead

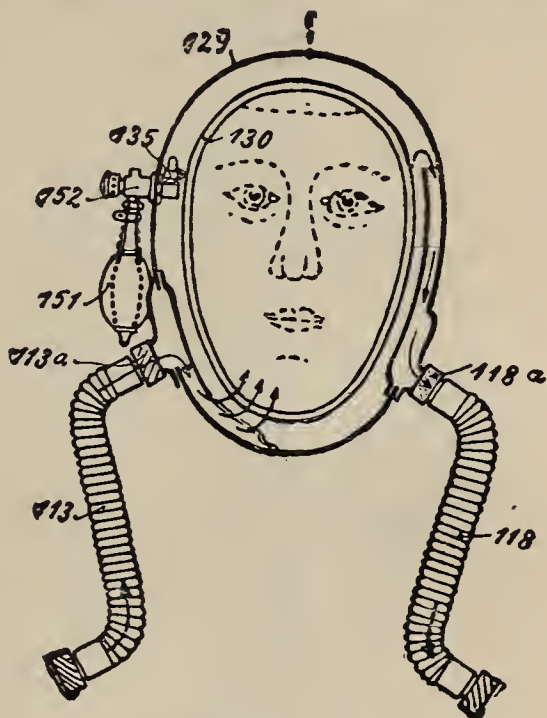


FIG. 668.

of glass, and the exchange can be effected by removing the spring ring (145).

The working of the apparatus is as follows:—After opening the oxygen valve (102*a*) (fig. 671), the compressed oxygen contained in the steel cylinders (101) passes through the T piece (104*b*), which is provided with a pressure gauge (103), and so reaches the reducing valve (105), which is fitted with a safety valve (106). The oxygen is reduced to a predetermined pressure and the flow of same controlled by means of the reducing valve, from where it passes to the injector (107). The high velocity of the oxygen passing through the injector sucks the purified exhaled air passing out of the regenerator (110) through the cooling pipe (146). This air and oxygen are mixed together and forced through the tube (144) and then through the hoses (148, 112, and 113) into the mouthpiece (114) or into the

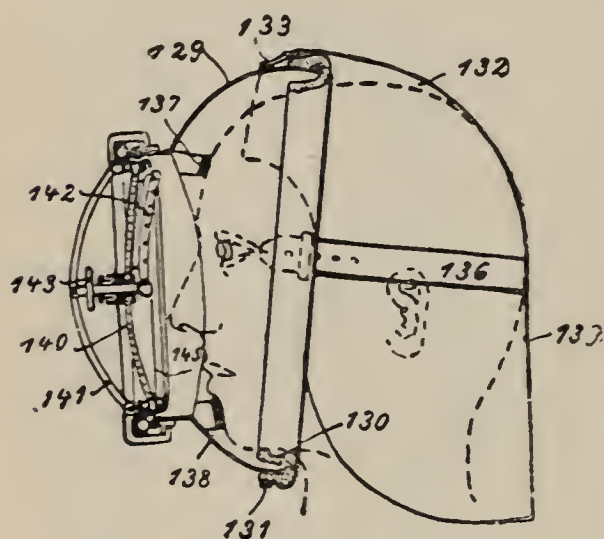


FIG. 669.

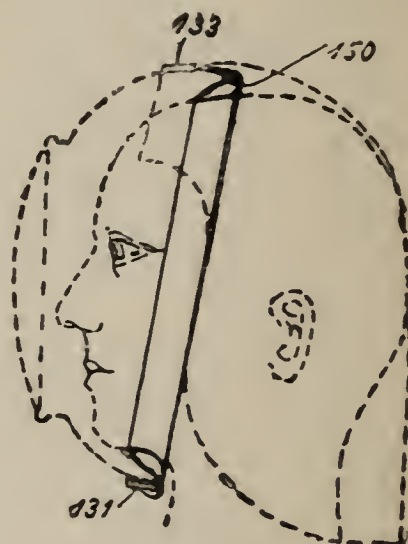


FIG. 670.

helmet (129), fig. 669. In passing through from the oxygen cylinders to the injector jet, the oxygen passes through three sets of gauzes and felt washers, so that it is impossible for any dirt or impurities to get into the injector jet. When the apparatus is used for mouth breathing the nose is closed by means of a spring screw noseclip. The mouthjet (114), with the mouthpiece (115), is held in position by means of the straps on the skull-cap.

The exhaled air is sucked away from the mouth by means of the injector, direct into the regenerator (110). It passes through the pipe (147) into the bottom part of the regenerator (110). The path of this air through the regenerator is shown by arrows, and it passes out at (146*a*) as shown in illustration. The regenerative material in the regenerator (110) is arranged in trays (125), which consist of wire gauze, the upper as well as the lower surface of the regenerative material being utilised for absorption. In the middle of each tray (125, 125) is placed a strip of impervious material which absorbs any moisture which may be formed, and compels the air passing through the regenerator to travel in a zig-zag manner

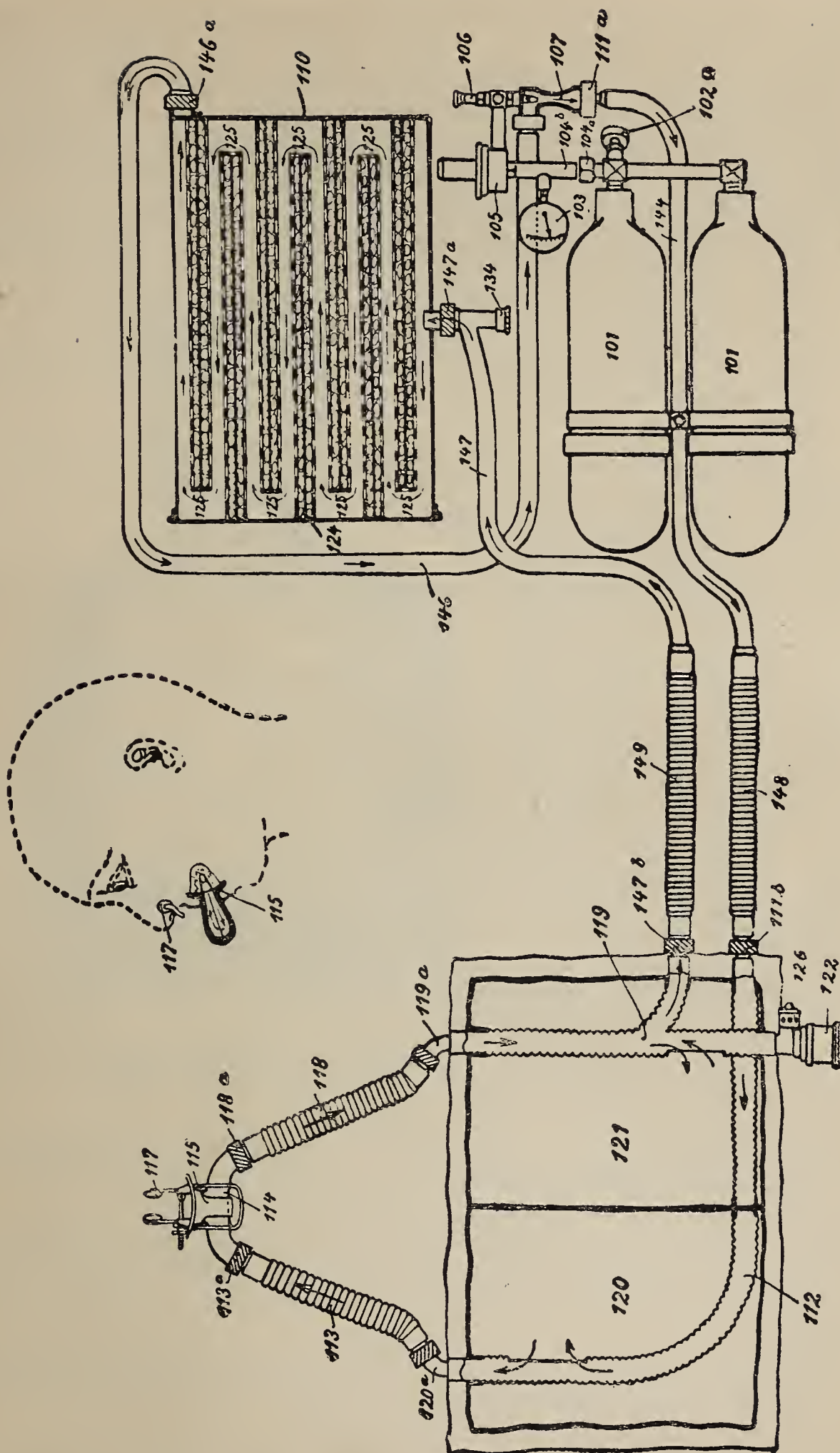


FIG. 671.

The regenerator is supplied either of the interchangeable type, which has to be thrown away after use, or of the refillable type, which can be refilled *in situ* and again utilised. To exchange the regenerator, all that is necessary is to loosen the union (147a) on pipe (147) and the union (146a) on pipe (146), and the regenerator can be removed. The hoses (112 and 119) pass through the inhalation bag (120) and the exhalation bag (121) respectively, and are connected with the bag itself by means of an aperture. The inhalation bag (120) serves as a reservoir for the purified oxygenated air for use in cases where an unusually large demand is put upon the apparatus, and sufficient air is stored in the breathing bag to overcome this. To meet this demand the exhalation bag (121) serves in a similar manner for the exhalation side and permits of comfortable and easy breathing.

The saliva which may be generated with the mouth-breathing apparatus flows through the hose pipe (118, 119) (fig. 671) direct into the removable saliva trap (122).

As the apparatus supplies more purified oxygenated air than is required during ordinary exertion, an excess relief valve (126) is fitted on the lower part of hose (119), which relieves the apparatus of any excess pressure.

To maintain the apparatus ready for immediate use the regenerator must be protected from the outside atmosphere.

Draeger Apparatus.—This apparatus, which is made by Richard Jacobson of London, consists of two main parts, one being carried on the chest and the other on the back of the wearer, the back and front portions being connected together by two flexible circulation pipes. The apparatus can be employed either with a helmet or mouth-breathing appliance. The back portion of the apparatus, which serves to purify and renew the air, consists of—(a) the supporting frame, with the back rests, the protecting arched rods, the cartridge carrier, the automat, and the cooler (K, fig. 672); (b) the potash cartridge P; and (c) the oxygen cylinder W. The front portion serves for respiration, and consists of (d) the helmet H, the mouth-breathing attachment and the two breathing bags, L₁, the inhalation, and L₂, the exhalation bag.

The automat is an important portion of the apparatus, and is provided with a protecting device, which, by conducting or absorbing the heat, obviates the burning out of the valves.

The 'finimeter' is a high-pressure gauge of special construction, and indicates the quantity of oxygen in the steel cylinder. The duration of the working period in each case can be read off on the dial plate.

The reducing valve is charged with the important duty of lowering the pressure of the oxygen in the cylinder (up to 150 atmospheres) to a constant working pressure for the injector.

The safety valve blows off in the event of the working pressure

being exceeded. It has no connection with the actual working of the apparatus, and is provided solely in the interests of general safety. The injector acts as a motor for keeping the air of the apparatus in circulation, the motive power being furnished by the oxygen flowing through the nozzle. The latter also acts as a mixer, restoring to the air in circulation the oxygen consumed in the process of respiration. The volume of air set in motion by the injector should amount to at least 50 litres ($1\frac{3}{4}$ cub. ft.) per minute. The

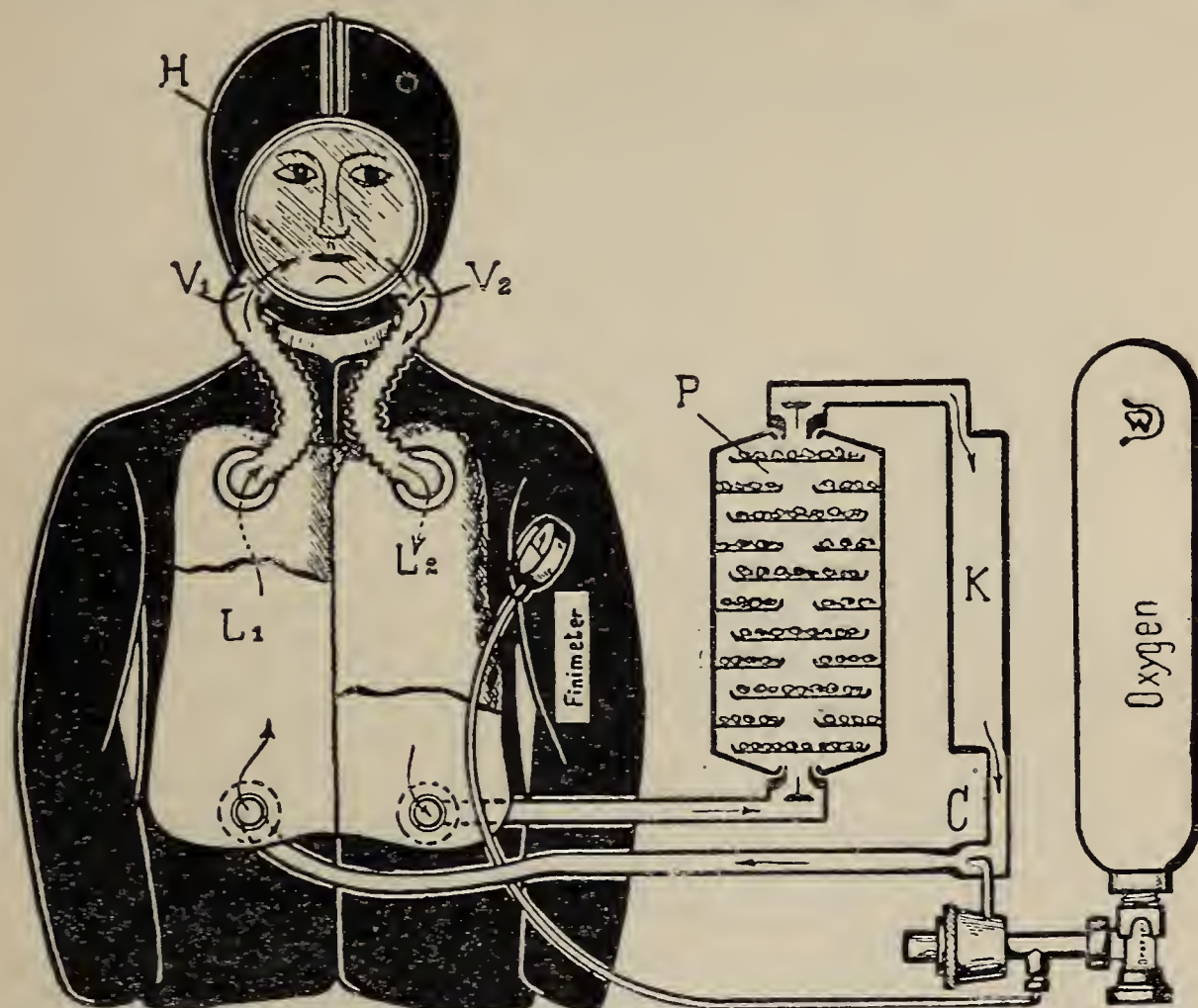


FIG. 672.—Diagram of Draeger Apparatus.

P = Draeger's Alkali-Cartridge.
L₁ = Purified Air.

K = Cooler.
L₂ = Expired Air.

C = Injector.

measured quantity of oxygen supplied by the injector is 2 litres (120 cub. in.) per minute, and is measured out automatically. The oxygen cylinder valve enables the apparatus to be either set in full action or entirely stopped. The cartridge becomes heated in consequence of the absorption of water and carbon dioxide, and hence the effluent air is also warmed. The liberation of heat increases with the amount of work done by the wearer. To prevent the air of respiration from becoming too hot, the apparatus is provided with the cooler K, the metallic surfaces of which disperse a portion of the heat into the surrounding air by radiation. In the latest of the Draeger apparatus, the two breathing bags are employed as cooling

surfaces, the circulating inhaled and exhaled air being obliged to flow through the bags.

An individual performing heavy work continuously for two hours exhales about $3\frac{1}{2}$ cub. ft. or 7 ozs. of carbon dioxide. The whole of this carbon dioxide is retained by the potash cartridges, which contain a large number of small granules of caustic potash or caustic soda, lying on about 20 shallow trays. These trays are arranged in such a manner that the air to be freed from carbon dioxide is compelled to pass over all the trays and granules in succession, without any hindrance being offered to its movement (see fig. 672).

The charge contained in the so-called potash cartridges consists of only one-third of caustic potash granules, the other two-thirds being granulated caustic soda. This material has a porous, crystalline structure, which enables the granules to be converted into the carbonate of potash or soda respectively, without deliquescing. The absorption of moisture—chiefly by the caustic potash—is only intended to be effected to such an extent as to prevent any deposition of water from the purified air, whether in the cooler, injector or flexible pipes, the air retaining sufficient moisture to render it agreeable to breathe. Each cartridge is provided with a detachable label on which the weight of the cartridges is printed, so that the increase in weight after a practice can be ascertained.

The helmet contains the following parts: pneumatic tube with bulb inflator, air escape valve, leather neck guard and neck straps, window and lattice, air valve, window cleaner, and two mica-plate valves.

The mouth-breathing attachment can be used in place of the smoke helmet. Many men are able to work with mouth breathing; but, generally speaking, it seems that more careful training is needed for mouth breathing than when a helmet is used.

The actual mouthpiece is made of rubber; the nostrils are closed with a nose-clip, and the eyes are protected by smoke goggles. The breathing bags consist of a bag for the inhalation air and one for the exhaled air. Screw connections are provided to enable the bags to be unscrewed for cleaning.

The two bags are connected by a relatively narrow communicating passage, so that when an unusually large supply of air is required, the contents of both bags may be drawn upon for filling the lungs.

A blow-off valve is also provided in connection with the breathing bags to enable the surplus air to escape.

When the apparatus is donned, the lungs and respiratory organs of the wearer are completely shut off from the external air, and form, as it were, a portion of the apparatus. In this total system, into which air is no longer admitted from the outside, there are only two forces in operation for actuating the air, viz., the lungs of the wearer and the injector C of the apparatus. The injector is operated by the power of the compressed oxygen. Between these

two forces are interposed the breathing bags L_1 , L_2 , as an air reservoir, and at the same time acting as a compensating buffer between the intermittent respiration of the human lungs and the continuously operated circulation of air in the apparatus. The lungs actuate the air in order to nourish the body with oxygen and to expel the carbon dioxide, whilst the apparatus sets the air in motion for the purpose of eliminating the carbon dioxide with which it is contaminated, and of enriching it once more with fresh oxygen.

The exhaled air passes through the exhaling valve V_2 , into the exhaled bag L_2 , from which it is drawn in a uniform manner. It next flows through the potash cartridge P , where the carbon dioxide expelled from the lungs is absorbed and thus rendered harmless. At the same time the cartridge becomes heated. The air, now free from carbon dioxide, next reaches the cooler K , where a portion of the heat it has absorbed is dissipated by radiation, and whence it passes to the injector C . Here it is revived with oxygen supplied from the oxygen cylinder at the rate of 2 litres (120 cub. in.) per minute. The air mixture is now again respirable, and flows through the second circulation pipe into the inhaling bag L_1 , whence it is inhaled through the inhaling valve V_1 . The cycle of operations then begins afresh. The apparatus is able to purify about 3000 litres (105 cub. ft.) of air per hour, supply about 120 litres ($4\frac{1}{2}$ cub. ft.) of oxygen, and absorb 50 litres ($1\frac{3}{4}$ cub. ft.) of carbon dioxide.

‘Weg’ Breathing Apparatus.—The ‘Weg’ breathing apparatus, designed by Dr W. E. Garforth, is of the regenerator type, wherein the same volume of air is breathed repeatedly, having the products of respiration abstracted and the oxygen replenished from cylinders of compressed oxygen (see figs. 673 to 676). $A A$ represent two oxygen cylinders, made of steel, curved to fit the contour of the body, and containing oxygen at a pressure of 120 atmospheres, or 1800 lbs. per sq. in., their capacity under this pressure being about $10\frac{1}{2}$ cub. ft. They are fitted with cocks $B B$, which can be operated at will, the usual practice being to utilise the contents of the left-hand cylinder first. The exhaustion of this cylinder when exploring may be taken as a warning to return to the shaft or other place of safety, the contents of the right-hand cylinder being used for the return journey. A very fine wire sieve is fitted inside the

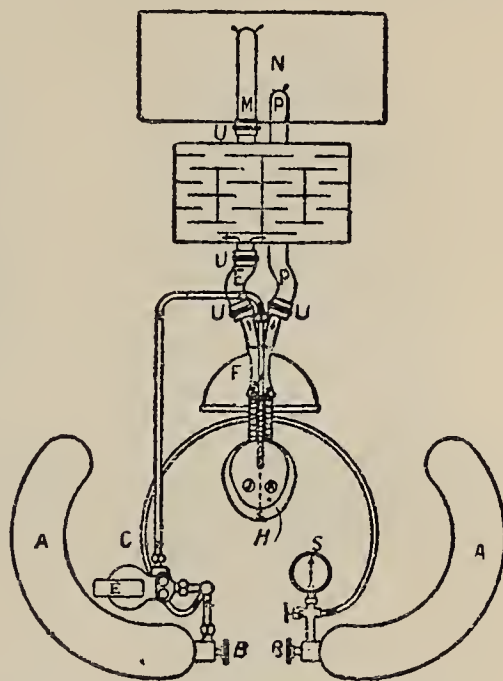
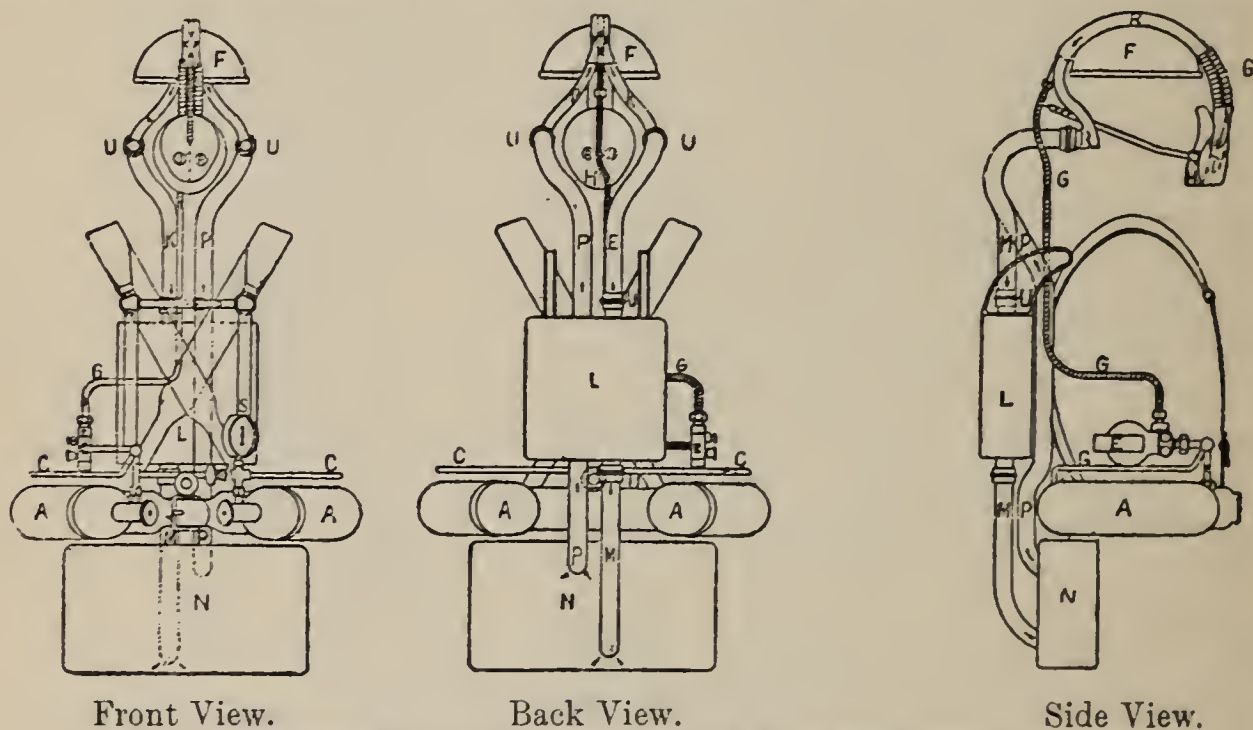


FIG. 673.—‘Weg’ Apparatus ;
Diagram of Head Piece,
Oxygen Cylinder, Purifier
and Connections.

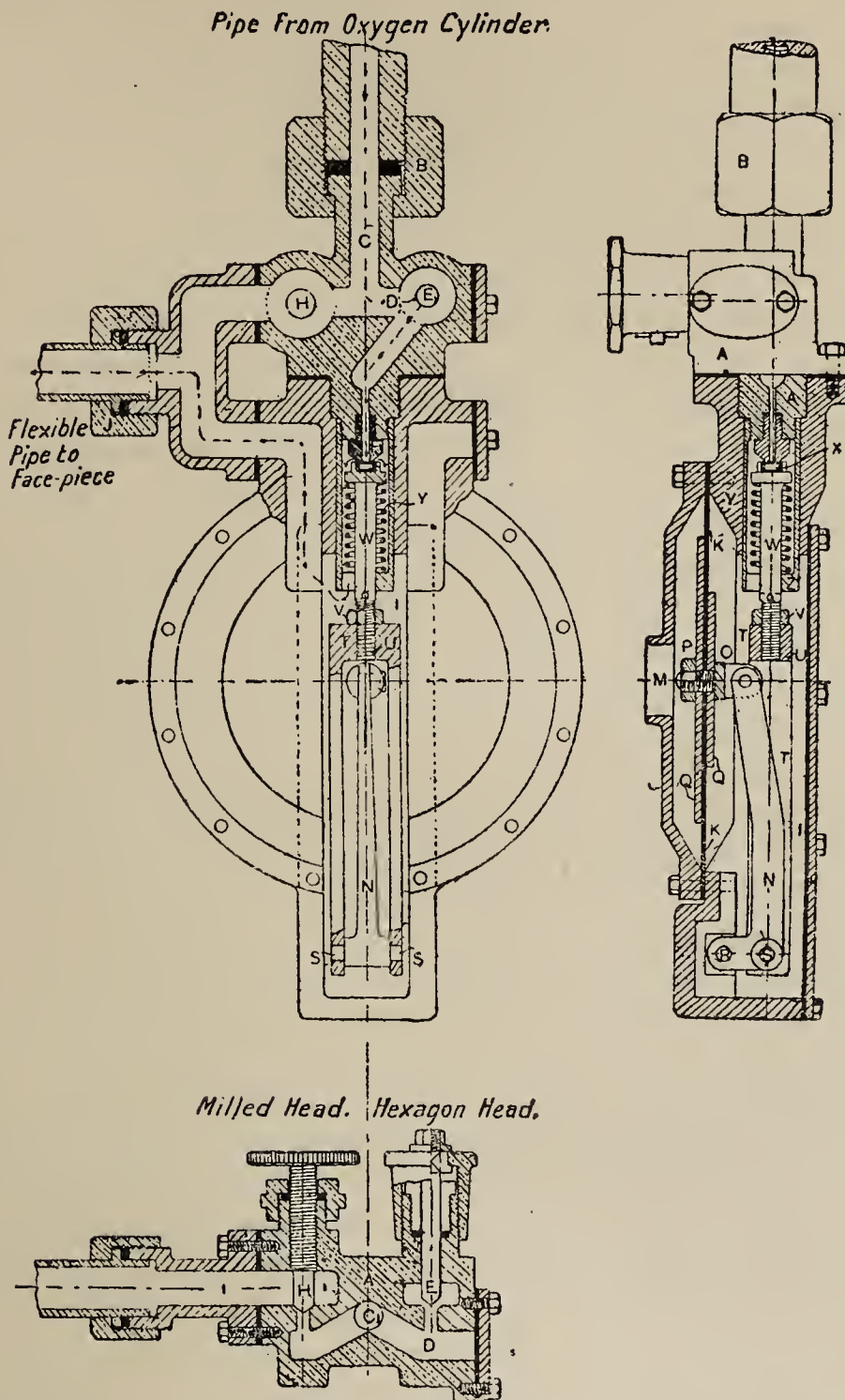
cylinder to prevent rust from the cylinder finding its way into the valves and choking them. The two cylinders are coupled together by a small copper pipe C, on which a pressure gauge S is fitted, the hinged cover being provided with a mirror, which reflects to the wearer the pressure and quantity of oxygen remaining in the cylinders. The oxygen passes from the cylinder through a reducing and lung-governing valve E to the mouthpiece H. This lung-governing valve forms one of the principal features of the apparatus. Fig. 677 is a plan of the valve partly in section; fig. 678 is a cross-section through the base, and fig. 679 a longitudinal section. The base A is made of gun-metal, and is secured by a pipe and union to the oxygen cylinders. On the stop valve of the oxygen cylinder



FIGS. 674, 675, and 676. — 'Weg' Apparatus.

being opened, the high-pressure oxygen passes through the port C to the centre of the base. At this point the port C divides; to the right the port D leads through a stop valve E, which is normally left open, to the seat F of the lung-governing valve, and to the left the port C communicates with the port I, which is normally closed by the 'by-pass' or emergency valve H. When the by-pass valve is open, oxygen passes direct to the mouth and lungs without going through the lung-governed valve, and the user is thereby enabled to supply himself with any required amount of oxygen additional to that passing through the lung-governed valve. The stop valve E is provided to cut out the lung-governed valve in case of an injury or breakdown of the mechanism of the latter, and the wearer then supplies himself with oxygen at his discretion through the by-pass. In order to prevent the wearer from making any mistake in the event of it being necessary to operate any of these valves in the dark, the emergency valve H and the stop valve E are provided

with differently shaped heads, the former being circular with a milled edge, and the latter octagonal. As the high-pressure oxygen does not extend beyond the base A, the case of the lung-governed valve is made of aluminium for the sake of lightness. The valve



FIGS. 677, 678, and 679.—'Weg' Apparatus ; Lung-Governing Valve.

case is closed on one side by the rubber diaphragm K, which is provided with a protective cover L. One side of the diaphragm is in communication with the outside atmosphere by means of the hole M in the protecting cover, and is acted on by the pressure of the atmosphere ; the other side of the diaphragm is in communication with the lungs by means of a flexible pipe connected with

the mouthpiece, and is acted on by the lung pressure. To the diaphragm a bell crank lever N is attached by means of an eye-bolt O and a nut P. The hole in the diaphragm through which the eye-bolt passes is made gas-tight by means of the washers Q, the inside washer being large enough to prevent the rubber diaphragm from being drawn in too far by encountering a stop piece which limits the travel of the diaphragm. The bell crank lever is suspended from the fulcrum pin R, and by means of a pin S moves the compound lever T, which is fastened by the adjusting screw U and the nut V to the plunger W, holding the main valve X, which is held against its seat F by the spring Y, which works around the plunger and is contained in the cylinder Z. The valve works as follows:— Except during inspiration, the diaphragm K occupies the position shown in fig. 678, and the valve is then held tight against its seat. When, owing to the action of inspiration of the wearer, the pressure in the interior of the valve chamber falls, the diaphragm is pressed inwards by the pressure of the outside atmosphere. This movement of the diaphragm is transmitted by the bell crank lever and the compound lever to the valve. The excess of pressure on the large area of the outer side of the diaphragm, due to the disturbance of the equilibrium by the action of inspiration, multiplied by the leverage obtained by the bell crank and compound levers, and assisted in some degree by the pressure of the oxygen on the under side of the valve, is sufficient to compress the spring and lift the valve from its seat. At the moment inspiration ceases, the spring closes the valve until the next inspiration begins. As the movement of the diaphragm is caused by the inspiration of the wearer, and as the duration or depth of the inspiration is a direct measure of the demands of the lungs for oxygen, it follows that the deeper the inspiration the greater the quantity of oxygen which will pass through the valve to the mouth and lungs. The lungs themselves may therefore be said to control automatically the supply of oxygen to the wearer; and there is, firstly, always the requisite quantity of oxygen available for each breath, and, secondly, there is no waste of oxygen. The oxygen then passes through the flexible metallic tube G (fig. 676), which is carried round to the back, and over the head of the wearer to the mouthpiece. The mouthpiece is made of special metal, and is of limited capacity and size, so that the force of the lungs may expel all the vitiated air contained in it. It is also of small area, so as to enclose only part of the nose and chin, allowing of both free breathing and perspiration, also leaving the ears open to every cry for help, or noise caused by falls of roof, and at the same time allowing the wearer to talk, and also speak through a telephone which may be attached to the mouthpiece of the leader of each exploring party.

The mouthpiece is connected to the face by means of a pneumatic joint, which renders it air-tight, and it is held in position by the

inhalation, exhalation, and oxygen tubes P, K, and G (figs. 674–676), which take the form of a three-way metal tube, carried over the head, and fastened to an ordinary miner's leather cap. The three-way tube, besides securing the mouthpiece, acts as a protection to the head, something like a fireman's helmet, and is provided in front with a flexible metallic hinge, which allows the helmet to be worn within certain limits of size by any person. As an additional safeguard, leather straps with pieces of elastic inserted are attached to a leather flap at the back of the cap, and buckled to two straps attached to each side of the mouthpiece, which thereby holds the mouthpiece firmly in position, and takes away all fear from the mind of the wearer of its being displaced, whilst it allows of free movement to the head. The expired breath is carried through an exhale valve J fitted in the mouthpiece, and which opens downwards or away from the mouth. This valve is of simple construction (figs. 677–679), and made from mica to avoid sticking, which might be caused by the saliva; and it prevents the impure air from returning to the lungs during the next inhalation of oxidised and purified air. The expired breath then passes by means of the flexible corrugated pipe K into the regenerator L, which is in the form of a rectangular tin box, fitted with baffle plates, and containing caustic potash, which absorbs the carbon dioxide exhaled by the lungs. The nitrogen thus freed from the carbon dioxide passes on to a pure reservoir bag N, which, being flexible, enlarges and diminishes in volume in direct correspondence with the requirements of the lungs; thence it travels by the inhalation pipe P and through the inhalation valve R to the mouthpiece, where it unites with a fresh supply of oxygen. The *modus operandi* may be explained as follows:—When inhalation takes place the valve R closes and J opens, the exhaled air passing up over the head and down through the regenerator into the breathing bag N. When inhalation takes place the valve J closes, R opens, and the purified air is drawn through the pipe P into the mouthpiece. But as soon as inhalation commences the lung-governed valve admits the necessary quantity of oxygen through the tube G into the mouthpiece, where it mingles with the inflowing current of purified air, and is then inspired into the lungs. The path traversed by the currents of gas through the apparatus can be traced by means of the arrow heads.

The apparatus, when fully charged, will allow of two to three hours' arduous work, but when engaged in very light work it will last for a period of five or six hours: The weight of the apparatus charged is 32 lbs., and it is attached to belt and shoulder straps, which distribute the weight over the shoulders and other parts of the body. Goggles can be attached to the helmet, and are used to protect the eyes when exploring in smoke.

Ærophor Apparatus.—This apparatus depends for its action on the fact that liquid air expands to about 800 times its own volume

when returning to its gaseous state, furnishing some 8 cub. ft. of oxygen from each pound of liquid. As the apparatus is chargeable with about 9 lbs. of liquid air, the supply of oxygen is ample, it being remembered that a man only absorbs about 1 cub. ft. per hour. At the rate of evaporation allowed by the apparatus, the charge of 9 lbs. of liquid air lasts some three hours, increasing in richness all the time. The liquid air used consists, in the beginning, of 60 per cent. of oxygen, but, owing to the nitrogen evaporating slightly

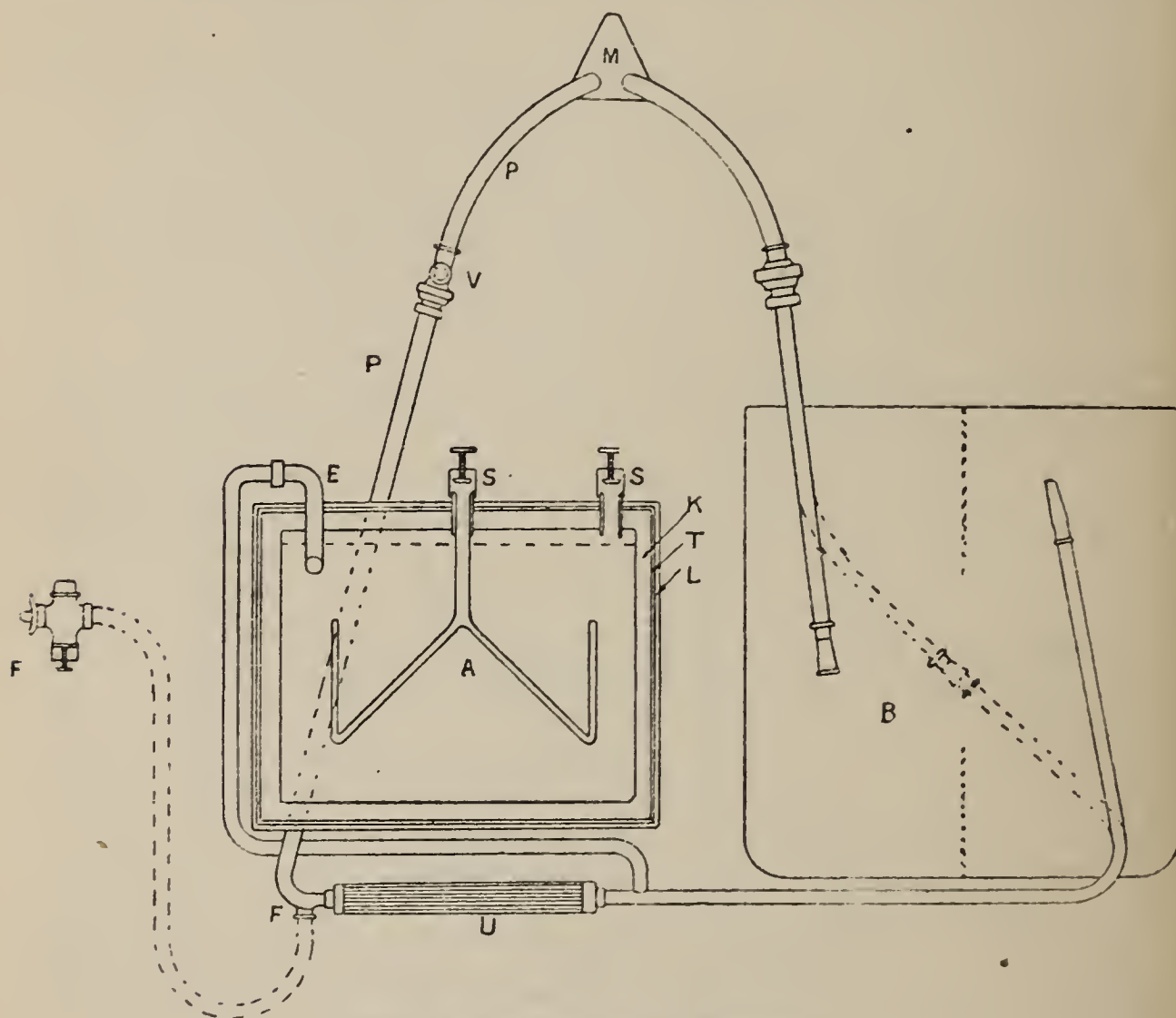


FIG. 680. — Diagram of Ærophor.

faster, gradually becomes richer as the charge in the breathing dress becomes less. The action is strictly opposite to that of breathing dresses with oxygen service, in which, during the latter stages of re-breathing the same air is chemically deprived of carbonic acid gas, there is an increasing percentage of nitrogen, and, therefore, it is alleged, increasing danger.

The apparatus consists of a knapsack A filled with asbestos wool, and insulated from the relatively hot outer atmosphere by a layer of special insulation material K, a layer of felt T, and an outer leather cover L. The liquid being poured into the knapsack at S is absorbed in the asbestos wool, and thereafter evaporates at a breathable rate

into a metal pipe E, where, being warmed up to a refreshing temperature, it is led to the breathing bag B, and so, by the nose and mouth mask M, to the wearer's lungs. It is exhaled again into the exhaust pipe P, a return to the breathing bag being made impossible. The pipe P leads the air past the exhaust valve V and a fitting F to a purifier U, where any balance of air which is required to maintain a uniform supply in the breathing bag may be freed from carbonic acid gas and dried. The fitting F supplies a means which no other breathing dress possesses. The wearer can, by its means, attach an easily carried face and mouthpiece, and with his surplus air, which still contains a very considerable percentage of oxygen, serve a person in distress over a period of emergency. The mask M is of a specially constructed composition of indiarubber and canvas, closely fitting over mouth and nose and securely strapped to a skull-cap, which, being made of asbestos, protects the head and neck from heat. It is claimed that the mask is specially comfortable and hygienic. The whole apparatus is mounted on a strong canvas jacket and is put on in a few seconds. For the storage of liquid air special metal vacuum vessels are used, of a capacity of 50 lbs. of liquid air each, reducing the evaporation to a minimum of no consequence.

In the *Ærophor* provision is made, in the event of the breathing bag being torn by accident, whereby pipes inside the bag may be coupled up direct from the pack in which the liquid air is stored to the mouthpiece. By this means, whilst breathing would be rendered more difficult, it could at least still be maintained, whereas without this coupling arrangement a man whose breathing bag became torn would run a great risk of speedy asphyxiation from foul air.

The arrangement whereby a pipe can be attached to the fitting valve F at the base of the pack so as to supply air to any partially gassed man who may be found in the mine, enables a rescuer to counteract the foul air from which the affected person is suffering, and to impart to him an immediate supply of air, which, although the rescuer himself has exhaled it, is still rich enough in oxygen to sustain life. This type, however, is being gradually superseded by the newer *Ærophor* apparatus, the patents of which are held by Mr Blackett. It is claimed for the *Ærophor* that it is much safer than other apparatus, being beyond the control and interference of the wearer; it is lighter and simpler and free from delicate parts; it requires less skill, knowledge, and practice in its use; it is more comfortable to the wearer, its air being cool and refreshing; it is far less costly in upkeep for a central station; and it possesses a special fitting to serve air to persons in distress—a feature which no other breathing dress possesses.

While this type of apparatus has advantages, there is the disadvantage that it would be difficult to keep an adequate supply of liquid air ready to hand unless a liquid air plant was erected at each station, which would entail very heavy expense.

Smoke Helmets.—Where the irrespirable zone is of a limited area, and where respirable air exists within a comparative short distance



FIG. 681.—Smoke Helmet with Telephonic Apparatus.

of the danger zone, smoke helmets may be used. The helmet is connected by tubing to a hand pump, by which ordinary air is delivered into the helmet. In the newer type of this apparatus a

telephone wire is carried between the person operating the air-pump and the wearer of the helmet, so that instructions can be transmitted between the two parties. The telephone receiver is fitted inside the helmet directly in front of the mouth, so that the wearer can speak into it without moving the helmet.

In certain circumstances these smoke helmets may prove very useful, such as in the case of an underground fire which has been localised; but if a long distance has to be traversed in a poisonous

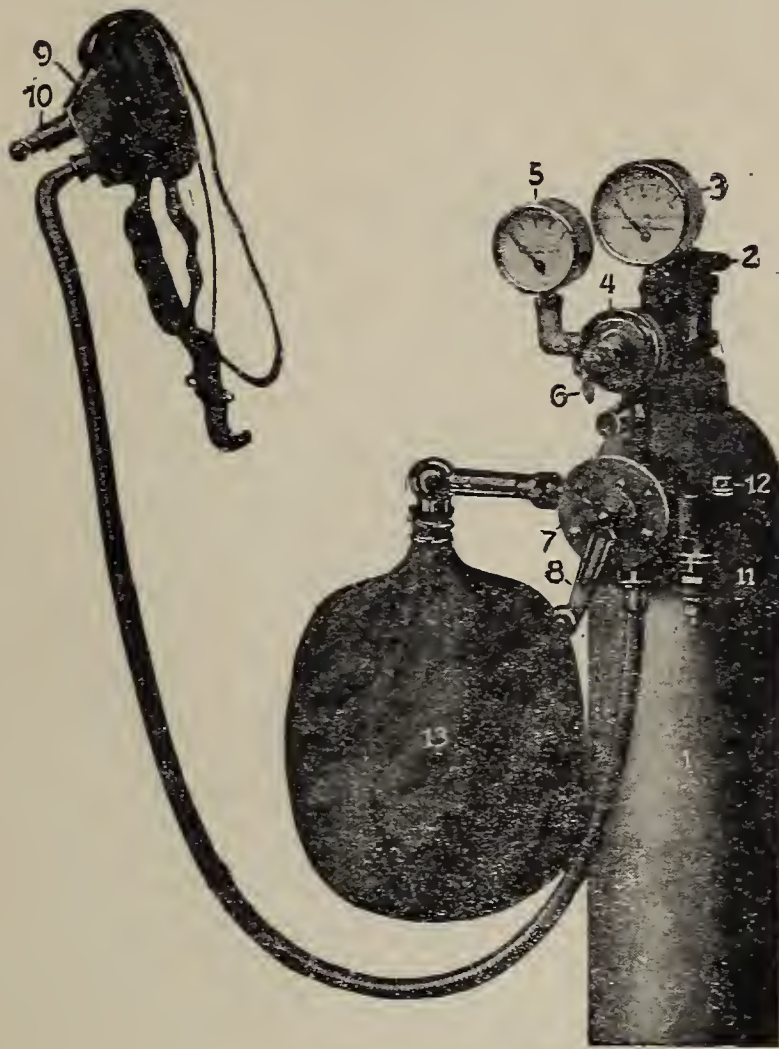


FIG. 682.

atmosphere, it would be difficult to use such an apparatus owing to the length of tubing that would be required.

Oxygen Reviving Apparatus.—In cases where in consequence of the inhalation of poisonous gases no breathing is perceptible in the asphyxiated person, that is to say, where natural respiration has almost ceased, the emptying of the lungs of the poisonous gases (exhalation), and the supplying with oxygen (inhalation) can only be effected by artificial means. In such cases an apparatus only capable of forcing oxygen into the lungs is not of much use, as it is of far greater importance to empty the lungs of the foul gas. The usual method of artificial respiration by moving the arms of the patient often fails, particularly in the case of gassing, because a man well experienced in the method is necessary, and it is not

feasible should the patient have received injuries to his arms or ribs.

The 'Meco' Oxygen Reviving Apparatus (Dr Bratt type) is used for this purpose. The foul gas is drawn off the lungs by the vacuum produced by means of an injector. The following is a brief description of the operation :—Valve (2) of the oxygen cylinder (1) is opened. The larger pressure gauge (3) then indicates the existing pressure from which the available supply of oxygen is ascertained, and care should be taken that the cylinder is kept fully charged and ready for use, that is, to a pressure of 120 atmospheres (about 1770 lbs. per sq. in.). At this pressure a cylinder of say 2 litres (about half a gallon) will last for a working period of about 20 minutes, which is quite sufficient in most cases.

By inserting a square cylinder the duration of the working period may of course be extended.

The reducing valve (4) is so adjusted that the pressure gauge (5) connected therewith indicates 1·0 atmosphere (about 15 lbs.). The adjustment of the pressure takes place by means of the winged screw (6) provided on the reducing valve (4). The pressure of 1·0 atmosphere has been found by practical tests to be the most suitable for the production of the required vacuum, and by the special arrangement of the ports the pressure of oxygen administered to the patient is also suitably adjusted.

When the artificial respiration is commenced, care should be taken in the first place that the pointer of the small handle (7) be set to 'artificial' and the pointer of the longer lever (9) to 'inhalation,' while the valve (10) on the mask should be closed by being turned down. The mask (9) is attached to the patient by the strap in such a manner that the mouth and nose are perfectly closed against access of outside air. By bending the wire in the pneumatic lining, the mask may be adapted to the shape of the face of the patient. This lining is inflated by means of the small stop-cock provided.

The valve on the face is so arranged that excessive pressure which might be dangerous to the patient is obviated. The point of the lever (8) is now moved to 'exhalation,' which causes the injector (11) to come into operation, producing the vacuum which is indicated by vacuum gauge (12), and emptying the lungs.

By moving the lever (8) at regular intervals, corresponding to the intervals of the natural breathing from inhalation to exhalation, the act of respiration is perfectly simulated.

To ascertain whether the patient is able to breathe without the help of apparatus, the valve on the mask (9) is opened by swinging it upwards, and the pointer of the small handle (7) is adjusted to indicate 'natural,' while the pointer of the longer lever (8) indicates 'inhalation.'

This allows the oxygen to flow into the bag (13), the movements of which indicate whether the patient can breathe unaided.

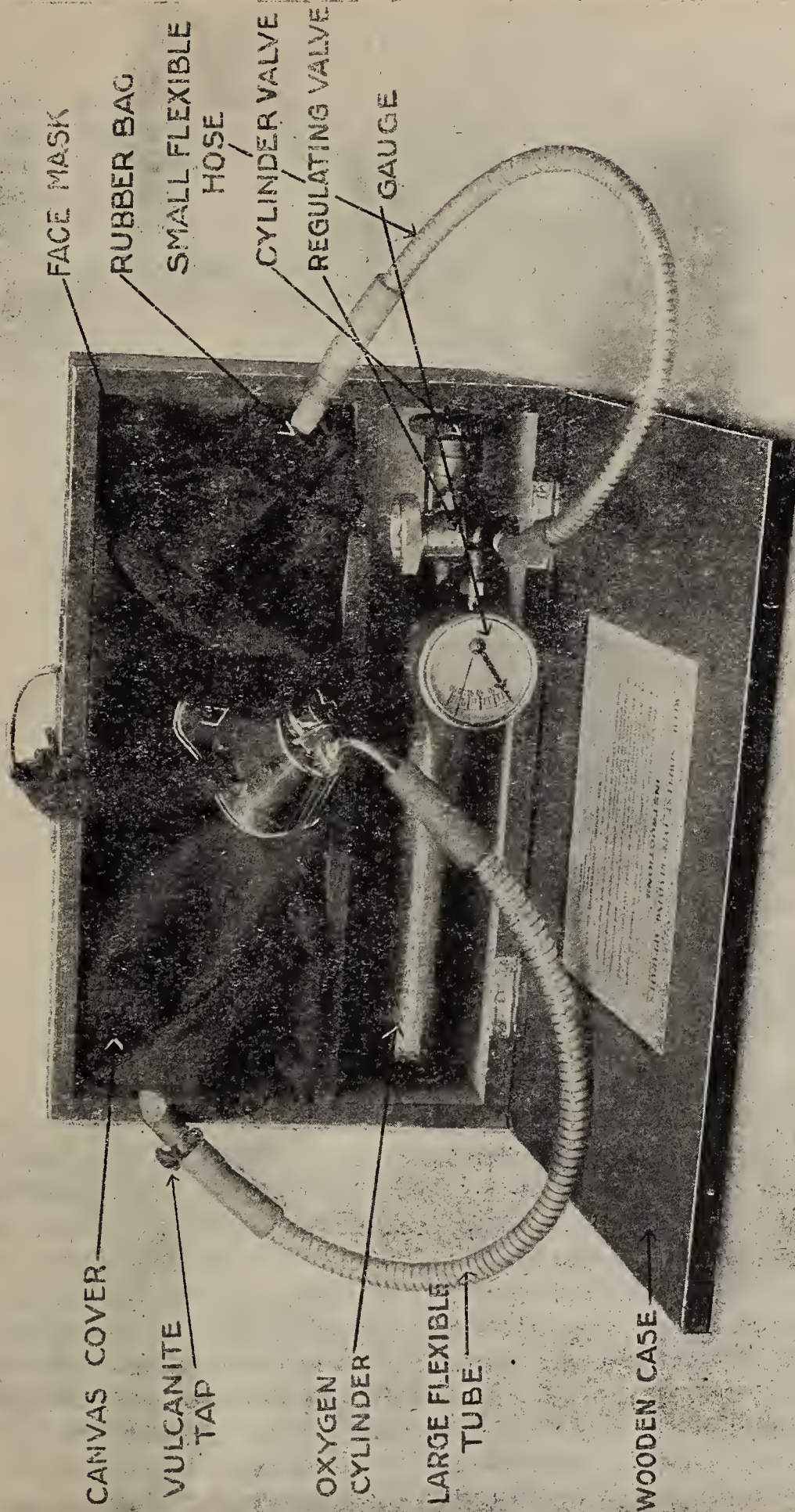


FIG. 683.

In slighter cases where the patient still breathes, to admit fresh oxygen without emptying the lungs, the mask (9) should be put on as described above, and the valve (10) connected therewith by moving the valve upwards.

The pointer of the longer lever (8) placed on 'inhalation' and that of the small handle (7) on 'natural,' by which means the admission of the oxygen takes place by inhalation through the bag as described.

The Mining Engineering Company also supply a simpler and cheaper type of reviving apparatus, which is now largely used at collieries. This apparatus is illustrated in fig. 683.

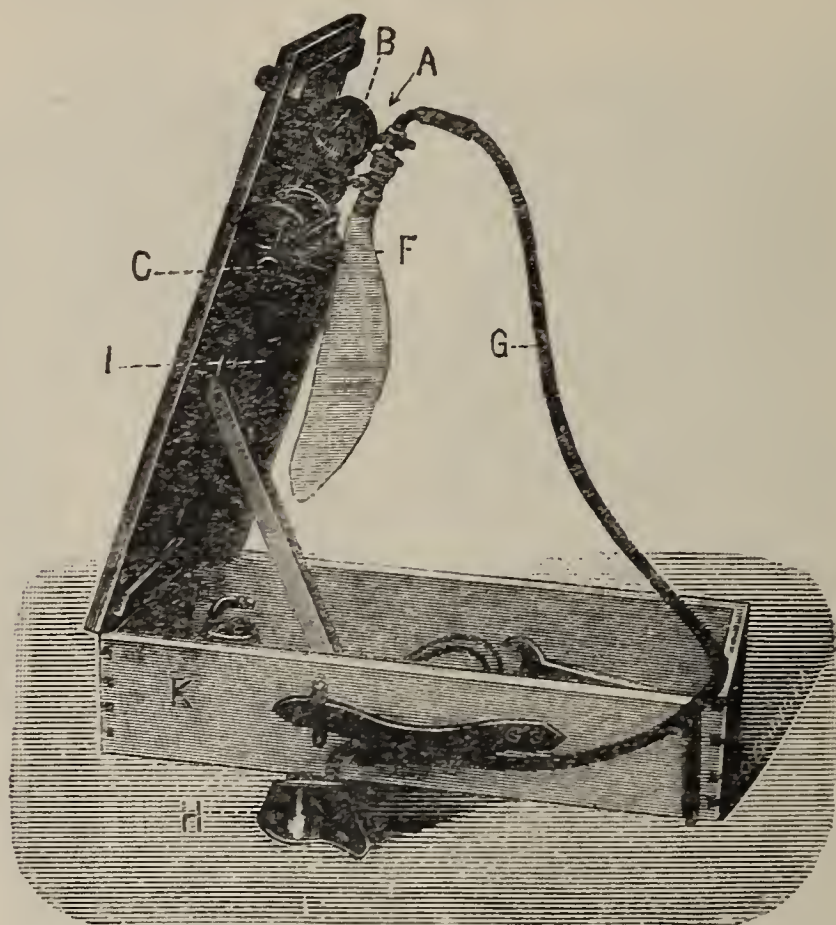


FIG. 684.—Draeger Reviving Apparatus.

The 'Meco' Simplest Type Reviving Apparatus.—To work the apparatus, gradually open the valve on the oxygen cylinder, the gauge then indicates the pressure in the cylinder. Now open the regulating screw with a turn to the left; this allows the oxygen to pass into the inhalation bag. Fix the mask which fits tightly over the mouth and nose of the patient, and the oxygen passes from the inhalation bag into the lungs. The flow of oxygen can be regulated as required from time to time by means of the regulating screw. Should the mask not have the desired effect it can be replaced by the two indiarubber fingers which are inserted into the nostrils so that oxygen passes through the nose into the lungs. Artificial respiration should be practised by moving the arms of the patient at the same time as the oxygen is being applied.

Draeger's Oxygen Reviving Apparatus.—The apparatus (fig. 684) is mounted inside the lid of a strong oak box, so as to come into position for use when the lid is opened. The apparatus comprises a cylinder (containing $4\frac{1}{2}$ cub. ft. of oxygen), a gauge showing the actual amount of oxygen in the cylinder, and an automatic inhaler with breathing bag, metal tube, two face-pieces, and one mouth-piece. A strap for attaching the mask to the head, so as to leave the hands of the operator free, and a spanner complete the equipment. After raising the lid, open valve of oxygen cylinder, and check quantity of oxygen from gauge, which when full should read 120 atmospheres. Fasten face mask by means of head-strap if necessary, then raise lever C upwards, when automatically a quantity of oxygen will flow into the rubber bag at the rate of 5 pints per minute, from whence it is conveyed through the flexible tube to the patient. If animation cannot be restored, artificial respiration can with advantage be resorted to, in conjunction with oxygen inhalation.

The Pulmotor Resuscitation Appliance.—The Pulmotor automatic oxygen reviving apparatus consists of a pair of concertina-like bellows B (figs. 685 and 686), alternately inflated and collapsed by the pressure of the flow of oxygen and air, and thereby operating a reversing valve. The oxygen is supplied from the oxygen cylinder C by turning the valve V through the reducing valve D to the suction nozzle S, which at the same time draws in a suitably large volume of air. This nozzle supplies the power for inflating and deflating the lungs. The nose mask, which is held in position by a bow, can be pressed down until the mask fits tightly against the nose, where it is retained automatically. The mask is connected to the Pulmotor by two flexible pipes, of which one serves for inflating the lungs with fresh air, whilst the other conveys the vitiated air for discharge through the apparatus. When applied to a man the lungs are first filled with air until the normal pressure of 8 in. water column is reached. This pressure acts on the bellows and inflates them, thereby drawing the reversing valve in the chamber L from the position shown in fig. 685 to that of fig. 686. In this position of the valve the current produced by the nozzle S draws the air out of the lungs under a negative pressure of $4\frac{1}{2}$ in. water column, and discharges it into the open air. As soon as the lungs are emptied the bellows collapse and return the reversing valve into position for re-inflation; and this cycle of operations is repeated automatically as long as required. The natural respiration rhythm is exactly imitated, the rhythm depending on the capacity of the lungs. Moreover, the same rhythm is maintained regularly throughout, which is very important in connection with the resuscitating of unconscious men. Artificial respiration, as usually performed, is very exhausting to the operator, who, in consequence, becomes slow and irregular in his movements, and may be tempted to abandon his efforts too early through sheer physical exhaustion. The Pulmotor, however, never grows tired,

and it possesses still another feature of importance in that it mixes a proportion of oxygen with the air introduced into the lungs. Ordinary air contains oxygen in the proportion of 1 to 5, sufficient for a healthy subject, by whom it is absorbed by the hæmatin of the blood, which conveys it to the tissues, where it is again yielded up. In cases of gassing, however, we have to deal with an abnormal condition, in which the hæmatin has formed wholly or partially a staple compound, and incapable of absorbing oxygen. When the change has only partially occurred the introduction of fresh air alone may

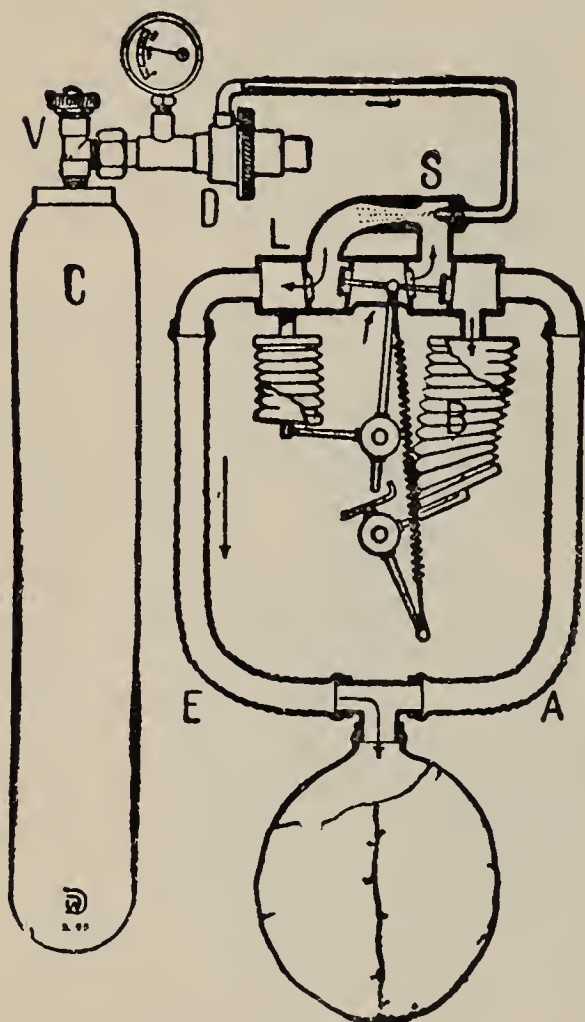


FIG. 685.—Inflating.

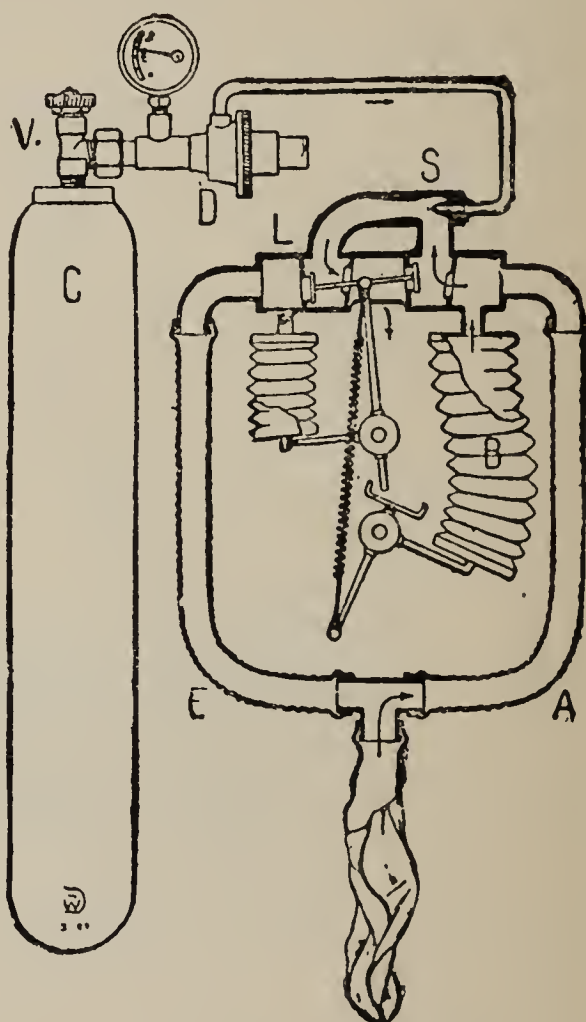


FIG. 686.—Exhausting.

The Pulmotor Resuscitation Appliance.

bring about the desired result, but in more advanced cases fresh air alone would be useless. In these cases the plasma of the blood must be saturated with oxygen, and thus conveyed to the tissues until new hæmatin is formed, when the normal course is resumed. Now the tension of oxygen in air is not sufficient for this purpose, and in order to bring about the desired result an extra supply of this gas must be introduced, as in the Pulmotor. The administration of oxygen combined with artificial respiration forms an ideal method, and there can be no doubt that in the future lives will be saved which under the older methods would have been sacrificed. Where fractures of the arm, bones, or ribs, or abdominal injuries

exist, it may be dangerous, if not impossible, to perform artificial respiration by ordinary means. Here such an apparatus as the Pulmotor may not only save life, but also be the means of preventing serious injury and suffering. After the injured person has recovered consciousness, or in cases of a person only partially overcome, a supply of oxygen can be given to them by means of an inhalation apparatus, which is mounted on the inside of the lid of the wooden box in which the Pulmotor is fitted. This apparatus delivers oxygen at the rate of 170 cub. in. per minute.

INSTRUCTIONS FOR THE EQUIPMENT AND MAINTENANCE OF RESCUE CORPS.*

OBJECTS OF RESCUE CORPS.

§ 1. Rescue corps are established for the purpose of preserving life and property in cases of explosion, pit fires, or other occurrences necessitating the intervention of trained workers.

COMPOSITION AND NUMBERS OF A CORPS.

§ 2. A rescue corps should be established at each separately worked pit.

§ 3. A corps should consist of the following members: a superintendent (head deputy, fireman, etc.); two or more leaders (officials engaged at the colliery), one of them acting as permanent deputy to the superintendent; eight or more rank and file (pit officials, overseers, mechanics, firemen, head shot firers, waterers or experienced hewers); and a man to look after the appliances (mechanic or smith employed at bank).

(For the sake of uniformity in rescue work, it is desirable that the rescue corps of several adjacent collieries under the same ownership should be placed under the command of one superintendent. This is more particularly the case when the breathing appliances are kept at only one of the pits. In such event a single attendant will be sufficient to look after the apparatus. It is desirable that all available officials should be trained in rescue work.)

§ 4. All the appliances intended for use in rescue work, working in irrespirable gases, or extinguishing and shutting off pit fires, must be stored, under supervision, in a special store. One key of this store is to be carried by the attendant, and other keys kept in the manager's office and timekeeper's office respectively.

§ 5. Practice rooms must be provided for the corps.

* The author is indebted to Mr R. Jacobson for kindly granting permission to reproduce these instructions.

ENLISTMENT.

§ 6. (1) Membership of a rescue corps must be confined to miners who are well able to speak and write the language of the country, have been engaged underground for at least three years, and are between the ages of 21 and 49.

(Only men of determination, sobriety, and quiet habits are suitable; rashness and a tendency to over indulgence in alcoholic liquors being a total bar to enlistment. The members of the rescue corps should live as near the pit as possible.)

(2) Membership is voluntary. Applications for enlistment must be made to the superintendent, who will examine applicants as to their fitness for membership.

(3) Before they can be accepted applicants must produce a medical certificate to the effect that they are fit to undergo the heavy work they will be called upon to perform as members of a rescue corps. (The cost of such medical certificate will be borne by the colliery.)

(4) Applicants will not be admitted into the corps until they have gone through the course of training specified in § 7, and passed the prescribed tests.

(5) The act of admission consists in the applicant's name being inscribed in the corps roll; and the applicant must, in the presence of the superintendent, sign an undertaking to carry out the conditions of these instructions.

TRAINING.

§ 7. (1) The training is to be carried out by the superintendent in person, or by a corps leader under his supervision. It is divided into two sections: theoretical and practical.

(2) The theoretical training must precede the practical course. It comprises instruction in the phenomenon of breathing in the human subject, and in the arrangement, working, treatment, and use of the appliances intended for use in rescue work. The men must also be made acquainted with such of the present instructions as it is desirable for them to know.

(3) In the practical course the applicant must become accustomed to work with the breathing apparatus in irrespirable gases, performing work with a work meter, discharge practical mining tasks, and move about in the practice room.

(4) When the course is ended, the applicant will be put through an examination by the superintendent, consisting in working in an atmosphere of smoke for two hours on end, whilst wearing a portable breathing apparatus. During this test he must do work to the extent of at least 72,330 ft.-lbs., as registered by the work meter, and in the intervals of work must traverse the practice room. Interruption of the exercises implies failure to pass the test. The

applicant must also show himself, in a *viva voce* examination, to possess the necessary acquaintance with the arrangement and use of the rescue appliances.

RESIGNATION OF MEMBERSHIP.

§ 8. Members who are no longer able to satisfy the requirements expected from candidates must resign from the corps.

DUTIES OF MEMBERS OF A RESCUE CORPS.

§ 9. (1) The members of a rescue corps must practise for two hours at a time at least once a quarter, the exercise to be continuous in order to count as a practice. The practices shall be carried on in accordance with the detailed instructions and under the supervision of the superintendent, either in an atmosphere of smoke in the practice room, or in the form of special tasks—such as may arise in cases of accident—performed in the pit.

(2) In addition to these four practices with portable rescue apparatus per annum, there must be one exercise, of half an hour's duration, with air pump (flexible pipe) apparatus every year.

§ 10. The leaders and superintendent must also take part in the exercises prescribed in § 9, and, in addition, must supervise the practices of the corps in turn. They must also attend the lecture specified in § 16.

§ 11. The members of the rescue corps must report at once for duty in the event of any serious accident, explosion, pit fire, etc., occurring at their own or any adjoining pit which their pit is called upon for aid in accordance with either the general scheme of rescue operations or the instructions of the central body controlling rescue work.

SPECIAL DUTIES OF LEADERS AND MEN IN CHARGE OF APPLIANCES.

The following special duties are incumbent on the superintendent :

§ 12. (1) The training and instruction of the rescue corps, and its maintenance at full strength. When a member leaves, his place must be filled within a month.

(2) Supervising the training of members of the corps, and personally conducting the entrance examination.

(3) Keeping the roll of members of the corps.

(4) Supervising the store of equipment and seeing to the replacement of same.

(5) Drawing up a written report to the management in all cases where the rescue corps or appliances have been employed in connection with an accident. Drawing up an annual report to the administration by 15th February in each year. These reports must

give any special occurrences that may have happened, and any important observations made with regard to the appliances. The following details must accompany the annual reports:—

- (a) List of the members of the corps.
- (b) List of the breathing apparatus in stock (specifying the kind of apparatus) and of any other appliances for use in connection with rescue work, so far as these are at present in the store.
- (c) A statement of the number and nature of the practices carried out.
- (d) A statement of the expenses in connection with rescue work during the year.
- (e) Particulars of the dates of practices and lectures during the year under report. (No alteration may be made in this respect, except by special permission.)
- (6) He is responsible for the carrying out of the prescribed practices by the rescue corps, and for the delivery of the lecture on rescue work as prescribed by § 16.

(7) The superintendent must always have a ventilation plan at disposal, and must keep himself informed of the actual condition of the ventilation and ventilating appliances. The ventilation plan must be accurately marked with all the air doors, fire doors, or fire dams, and the position of fire extinguishing appliances.

(8) The superintendent is responsible for the members of the corps being distributed among the several shifts in such a manner that there will at all times be at least one leader and three men at bank or near the shaft underground. (With this object he must investigate the distribution of the corps members among the shifts at least once a month.)

(9) The superintendent is also responsible for the observance of the following regulations:—

- (a) Practices with the portable breathing apparatus must be carried on at least once in six weeks, so that the appliances may not be left unused beyond a certain time.
- (b) A record must be kept, and entered up to date, of the names, occupations, tally numbers, districts, positions in the corps, and addresses of all members of the rescue corps; and a copy of this record must be hung up for inspection in each of the following places:
 - 1. The manager's office.
 - 2. The deputies' room.
 - 3. The timekeeper's office.
 - 4. The room where the rescue appliances are stored.
- (c) So much of the rescue scheme, drawn up by the central body, as applies to the colliery, must also be hung up in the same places.

§ 13. (1) The leaders have—in accordance with these regulations

and with the instructions issued by the superintendent—to train the candidates for membership, and take part in the entrance examinations and the practices of the corps.

(2) They must keep the practice book, unless this is done by the superintendent.

§ 14. (1) The attendant in charge of the appliances must be thoroughly acquainted with the methods of using, testing, storing, cleaning, and keeping in good condition all the appliances and materials used.

(2) He is responsible for seeing that all the equipment of the store practice room are available at all times in the prescribed number or amount, and perfectly fit for use; and must keep a stock book. The stock must be verified once a month, the date being entered in the stock book with a note of any parts that have been found defective.

DUTIES OF OFFICIALS WHO ARE NOT MEMBERS OF THE RESCUE CORPS.

§ 15. The manager is responsible for arrangements being made at the pit to summon the rescue corps quickly on the occurrence of an accident.

§ 16. Once a year the officials of the colliery are to attend a lecture, given by the superintendent or manager, dealing with the whole subject of rescue work in mines, the details of the present Instructions, and the measures adopted at the colliery for preserving the pithead buildings in case of fire or after explosions.

THE RESCUE CORPS IN ACTUAL EMERGENCIES.

§ 17. As soon as the members of a rescue corps are notified, in any way, of the need for the intervention of their corps, whether at their own colliery or another, they must repair as quickly as possible to the store where the rescue appliances are kept, and place themselves at disposal for duty.

§ 18. Each set of breathing apparatus must be tested for its fitness for use, before being employed.

§ 19. The rescue operations will be performed in accordance with the directions of the superintendent or corps leader.

§ 20. (1) Immediately on the occurrence of a pit accident necessitating the intervention of the rescue corps, the manager or his representative must see that notification of same is forwarded to the head rescue station.

(2) When it seems probable that the pit's own rescue corps will not be sufficient for the task in hand, notification must be sent to the collieries that are in the first-call zone of the general scheme of operations. The head station for rescue work will, if necessary, see that additional aid is summoned.

PROCEDURE ON IRRESPIRABLE GASES.

§ 21. The actual leader of a rescue detachment must take care that the men do not run into unnecessary danger.

§ 22. Spaces that are filled with irrespirable gases must, in the first instance, be entered only by a full troop (leader and three to four men).

§ 23. During the time the troop is actively engaged, the leader must note, as continuously as possible, the reading of the gauge indicating the length of time the breathing apparatus worn by his men may still be used. He must order the retreat at such a time that the apparatus with the lowest reading will suffice for a period at least twice as long as the period covered by the retreat.

§ 24. When rooms, etc., filled with irrespirable gases are entered, a post for rendering aid and equipped with breathing apparatus and trained rescuers must be established in the immediate vicinity.

CHAPTER XV.

SURFACE ARRANGEMENTS, COAL CLEANING, ETC.

Siding Accommodation.—For the handling of a large daily output of coal, plenty of siding accommodation both for loaded and empty waggon should be provided. Many large collieries provide sidings for 200 or 300 loaded trucks, or for one day's output. This will be all the more necessary if the colliery is situated at a considerable distance from the main line, and a clearance is effected only once or twice daily. With the modern practice of separating the coal into

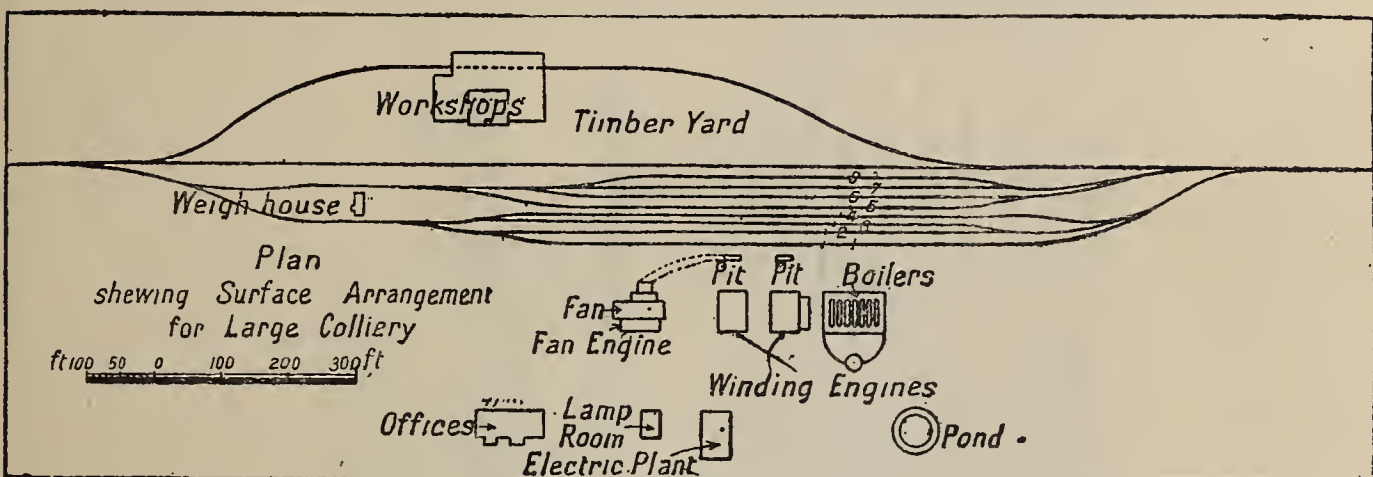


FIG. 687.—Plan of Surface Arrangements.

many different classes for the market, the number of sidings requires to be greater than formerly, when it was customary to simply separate the dross from the round coal. From various causes the ground at disposal may be limited in area for sidings, in which case it is impossible to provide large storage lyes for loaded waggon. As to the general arrangement of surface buildings, such as engine-houses, boilers, screens, etc., the area and disposition of the ground enclosed will largely determine the matter for each individual colliery. A general plan of surface arrangements is shown in figs. 687 and 688, taken from those of large collieries raising from 1200 to 2500 tons of coal daily.

Inclination of Sidings.—The proper inclination for colliery sidings is an important matter. From the author's own experience he has

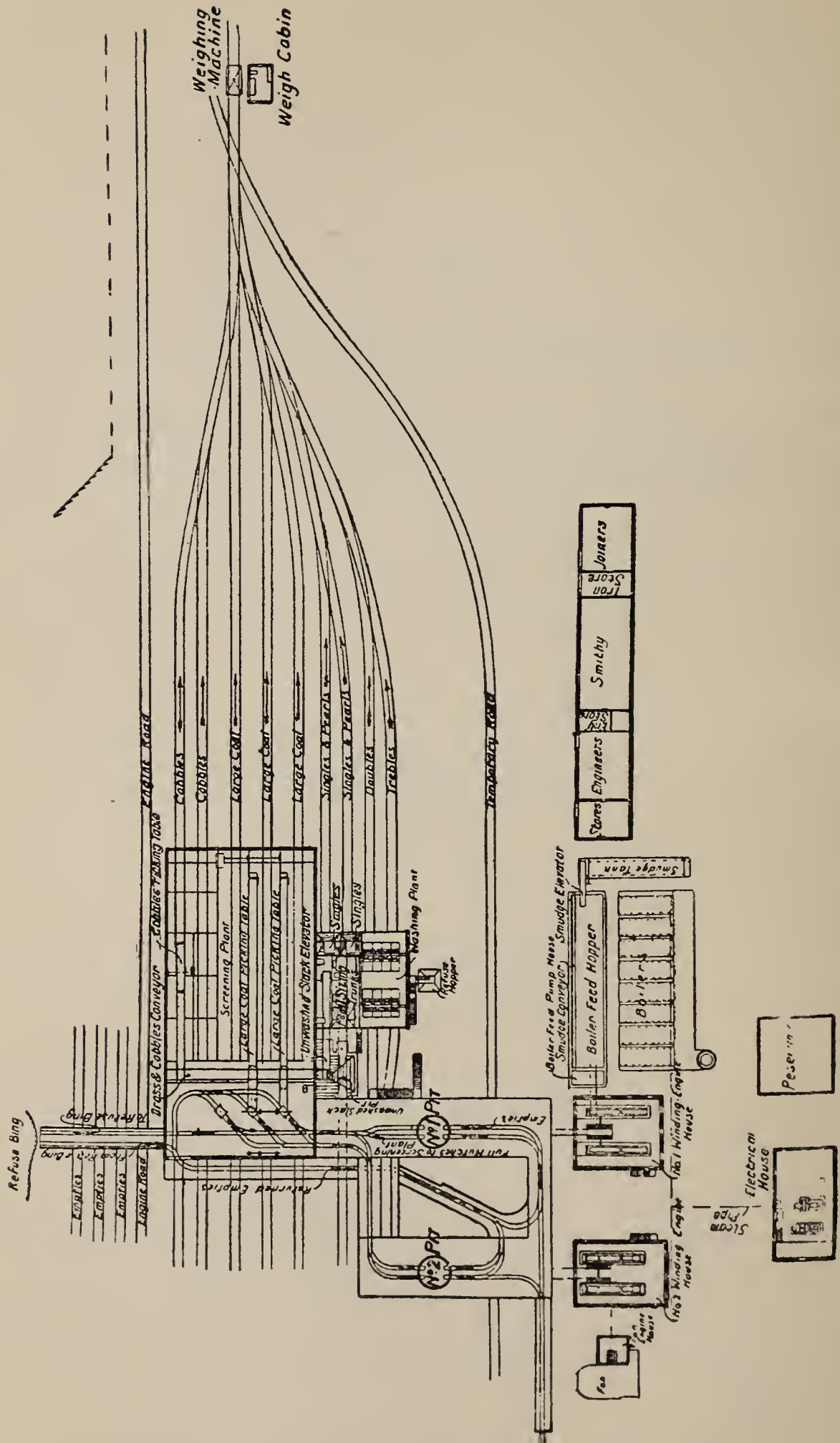
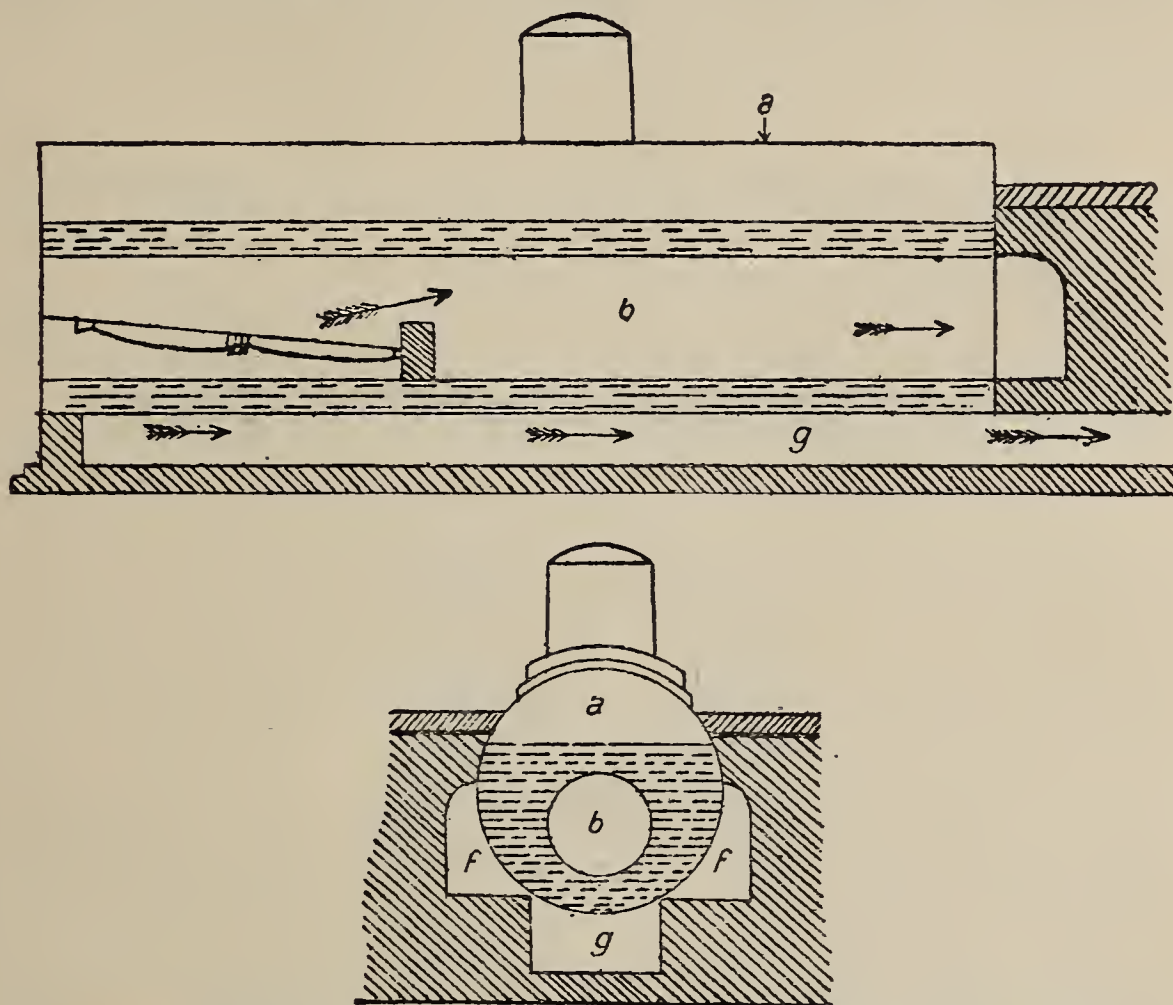


FIG. 688.

found that the best gradient behind the screens is about 1 in 70 for the empty waggons, with the inclination towards the shaft, while 1 in 60 to 66 is best suited for the loaded waggons in front of the screens. With these gradients the waggons should move freely in all weathers.

Boilers.—The boilers used at collieries may be of either the ordinary egg-end, or of the Lancashire or Cornish types. Where rapid steaming and high pressures are required, the Lancashire boiler is the best to use, and also, as a rule, more economical, convenient, and durable. If large grate area is required, low pressures, and low first



FIGS. 689 and 690.—Longitudinal and Cross-Sections of Cornish Boiler.

cost, the egg-end type of boiler may be found better suited, but this type is now rarely used at collieries.

Cornish Boilers.—Cornish boilers consist of a cylindrical shell *a*, with flat ends, and having near the bottom a smaller shell or tube *b* (figs. 689, 690), which passes through the larger one and forms the furnace. The products of combustion pass from the furnace to the end of the tube, and return by the two side flues *f f* to the front of the boiler, pass into the bottom flue *g*, and so reach the chimney. By this arrangement the gases are reduced in temperature before coming into contact with the bottom of the boiler, where all the sediment collects, and there is therefore no danger of burning the plates on the under side of the boiler.

ought to be supplied with a safety valve, steam-gauge, glass water-gauge, stop valve, water float, sludge, steam and feed water pipes, blow-off cock and pipe, damper, etc.

Accidents to Boilers.—Accidents to boilers are generally due to the following causes, viz., defective water supply, corrosion and incrustation of shell, or defective safety valves. Accidents also happen from plates getting worn thin, or rivets becoming fractured.

In the case of a defective water supply, if the water-level becomes dangerously low in the boiler the fire should on no account be withdrawn, as this procedure may accelerate an explosion by the sudden cooling of the plates; the fire should be damped with small coal or ashes, the damper put down, and the boiler allowed to cool gradually before water is put in.

Incrustation in steam boilers generally arises from the presence of salts in the feed water, such as sulphate or carbonate of calcium. If the impurity is found to consist almost entirely of carbonate of calcium, the feed water may be treated by the addition of caustic lime or milk of lime, or by what is known as Clark's process.

When carbonate and sulphate of lime are both known to be present, the feed water may be treated with caustic soda or soda ash. With both of these methods, considerable expense is involved, as large tanks are required to hold six to eight hours' water supply; and instead of treating the feed water in any way, it is allowed to enter the boiler and the lime precipitated in the boiler itself, and the sediment blown off frequently.

Corrosion of the boiler plates is due to either chemical or electrical action. The greatest cause of corrosion is probably due to the presence of animal or vegetable oils in the feed water. These oils are composed of glycerine chemically united with acids, and when heated separate into free glycerine and free acids. This decomposition takes place at a temperature of about 220° F. The free glycerine is practically harmless, but the free acids attack the boiler plates and eat them away. Oils for feed pumps should, therefore, be carefully selected, and only those used which do not corrode.

Chloride of magnesium is one of the most corrosive compounds found in water. When heated it sets free hydrochloric acid, which attacks the plates. When chloride of magnesium is present in the water it should be treated with caustic soda. Pitting is a form of corrosion that is found in boilers, and is said to be due to air in the feed water, galvanic action, or impurities, such as slag, etc., in the boiler plates. To prevent galvanic action zinc plates suspended by straps of iron are used, or painting the interior of the boiler with zinc oxide is said to have a good effect.

Boilers ought to be carefully inspected, internally and externally, and reported on every three months by a competent boiler inspector.

Evaporative Power of Boilers.—The term 'nominal horse power' is now obsolete, and most boilermakers state in their specifications

how much water their boilers are capable of evaporating into steam per hour. The power of a boiler, therefore, depends on the quantity of water that can be evaporated in it, and this in turn is directly dependent on the quantity of fuel which can be efficiently burned on the firegrate. With good natural draught, as much as 24 lbs. of coal per sq. ft. of grate area may be burned economically per hour, and with coal of a fair quality an evaporation of $7\frac{1}{2}$ lbs. of water from an initial temperature of 100° F., and $8\frac{1}{2}$ lbs. from 250° F. may be reckoned on. In estimating the horse power of boilers it is necessary to take into consideration the weight of steam consumed by the engine, as this varies within wide limits with the type of engine adopted. Take, as example, a Lancashire boiler 28 ft. $\times$ 7 ft., with a grate area of 34 sq. ft. and a coal consumption of 25 lbs. per square foot per hour, and an evaporation of $8\frac{1}{2}$ lbs. per lb. of coal, the total evaporation of this boiler would be $34 \times 25 \times 8.5 = 7225$ lbs. If the engine is non-condensing, and using, say, 30 lbs. of steam per I.H.P. per hour, then I.H.P. of boiler = $\frac{7225}{30} = 240.83$. Theo-

retically 1 lb. of coal should evaporate from 12 to 16 lbs. of water, at 212° F., the actual amount depending on the quality of the coal. In practice a Lancashire boiler will evaporate 8 to $10\frac{1}{2}$ lbs. of water at 212° F. per lb. of coal burned, and a Cornish boiler 7 to 9 lbs. of water per lb. of coal.

The weight of water which can be evaporated per hour for any given size of engine may be found by the following formula:

$$W = \frac{D^2 \times .7854 \times L \times 2 \times R \times 60 \times a}{1728}.$$

Where W equals the weight of water to be evaporated per hour; D the diameter of cylinder in inches; L the length of stroke in inches; R the revolutions of crank per minute, and a the value of 1 cu. ft. of steam at given pressure.

Example.—In an engine with a cylinder 20 in. diameter, a 24 in. stroke, and making eighty revolutions per minute, what amount of water would require to be evaporated per hour if the steam pressure is 65 lbs. per sq. in.?

Here the value of $a = .1538$ lb. at 65 lbs. absolute pressure.

$$\therefore W = \frac{20^2 \times .7854 \times 24 \times 2 \times 80 \times 60 \times .1538}{1728} = 6442.34 \text{ lbs.}$$

Efficiency of Boilers.—The measure of efficiency of a boiler is the number of pounds of water evaporated per pound of fuel burned, the evaporation being reduced to the standard equivalent evaporation from feed water at a temperature of 212° F. into steam at the same temperature. The measure of capacity of a boiler is the amount of 'boiler horse power' developed, a horse power being defined as the evaporation of 30 lbs. of water per hour from 100° F. into steam at 70 lbs. pressure per sq. in., or $34\frac{1}{2}$ lbs. per hour from and at 212° F.

Rate of Combustion.—The rate of combustion of fuel in the furnaces of steam boilers is usually expressed in pounds of coal consumed per sq. ft. of grate surface per hour. In ordinary boilers the rate of combustion is from 15 to 20 lbs. of coal per sq. ft. of grate per hour.

Strength of Boilers.—The strength of boilers depends on their construction and upon the material of which they are made. Boilers are now generally made of best mild steel, with the exception of the manhole mouth-piece and longitudinal bolt-stays which are usually of wrought-iron.

The tenacity of wrought-iron is given at 50000 lbs. per sq. in., and that of steel for boiler plate 80000 to 90000 lbs. per sq. in., and a factor of safety of 6 to 8 is usually allowed.

Let t represent the required thickness of plate in in.; p the working pressure in lbs. per sq. in.; d the diameter of boiler in in.; and M the factor of safety.

Then for wrought-iron boilers $t = \frac{pd}{50000} \times M$ or $p = \frac{t \times 50000}{d \times M}$ (single riveted).

And $t = \frac{pd}{60000} \times M$ (double riveted).

For steel boilers $t = \frac{pd}{70000} \times M$ or $p = \frac{t \times 70000}{d \times M}$ (single riveted).

And $t = \frac{pd}{90000} \times M$ (double riveted).

Example.—What thickness of double riveted boiler plate would be required for a steel boiler $7\frac{1}{2}$ ft. diameter to work at a pressure of 80 lbs. per sq. in.?

$$\text{Here } t = \frac{80 \times 7.5 \times 12}{90000} \times 8 = \frac{6 \times 8}{75} = .64 \text{ in. or about } \frac{1}{16} \text{ of an in.}$$

Example.—What would be the bursting pressure of a single riveted steel boiler 7 ft. diameter, with plates $\frac{5}{8}$ of an in. thick?

$$\text{Here } p = \frac{t \times 70000}{d} = \frac{\frac{5}{8} \times 70000}{84} = 520.8 \text{ lbs. per sq. in.}$$

Low and Bevis give the following formula for the safe working pressure:

$$p = \frac{8000 \times 2t}{d}; \text{ where } t = \text{thickness in in., and } d = \text{diameter of shell in in.}$$

For the strength of boiler tubes, plain iron, Fairbairn gives the formula:

$$P = \frac{806300 \times t^{2.19}}{L \times D}; \text{ where } P = \text{collapsing pressure per sq. in.; } L = \text{length of tube}$$

in ft., and D = diameter of tube in in.

Where great accuracy is not required t^2 may be substituted for $t^{2.19}$.

Number of Boilers required.—The number of boilers required at a colliery will depend upon the class of boiler used, the kind of fuel supplied, and the number of engines to be simultaneously supplied with steam.

A good Lancashire boiler, consuming about 7 cwts. of good coal per hour, should suffice for an engine of about 200 horse power, but at collieries where the fuel supplied for firing is often of a very inferior quality, about 180 horse power would be furnished per boiler, or even less. One spare boiler should be allowed for every four or five in use.

An approximate method of estimating the boiler requirements is to allow one 28 ft. $\times$ 7 ft. Lancashire boiler for every 13000 cub. ft. of steam required per hour.

Boiler Chimneys.—Chimneys or stacks may be either square, octagonal, or circular, the circular form being the best shape, but at the same time more expensive to build, as the bricks require to be specially made for each section of the chimney. Plenty of area should be allowed in chimneys, as it is useless to make large flues at the boilers and not to have a corresponding area in the chimney. The area at the top should be about the same as at the boiler flues.

The following formula may be employed for chimneys, taking the consumption of fuel at 21 lbs. of coal per sq. ft. of fire grate per hour as a basis:— $A = \frac{1.5G}{\sqrt{H}}$ or $\sqrt{H} = \frac{1.5G}{A}$; where A = area of chimney in sq. ft. at top or smallest part; G = area of fire grate in sq. ft.; and H = height of chimney in ft. above fire bar level.

If the coal consumed is likely to be lower than 21 lbs. per sq. ft. of fire grate per hour, then the formula $A = \frac{.07W}{\sqrt{H}}$ may be employed, W being the actual weight of fuel consumed in lbs. per hour.

The height of a chimney will depend on the number of boilers and the total coal consumed, and is often determined by local considerations, especially if the colliery is situated near a town, as the authorities then often specify a certain minimum height for chimneys. Chimneys, as a rule, should not be much less than 90 or 100 ft. in height.

* The following table gives the height of chimney according to the weight of coal consumed:

Weight of Coal consumed per hour.	Height of Chimney.	Weight of Coal consumed per hour.	Height of Chimney.
100 lbs. and under	60 ft.	3000 lbs. and under	160 ft.
500 " "	100 "	4000 " "	180 "
1000 " "	120 "	5000 " and upwards	200 "
2000 " "	140 "		

The chimney should be built up on good solid foundations, rock, if possible, with a solid bed of concrete 2 to 3 ft. thick for a base, the brickwork being built up vertically for a certain distance and gradually tapered off in $4\frac{1}{2}$ in. courses to the required size. The thickness of brickwork for chimneys depends on their height, but except for very small chimneys it should never be less than 9 in. at the top. The outside batter should be from $\frac{3}{16}$ to $\frac{1}{4}$ in. per rising foot, or about 1 in 56.

Banking-Out.—Probably few things have undergone more changes in recent years than the banking-out arrangements at collieries. It

was, and is still at a large number of collieries, the general practice to cover the landing-stage with flat-sheets or iron plates on to which the tubs could be drawn when taken from the cage, and turned in any direction required. No doubt this system has its advantages, but it requires a great deal of hand labour that might otherwise be dispensed with. When new collieries are being laid out, flat-sheets are now often dispensed with altogether, and lines of rails laid up to the cage.

Fig. 693 shows a banking arrangement on this principle, laid out for dealing with a large daily output.

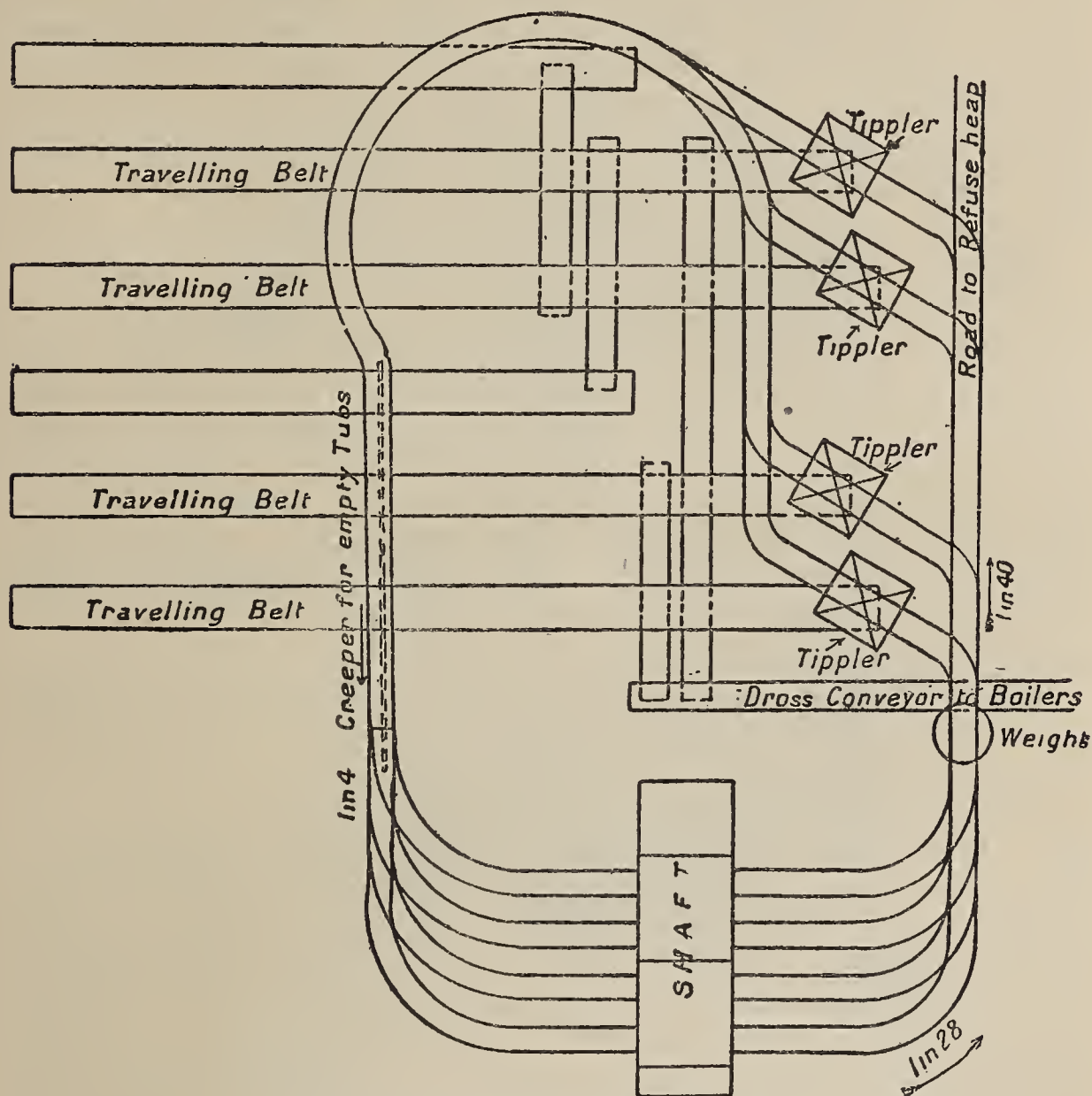


FIG. 693.—Banking Arrangement.

The successful working of such arrangements has been greatly assisted by the introduction of tipplers and the creeping chain. As the lines of rails from the cage on the full side must have an inclination towards the screens, to allow the tubs to run freely, there must be a corresponding inclination against the tubs on the empty lines of rails, which would involve labour to bring the tubs to the cage if

done by hand. To avoid this, the creeper chain is used. The creeper consists of an endless chain made on much the same principle as a travelling belt. At set distances apart are fixed upright pieces or 'fingers' for catching the tubs and carrying them forward to the desired position. The chain works round octagonal wheels at each end, and is supported throughout its length by rollers fixed in standards. On the side of the pit-head where the full tubs are taken off, the rails, as already stated, are laid at an incline in favour of the tub. The tubs are first taken over the weighing-machine, and then by their own weight run into the revolving tipplers, from which, when empty, they pass to the creeper and are hauled up to the level of the cages again. Instead of a 'creeper' a steam or hydraulic hoist is sometimes used to raise the empty tubs to the desired level. The disadvantage of such an apparatus is that it is not self-acting, and is therefore more costly in working. The hoists themselves are also expensive to erect, especially if the height is great, and the raising piston must of necessity be long in proportion.

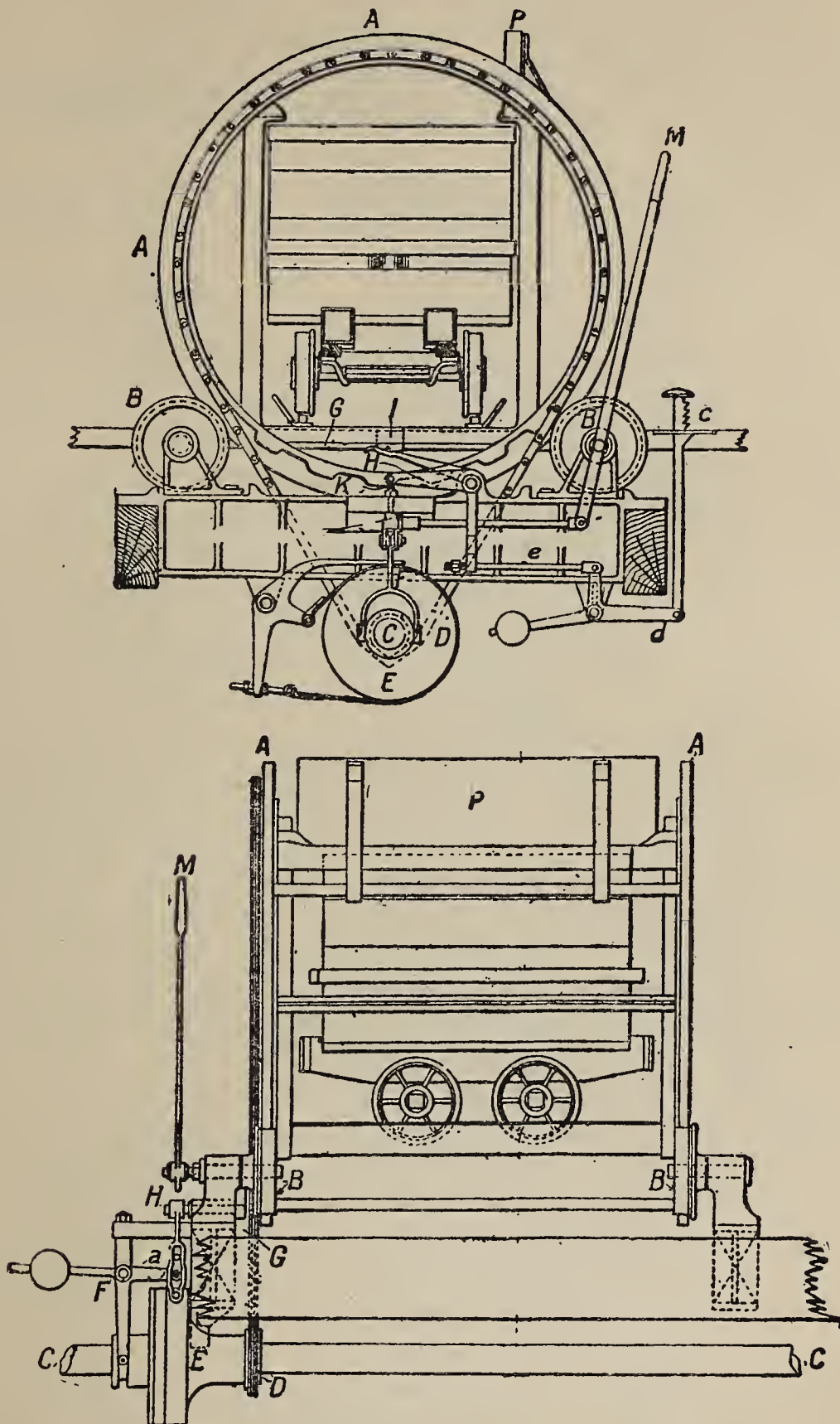
Tipplers.—The simplest and most common kind of tippler for emptying the contents of the tubs on to the screens used to be an arrangement of two rails bent up in the end, and pivoted on an axle placed beyond the centre of gravity from one end, so that, when the tub was run to it, it was tipped up and emptied, and had then to be drawn back into its original position. This tippler is now seldom adopted, as the coal had to fall too great a distance down to the screen, and also necessitated the tubs being provided with a door at one end for emptying, which is a disadvantage for underground work, as such doors are apt to fly open and allow the coal to fall on the road.

Rotary Tipplers.—During recent years a large number of tipplers have been designed, chiefly on the rotary principle, in which the tub is turned through either a half or full circle.

One of this class is Wood & Burnett's, which is shown in figs. 694, 695. It is driven by a chain and sprocket wheel arrangement, or by toothed gearing, the starting and stopping gear being the same in either case.* The tippler consists of circular cast-iron wheels A, riveted together by bars and angles in the usual way, and supported on four cast-iron rollers B. A counter shaft C, driven by the screen engine, carries the sprocket wheel D, attached to a friction clutch E, by means of which the tippler may be thrown into gear with the shaft C. The friction clutch also acts as a brake-wheel, a wrought-iron strap passing round it for that purpose. The sleeve of the friction clutch and the brake-lever are moved by means of a weighted bell-crank F, so arranged that when the end *a* is depressed the brake is applied and the friction clutch thrown out of gear. On releasing it the weight causes the brake to be released, and the friction clutch gears on again. The tippler is started by means of

* *Trans. I. M. E.*, vol. ix. p. 232.

the hand-lever M, so that the wedge L is withdrawn from the bolt K; the latter is thus shortened, allowing the weight on the lever F



FIGS. 694, 695.—Wood and Burnett's Rotary Tippler.

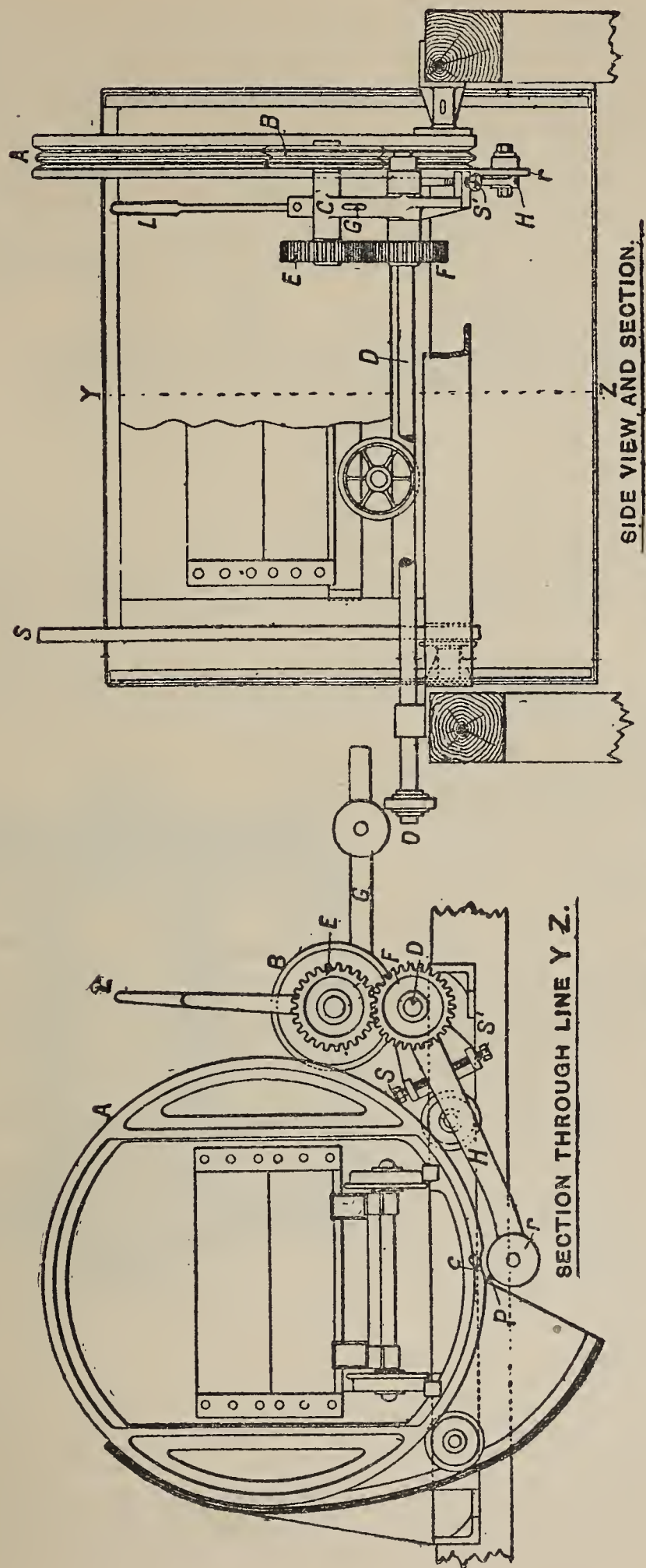
to fall, by which the brake is released and the friction clutch put into gear. At the same time the lever O falls clear of the stop I, and the tippler at once commences to rotate.

The hand-lever M is then allowed to fall back and to push the wedge L through the aperture in the box, thus lifting the end of the lever G into position ready to engage with the stop I, when the revolution of the tippler has been nearly completed. The lever G is pushed down by the stop I, and so applies the brake and throws the friction clutch out of gear automatically and simultaneously.

An auxiliary attachment, consisting of a treadle, lever, and rods *c d* and *e*, is applied, so that the tippler can be stopped during any portion of its revolution, if necessary.

Heenan and Froude's Rotary Tippler.—This tippler is also of the rotary type, having the cylindrical frame supported upon two pairs of rollers. The cast-iron ends *A s* (figs. 696, 697) of the tippler are dissimilar, *A* having two grooves on the edge, so as to engage the friction wheel *B* when in motion. This friction wheel is mounted on an arm *C*, which is pivoted on to the countershaft *D*, from which the motion is transmitted to the friction wheel by the spur wheels *E* and *F*. The countershaft *D* is driven from the screen engine, and is in constant motion. The arm *C* also carries a hand-lever *L*, by which the friction wheel is placed in contact with the grooved rim *A* of the tippler frame. An arm *G*, with a counterweight attached, is used to throw the friction wheel *B* out of gear, and to automatically stop the tippler at each end of the revolution. The lever *H*, having a small roller *r* and a pawl *p* mounted on one end, is pivoted loose on the shaft *D*, and its movement is transmitted by means of the adjusting screws *s* and *s'* to the arm *C*, and thence to the hand-lever *L* and friction wheels. After placing a tub on the tippler, the banksman pushes over the lever *L*, so that the roller *r* and the pawl *p* are lifted from the recess *c* on the rim *A*, and the friction wheel *B* is put into gear at one movement. On completing a revolution, the roller *r* drops into the recess *c*, throwing the friction wheel *B* out of contact with the rim *A*, and the pawl *p* brings the tippler to rest in the proper position for receiving another tub.

Travelling Belts.—After the coal has been passed over the screens to get rid of the small coal or dross, it is received on a travelling belt, where it may undergo a final hand-picking before being loaded into waggons. The belts are made of varying materials, according to the work to be done. For coal picking and conveying they are usually of iron or steel plates, link or woven wire, or of flat hemp or cotton belting with a hardened indiarubber face. Where only two kinds of coal have to be separated from each other, such as a cannel and a free coal, belting made of canvas faced with leather can be used with much advantage. The ordinary type of belt is made of steel plates 9 to 12 in. broad, $\frac{1}{8}$ in. to $\frac{3}{16}$ in. thick, fixed to iron or steel link chains that fit into a split polygonal drum at each end. The belt is supported usually on cast-iron rollers keyed to a shaft, which runs in pedestals fixed to the supporting beams. The delivery end is provided



SIDE VIEW AND SECTION.

SECTION THROUGH LINE Y Z.

Figs. 696, 697.—Heenan and Froude's Tippler.

with tightening screws and slotted seats to take up any slack on the chain (see fig. 698).

In driving, the usual type of hexagonal wheel is employed, two being placed at each end of the belt. These belts have usually a very slow motion, travelling at the rate of 40 to 80 ft. per minute. The length of the belt may be anything between 30 and 150 ft., but where they are used for picking the dirt out of the coal 30 to 40 ft.



FIG. 698.--Tightening Screws for Belt.

is a usual length. Their width varies from 2 to $4\frac{1}{2}$ ft., 3 ft. or $3\frac{1}{2}$ ft. being a good width where the picking is done from one side only. These belts are now often of steel wire rods woven across each other, as shown in fig. 699, but this type of belt can only be used for large sizes of coal.

Elevators.—When the small coal or dross is dropped through the screens it generally falls into a large hopper or dross pit, and is thence raised by means of elevators to be further treated either by dry or wet cleaning. These elevators are made on the same principle as the travelling belt. Two long endless chains with long single links fixed alternately inside and outside each other, having an iron bucket fixed to them, pass round a hexagonal wheel at each end. The length of these elevators and their speed depend greatly upon the quantity of coal to be handled, and the height to which it requires to be raised. An average speed is 300 to 400 ft. per minute.

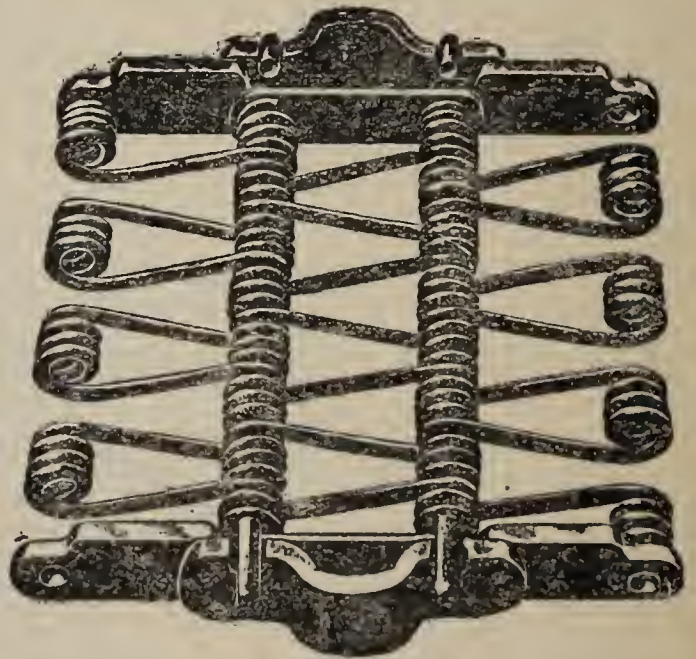


FIG. 699.--Segment of Wire Belt.

Arrangement of Cleaning Apparatus.—We have described the various parts of a coal-cleaning arrangement, either of which is seldom used by itself, but all of them being used in combination, whether the coal is subjected to a dry or wet cleaning process or not.

If the coal can be cleaned sufficiently well without washing, the arrangements are greatly simplified, and less outlay in plant is

required. Fig. 700 illustrates an arrangement of an ordinary cleaning plant. It must be remembered, however, that a given arrangement of plant will hardly ever suit two different collieries, as the screening arrangement will altogether depend on the quality of the coal, the amount of dirt in the coal, and the kind of coal in demand.

Coal Cleaning.—Under this heading may be included the various methods of washing and sorting the coal when brought to the surface. The processes employed may be divided into two groups:—

Cleaning by mechanical means;

Cleaning by taking advantage of the physical properties of the mineral.

Under the first head may be included (a) screening; (b) washing the mineral to cleanse it from mud, clay, shale, etc.; (c) hand-picking; and (d) sorting.

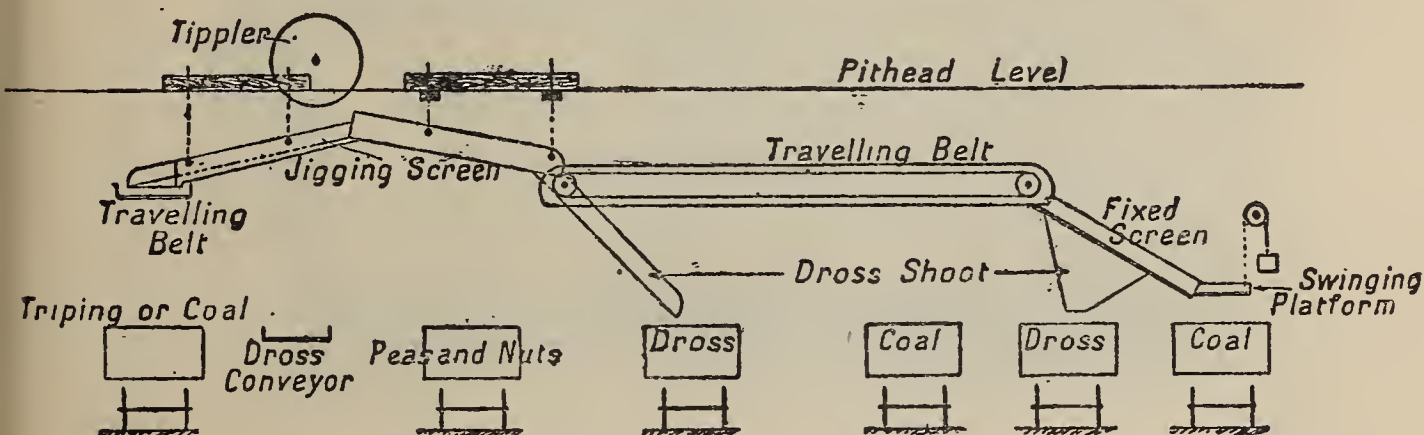


FIG. 700.—Cleaning Plant.

Screening the Coal.—This is usually the first operation to which the coal is subjected after it arrives at the surface. The screens employed may be divided into classes, viz.,

Fixed inclined screens, with movable bars.

Revolving screens.

Jigging or shaking screens.

Fixed Inclined Screens are very largely used, but they are fast being superseded by the shaking type of screen. When the former only are in use, the coal is picked by hand, and, as a rule, this is the only cleaning it undergoes. Fixed inclined screens are usually made of iron or steel bars set at an angle of about 30° from the horizontal, so that the coal can just move slowly forward. The bars may be made of different shapes, but are very often rectangular in section, about 1 in. thick, 3 to 4 in. deep, and 10 to 15 ft. long, according to the quality of coal dealt with. A soft coal generally requires a longer screen than a harder quality, as there is more dross to be got rid off. The openings between the bars vary from $\frac{1}{2}$ in. to $1\frac{1}{8}$ in.,

according to requirements, and sometimes by agreement with the landlord of the coalfield, as different 'lordships' or royalties are often payable on different qualities or sizes of coal.

Revolving Screens.—These screens are made like a large revolving riddle fixed on a central axis. They are largely used for dry-cleaning and sorting the smaller qualities of coal after they have been passed over a fixed or jigging screen. They may be made either of wire netting or sheet-iron plates with circular perforations to suit the size of coal required. A revolving screen is usually made to sort or separate two or three different sizes of coal, different parts of the screen being provided with differently sized apertures, as shown in fig. 701.

Jigging or Shaking Screens.—This class of screen is now almost exclusively used in place of the fixed bar type for the screening and sizing of coal. The screens are set at very slight inclination,

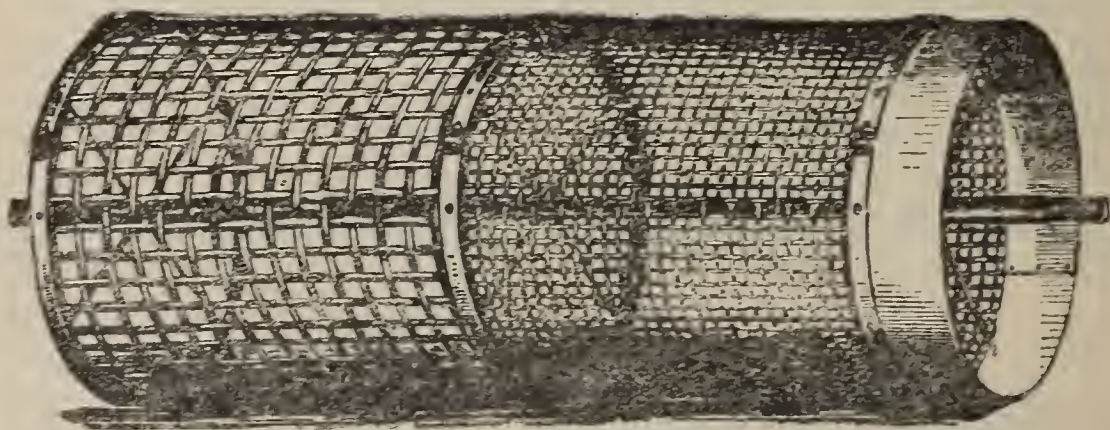
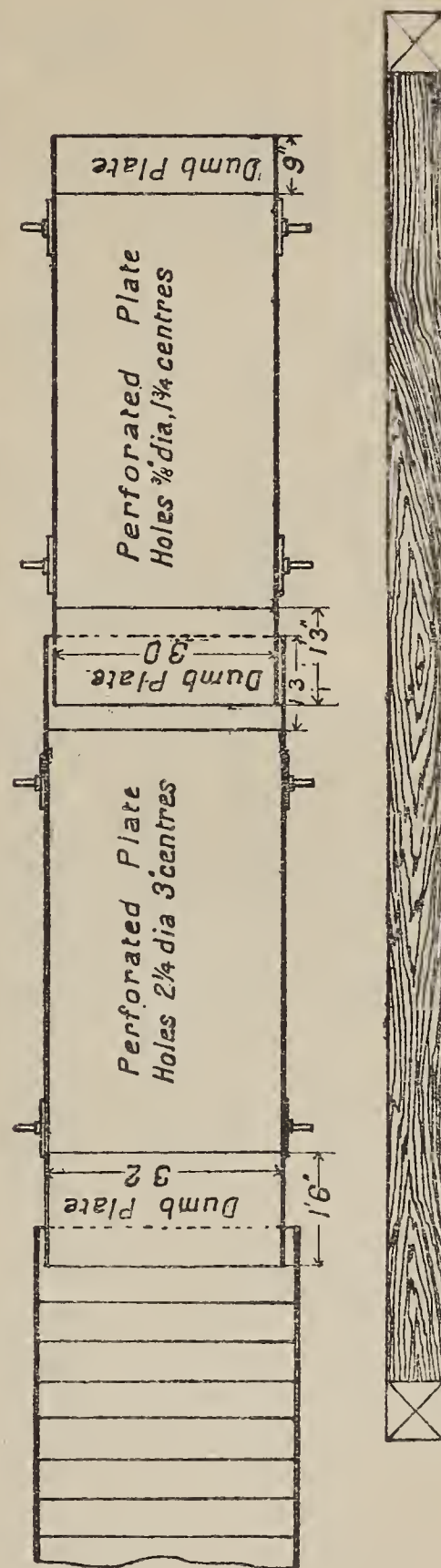
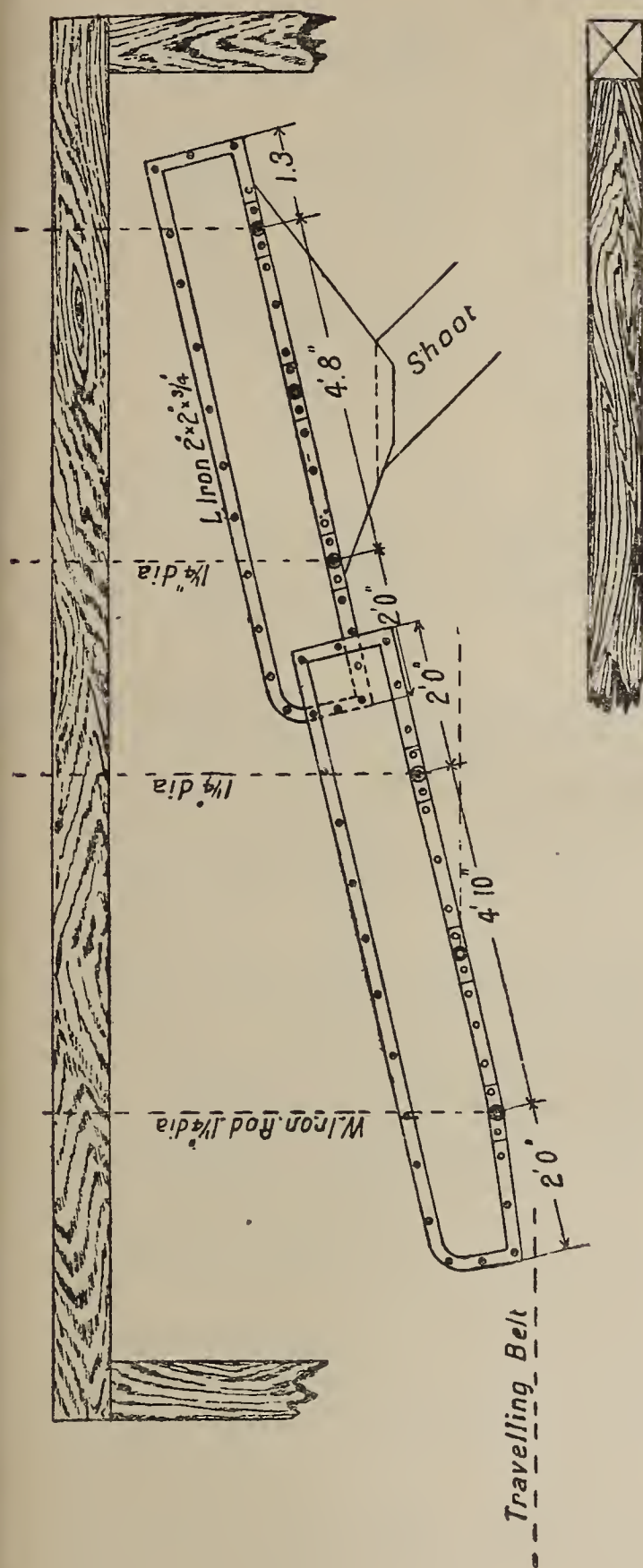


FIG. 701.—Revolving Screen.

10° to 12° from the horizontal, and hence the coal descends very slowly, gets very little broken during the process, and can be thoroughly picked and cleaned. The screens usually derive their motion from arms attached to the sides, and which work eccentrically, giving the screens a rapid to-and-fro motion, the coal at the same time moving slowly forward. They are usually suspended to beams by means of four rods fitted with adjustable screws for altering their inclination if required. The eccentric for driving has a short stroke, 3 to 6 in., and makes about 60 to 100 revolutions per minute. The screens may be constructed of wire netting fixed in a frame, or of sheets of iron or steel with circular openings. The latter are very largely employed, and are preferable to those constructed of wire netting, as they last much longer and size the coal much better. Figs. 702 and 703 illustrate the construction of a jigging screen and the method of fixing it.

Coal Washing or Wet Cleaning.—In washing coal the main idea is to separate the mineral from impurities, in other words, to reduce the percentage of ash. Washing coal also reduces the amount of sulphur present, and in consequence increases, in proportion to the



FIGS. 702 and 703. —Jigging Screens.

contents, the amount of fixed carbon. The following table shows the results of washing coal with different types of machines in the reduction of ash :—

Machine.	Ash in unwashed coal.	Ash in washed coal.
Coppée, ...	22·64	3·24
Ramsay, ...	25·35	13·51
Robinson, ...	27·23	18·24
Bell and Ramsay, ...	13·84	3·40
Sheppard, ...	15·37	9·06

As stated, the percentage of sulphur is also affected by washing, and from experiments made on American coal the following results were obtained :—

Volatile and Combustible Materials.		Fixed Carbon.		Ash.		Sulphur.	
Before washing.	After washing.	Before washing.	After washing.	Before washing.	After washing.	Before washing.	After washing.
29·55	30·69	60·36	63·51	9·98	5·78	1·45	1·25

These figures are averages of six different trials. The other objects to be aimed at in installing a coal-washing plant are :—

- (1) The employment of a minimum amount of labour.
- (2) Avoidance of breakage of the coal.
- (3) Complete separation of foreign material (principally shale or blaes) from the washed coal.
- (4) Avoidance of loss of coal, both in the form of nuts and fine coal in the foreign material which goes to the refuse heap, and in the form of slimes in the water employed in the washing.

A coal - washing installation which does not satisfy these conditions is an expensive and unsatisfactory addition to the plant of a colliery. Coal below a certain size, when mixed with foreign substance, is difficult to clean properly without washing. Some coals are so intermixed with impurities that no other method than wet cleaning would be of any use. As small coal usually contains the highest percentage of impurities, it is usually dealt with in this way. The cleaning of coal may be performed in many ways, all of which are, however, based on the same principle, *i.e.* the differences in density existing between the impurities and the coal. The different methods of washing coal are : * (1) by an ordinary inclined trough washer with fixed stops ; (2) by a fixed trough washer with movable scrapers, of which Elliot's patent is an example ; (3) by movable troughs, such as the Murton and the Wood and Burnett washers ; (4) by continuous ascending current washers, such as the Robinson washer ; (5) by intermittent ascending current washers, as in the

* *The Practical Engineer's Pocket Book*, 1897, p. 323.

Lührig and Coppée washers; (6) by compressed air, as in the Baum washer.

Choice of Washer.—In choosing a washing plant the most important consideration is the financial one, and this is generally where the difficulty lies, for a certain class of machine may be expensive in first cost and yet may soon recoup the outlay by economy in working and in producing a cleaner and better quality of coal, while a machine that may be erected at half the cost may prove much more expensive in other respects in the end. An important point also is that the coal must be adapted to meet the buyers' requirements, while different kinds of impurities require different methods of treatment. M. Callon states the position clearly when he says:—"In the case of coal cleaning we are dealing with enormous quantities of a substance of comparatively little value, the profit on which is reduced to a minimum by competition. We are consequently forced to employ special machines as simple as possible in construction, and capable of treating considerable quantities. We must abandon all idea of treating coal by frequent repetitions of the same process, or, at all events, do so sparingly, so as not to burden the undertaking with heavy costs, and the operations must be carried on with simplicity and economy."

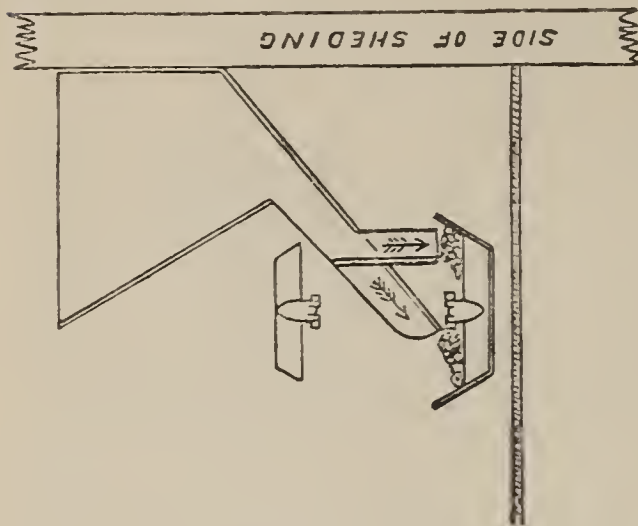
For small quantities of coal, and where not much sorting is required, the trough washer is often found satisfactory, or, if a more thorough cleaning is desired, the ordinary 'Bash' washer is as good as any for economical working.

When large quantities have to be dealt with, and the coal requires sorting into a variety of sizes, some of the more expensive machines, such as the Lührig or Coppée, may be adopted. Before any class of machine is fixed upon, the coal and its accompanying impurities should be submitted to a thorough analysis; the result of such a test often affording sufficient information for a decision to be arrived at. The three important points in coal cleaning are:—To remove impurities as far as possible; not to allow any coal to pass away with the impurities; and to achieve these objects in the cheapest manner with efficiency.

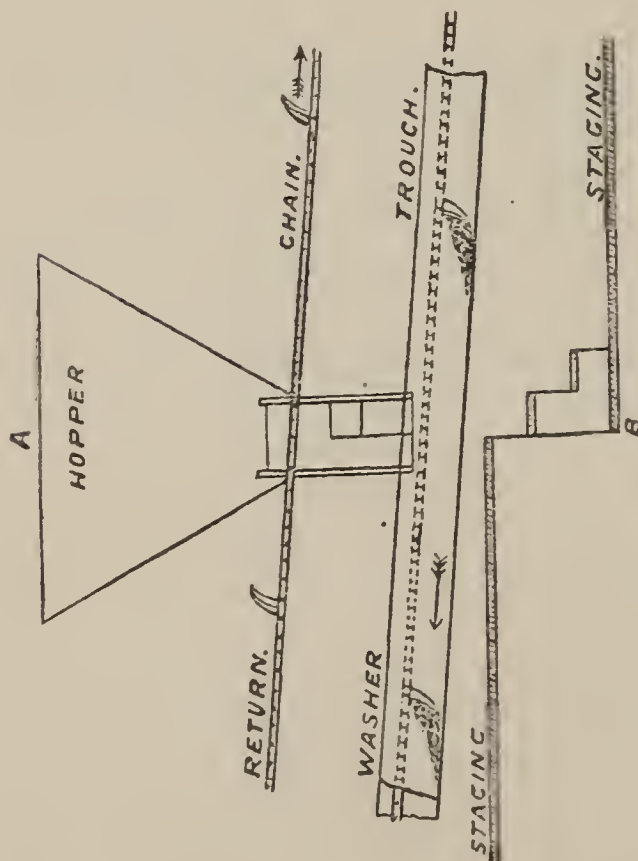
Trough Washer.—This is one of the earliest and simplest types of coal washers. It consists of a long, narrow trough divided into a number of stages by means of projecting pieces fixed to the bottom or sides of the trough. The trough is set at an inclination of 2 in. or 3 in. per yard, and a stream of water is allowed to flow continuously and sufficiently fast to carry the coal forward over the projections, while the heavier impurities fall to the bottom and are caught in the divisions of the trough. When a certain quantity of coal has been cleaned the flow of water is stopped and the refuse cleaned out, ready for another operation.

An improvement on this form of trough, with stationary dams, is a travelling scraper on the endless-chain principle, which, by moving

along the bottom of the trough against the stream of water and coal, delivers the impurities automatically at the upper end of the trough. The dimensions of these troughs are from 50 to 80 ft. long, 2 to 3 ft. wide, and 10 to 12



SECTION THRO' AB.



SIDE ELEVATION.

Figs. 704, 705.—Elliot Trough Washer.

in. deep. The trough washer is best suited for small quantities of coal; it requires a large flow of water and entails extra labour, but it has the recommendation of being simple in construction and cheap at first cost. A trough washer 60 ft. long and 2 ft. broad, with revolving riddle and connections, can be erected for about £180.

Elliot Trough Washer.

—This is an improvement on the old trough washer. It consists of a wrought-iron or steel trough, about 18 in. wide at the bottom and 30 in. wide at the top, and having sloping sides. At each end a sprocket wheel is fixed, round which an endless chain passes, and attached to the chain at intervals of 6 ft. are fixed scrapers to fit the inside of the trough. The scrapers form stops or dams, which are slowly moved by the chain along the trough in the opposite direction to the flow of

the water. The trough is set at an inclination of about 1 in 12, and the coal is admitted from a hopper A (fig. 704) at the centre of its length and the water at its upper end. As the water runs down it carries with it the coal, which is lighter than the dirt, while the latter settles in the scrapers and is carried against the stream of water and delivered at the opposite end.

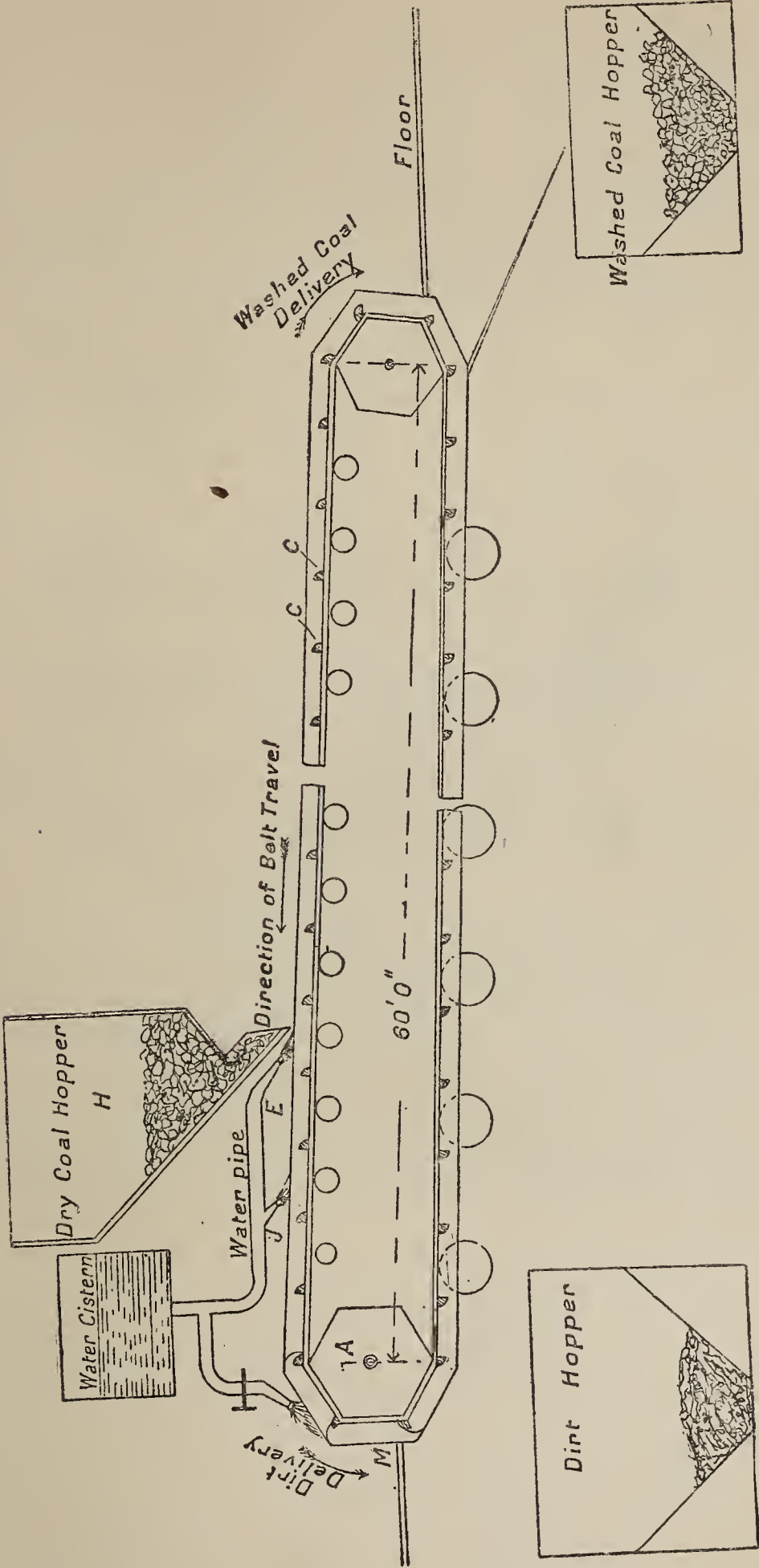


FIG. 706.—Murton Coal Washer.

Murton Coal Washer.—This machine, like the Elliot washer, is an improved trough washer. It consists (fig. 706) of an endless articulated steel trough belt, which is watertight. This trough revolves slowly round suitable drums at each end, the action being continuous and automatic. The clean-washed coal falls into a hopper at the lower end; the dirt or refuse is delivered into a dirt hopper at the upper end. The trough is constructed of steel 60 ft. long, 3 ft. wide, 8 in. deep, and fixed at an inclination of 1 in 18. The supporting drums and rollers are mounted on a suitable frame. Inside the trough, at intervals of 3 ft., are dams or stops about 2 in. high.

Method of Working.—The trough is set in motion and travels up the incline, and towards the coal and water supply, at a speed of from 8 to 10 ft. per minute. The coal from the hopper H (fig. 706) and water from the nozzle E are turned on in suitable proportions. The latter, carrying the coal with it, flows in a direction contrary to that of the trough to the coal delivery end, where it is delivered free from impurities. The dirt falls to the bottom of the trough, and is arrested by the stops *cc*, and carried forward by the motion of the trough beyond the point E, after passing which it is agitated in such a manner that any coal mixed with it is liberated, and carried back by a strong current of water from the supply pipe J to E, where it joins the regular coal feed, and is carried along with the rest of the clean coal forward. The refuse continues to be carried upwards to the end of the trough M, and as the belt moves over the drum A it falls off into the hopper. The trough is washed clean by the water spray, and passes to the drum at the lower end. The coal, when delivered at the lower end, falls into a spout, with a draining plate (with 125 perforations $\frac{1}{16}$ in. diameter per sq. in.), and the water is quickly drained away. About 450 gallons of water per minute are required when washing 400 tons of coal per day, but the same water can be used over and over again.

Greaves Washer.—This machine is capable of dealing with 15 tons of $\frac{7}{8}$ in. to $\frac{1}{2}$ in. small nuts per hour, and requires the attendance of one man only. The water consumption is approximately 20 gallons per hour.

Fig. 707 shows the washer in longitudinal section, with overall dimensions, and illustrates the plant. It consists of a box containing water in which the washing trough G is supported at the front end at the point K, and at the opposite end by the cams which cause the oscillations. The unwashed coal is fed into the machine at the end H, and by means of the oscillations is caused to travel in the direction of the arrow, the coal passing over the weir J and thence to the drainage elevator M, whilst the dirt, by reason of its greater weight, sinks to the bottom and is automatically discharged on to the elevator N. The manner in which the dirt is removed and automatically controlled is a special feature of the washer. By

means of this arrangement the machine is able to take its supply of fuel in variable and intermittent flow, exactly in the manner in which the screens usually deliver it. It adapts itself automatically to suit coals having varying percentages of dirt. Feeding hoppers of large capacity are unnecessary, as also constant attention for the manipulation or regulation of the dirt discharge, which is entirely automatic.

Fig. 708 illustrates the manner in which this control is automatically accomplished. A and B represent the sides of the washing cradle G (fig. 707), being a cross section on the line X, Y. Between the two sides is arranged a frame supporting a rocking table C, hinged at the point D, and covered by a perforated plate E. This rocking table is balanced by adjustable weights, and also for area, the latter being

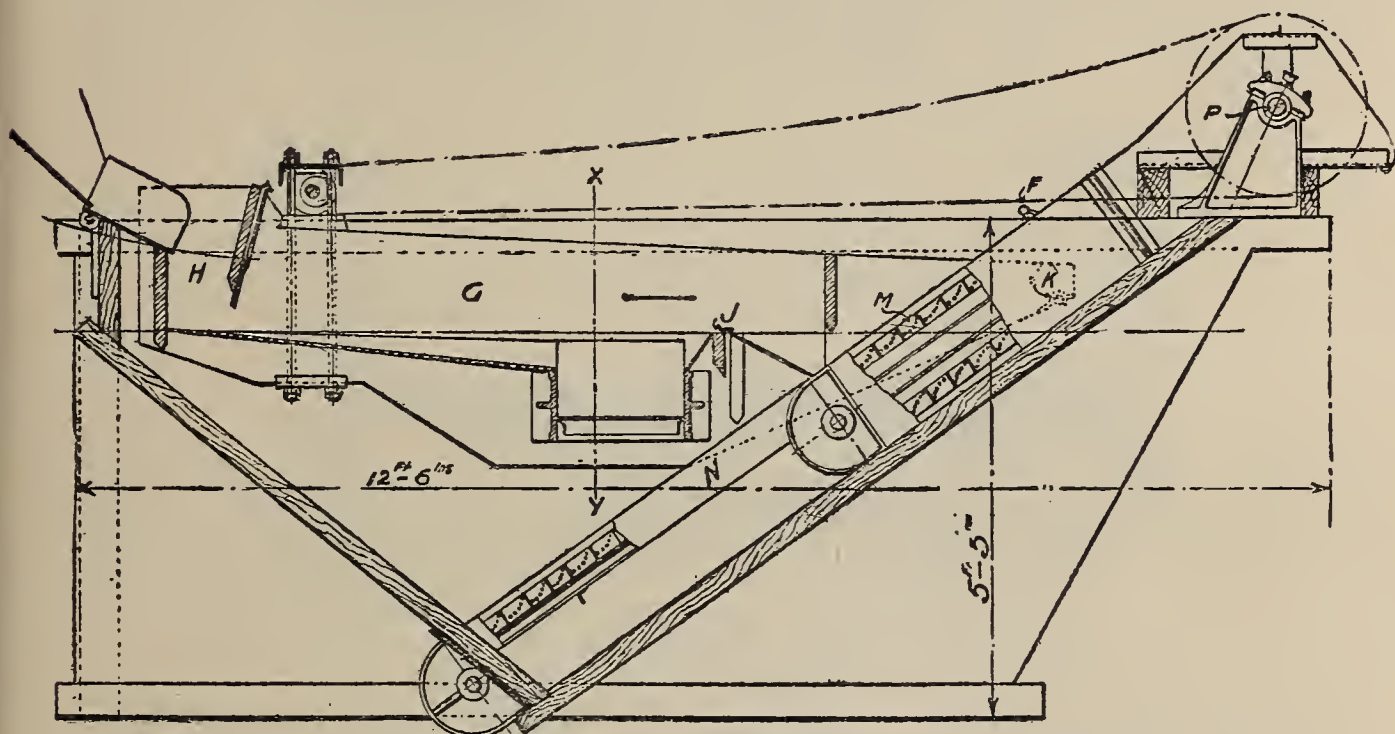


FIG. 707.—Greaves Coal Washer, Longitudinal Section.

necessary to equalise the forces set up by the oscillations. At the side B is arranged an indicator designating the position and amount of discharge, and which may also be used for adjusting the weight.

The *modus operandi* is as follows:—The dirt naturally arranges itself at the bottom, and the accumulation increasing up to a predetermined amount so that the weight of dirt is sufficient to overcome the balance-weight at the opposite side of the fulcrum, causes the table to rock about its fulcrum, and the dirt is discharged through the opening thus created, after which the balance-weights restore the position of the table. In this manner the depth of the dirt is maintained at a constant level, any excess being immediately discharged. In the working and balancing due account is taken of the respective weights of dirt and coal under water.

The coal, as it passes over the weir J (fig. 707), is conducted by the shoot on to the continuous bar conveyor. This latter consists of an

endless strand of U-links made of 3 in. by $\frac{3}{8}$ in. flat, with $\frac{3}{4}$ in. bar axes and $\frac{1}{2}$ in. diameter grid bars, each of the axes having a wearing surface about 3 ft. long. This endless chain is stretched over two pairs of drums, the lower of which are placed below the water level, and the upper pair keyed on to the shaft P, which takes its motion from the main shaft S which oscillates the washing trough. The conveyor is arranged on an incline, and the water draining off goes directly back into the box and not on to the succeeding coal.

Primus Washer.—The mechanical principle of the Primus washer, differs from other machines of the same type in the movement of the coal in the water. The uncleaned coal is fed into a perforated bottom reciprocating box (in motion), where displacement of the lighter by the heavier materials is effected by agitation; it will thus be seen that, as opposed to the plunger type of washer in which the

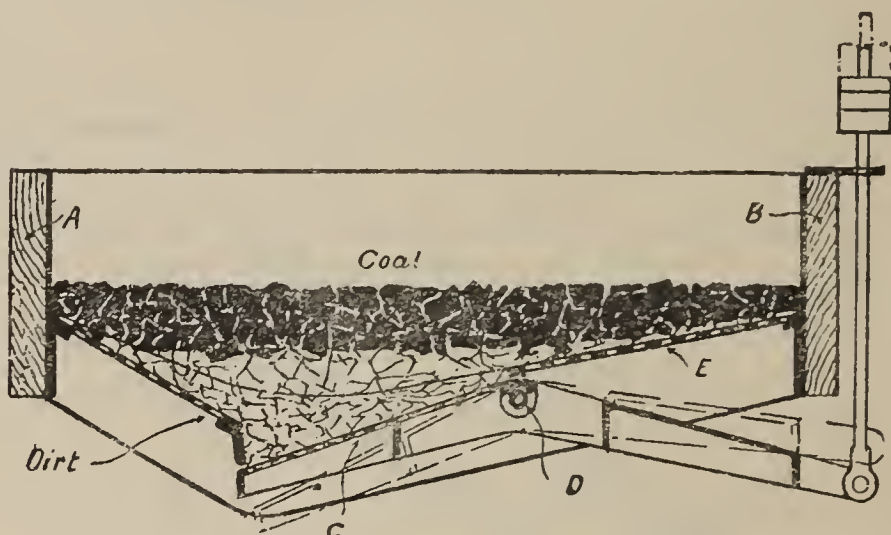


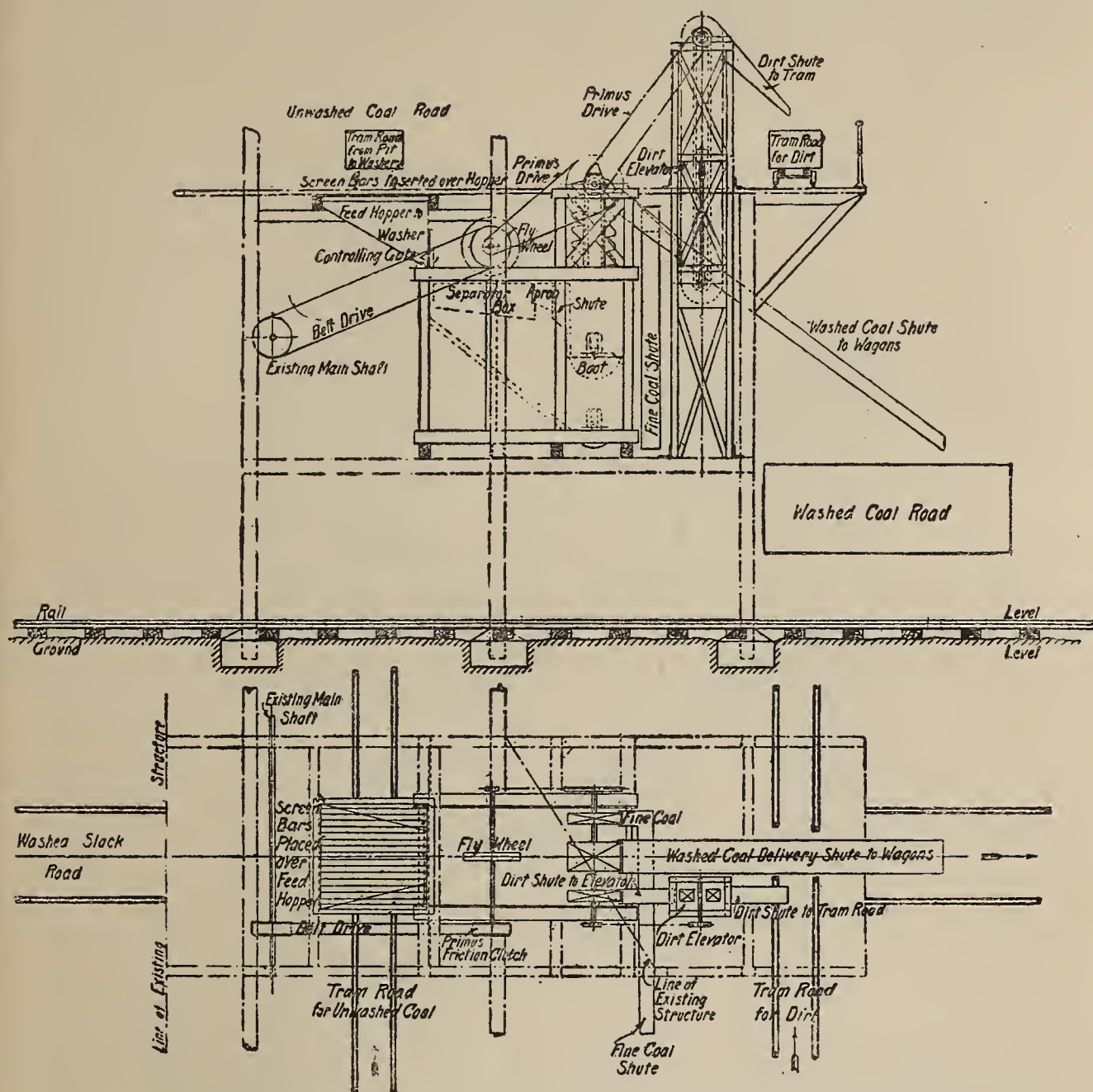
FIG. 708.—Cross Section on X, Y.

water is forced up through the coal, in the Primus plant the coal is dropped through the water. No felspar is employed, the bottom in the washer being formed by the heavier of the two materials, the depth or thickness of which bed is always under control, and stratification of the several layers being perfectly even. Positive control over the discharge of coal and waste is attained by the movement of an automatic cradle, through which the line of separation and depth of bottom are also controlled.

It will be seen from the illustrations that the 'bottom,' formed of the heavier materials in the coal, is retained at a given depth by the height of overflow of an adjustable cradle connected with levers above the water line. A single movement of these levers raises or lowers the height of refuse discharge, and thereby controls the delivery of coal. It is advisable, however, to fix the levers as soon as, by experience, the most suitable height has been ascertained for the particular coal to be washed. The stratified layers move forward automatically, the light coal over the apron and the heavier refuse to the cradle, under the impulse of the current generated from the

pulsating separator box and the slanting end of the water tank. The waste is held upon the perforated bottom until a sufficient amount is collected to displace the clean coal and force it up to a higher level into the delivery shute, both exits being adjusted by lever connections as described.

The supply of material is effected by means of an elevator, driven



FIGS. 709 and 710.—Primus Coal-Washing and Sizing Plant.

from the same motor that drives the agitation device, and this, operating reciprocally with it, secures a ratio of feeding exactly proportioned to the rate of delivery of discharge. The water is circulated continuously through the washer without any intervention of a settling pond, as by this process a very large proportion of the smudge is retained at the bottom of the tank, and is delivered by a separate elevator at the same level as the coal and stone. The water required only amounts to about 6 gallons per ton, or approxi-

mately the flow of a $\frac{3}{4}$ -in. pipe without head, while as much as 15 tons of some coals can be thoroughly cleaned over a washing area of $15\frac{1}{2}$ sq. ft., or rather less than 1 ton per sq. ft. per hour.

Robinson Washer.—This machine, which is shown in fig. 712, consists of a truncated inverted cone, made of steel plates about 8 ft. diameter at top and 2 ft. diameter at bottom and $6\frac{1}{2}$ ft. deep. A strong shaft is fixed vertically, to run in the centre of the cone; on this shaft is a strong cast-iron cross-head, to which are bolted four

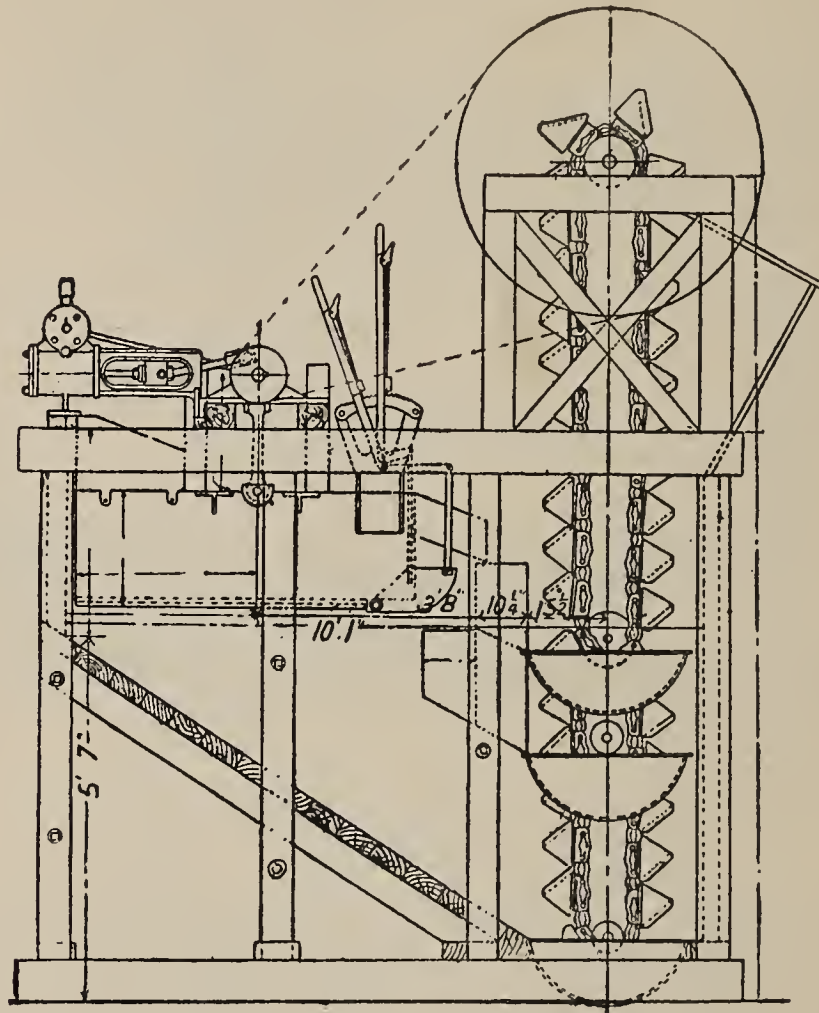


FIG. 711.—Enlarged Section of Primus Washer.

cross arms. To each of these cross arms are bolted three heavy wrought-iron bars, but outwards at the bottom, and projecting down so as almost to touch the sides of the cone or washer, and on the bottom of the driving shaft, are also bolted four shorter arms, as shown in illustration. In practice the coal passes into the washer from the spout A into the centre ring B, while the water supply is forced in at the bottom at E and through the perforations G.

The coal is kept in a continual state of agitation, and as it sinks into the washer it is met by the upward current of the water; and being lighter in weight than the impurities with which it is associated, is floated upwards, as indicated by the arrows, and overflows at D. The impurities sink downwards and are collected in the chamber J.

When this chamber is filled, the upper valve H is closed, and the lower valve H' opened; this allows the accumulated refuse to be discharged, after which the lower valve is closed and the operation repeated. The shaft, with cross-head and depending arms, revolves at the rate of 14 or 15 revolutions per minute.

In fig. 713 is shown a general arrangement of a Robinson washing plant.

The clean coal passes from the washer on to inclined perforated screens or shoots, and is here freed from the water, after which it is either carried by means of conveying machinery to storage bins, or falls direct into the waggons. The water from the overflow is collected in settling ponds, and by means of a pulsometer pump is again forced into the washer for further use.

The Robinson washer is cheap to construct and maintain, and requires little water, but it largely depends for its efficiency on the attention and skill of the man in charge, who often may be tempted to pass more coal through it than it can effectually deal with. It is also liable to give a good deal of trouble in the winter season, owing to the formation of ice jamming it, and on the whole it cannot be recommended where these conditions are likely to prevail.

These machines can be made to wash from 20 to 40 tons per hour. The cost of a Robinson washing plant to treat 20 tons per hour is from £250 to £350.

Lührig and Coppée Machines.—A paper read before the South Wales Institute of Mining Engineers * gives an elaborate description of the construction and working of these machines. The following abridgement † will, however, suffice to explain their action. The machines are made in two sizes, the larger dealing with coal from 5 in. to $\frac{3}{8}$ in. in thickness, and the smaller taking fine coal and powder.

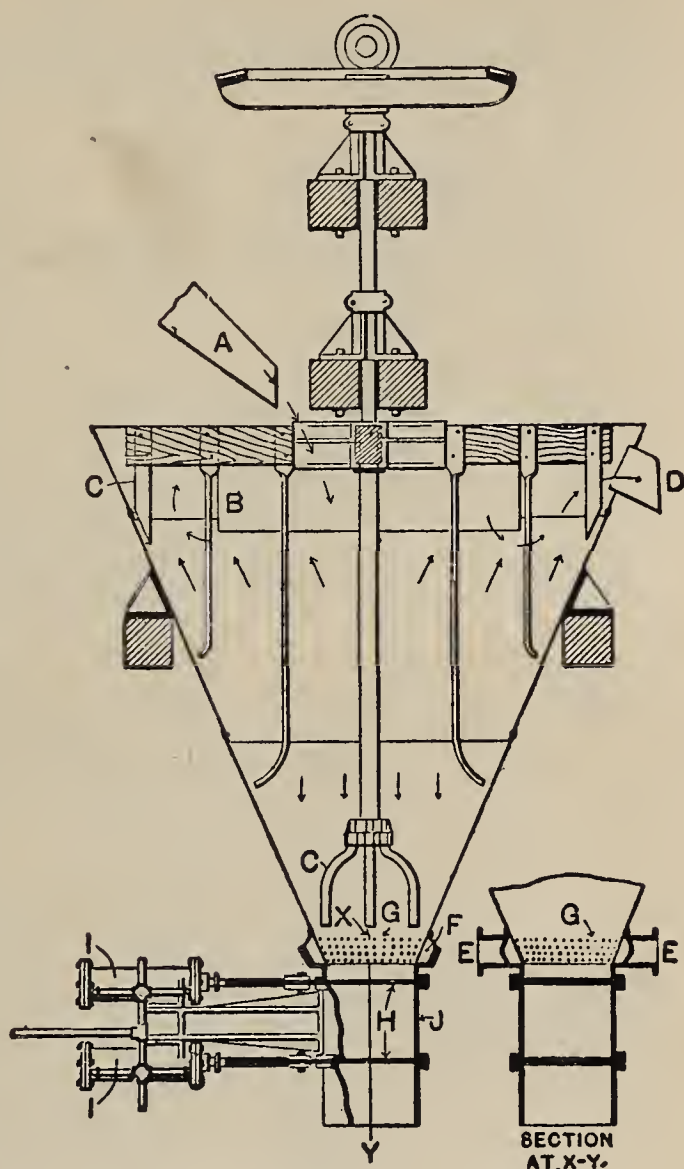


FIG. 712.—Robinson Washer.

* *S. Wales Inst. Min. E.*, xiv. pp. 88-102.

† *Hughes's Coal Mining*, pp. 416-424.

"The larger, or nuts machine (figs. 714, 715), is of the ordinary continuous jig type, and consists of two compartments A and B, in one of which the piston works, while the other is provided with a perforated strainer, slightly inclined from front to back. The piston P receives an up-and-down motion by being connected to cranks on a horizontal shaft, and the amount of this throw can be varied from $1\frac{3}{4}$ to 4 in. An opening W runs along the front of the washing compartment, and through this clean coal continuously passes away. The shale is discharged through a small cylindrical compartment D, connected to the

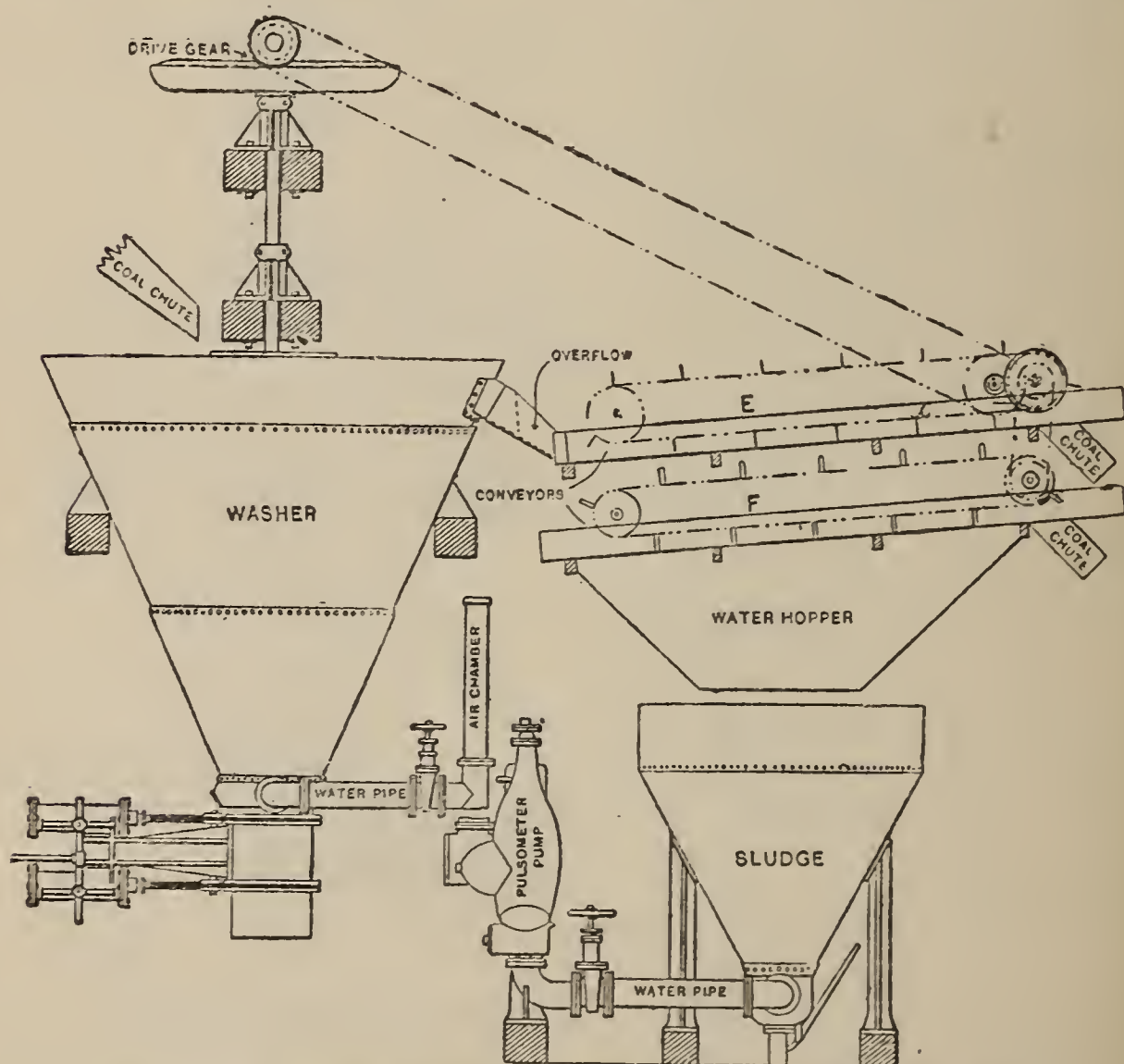
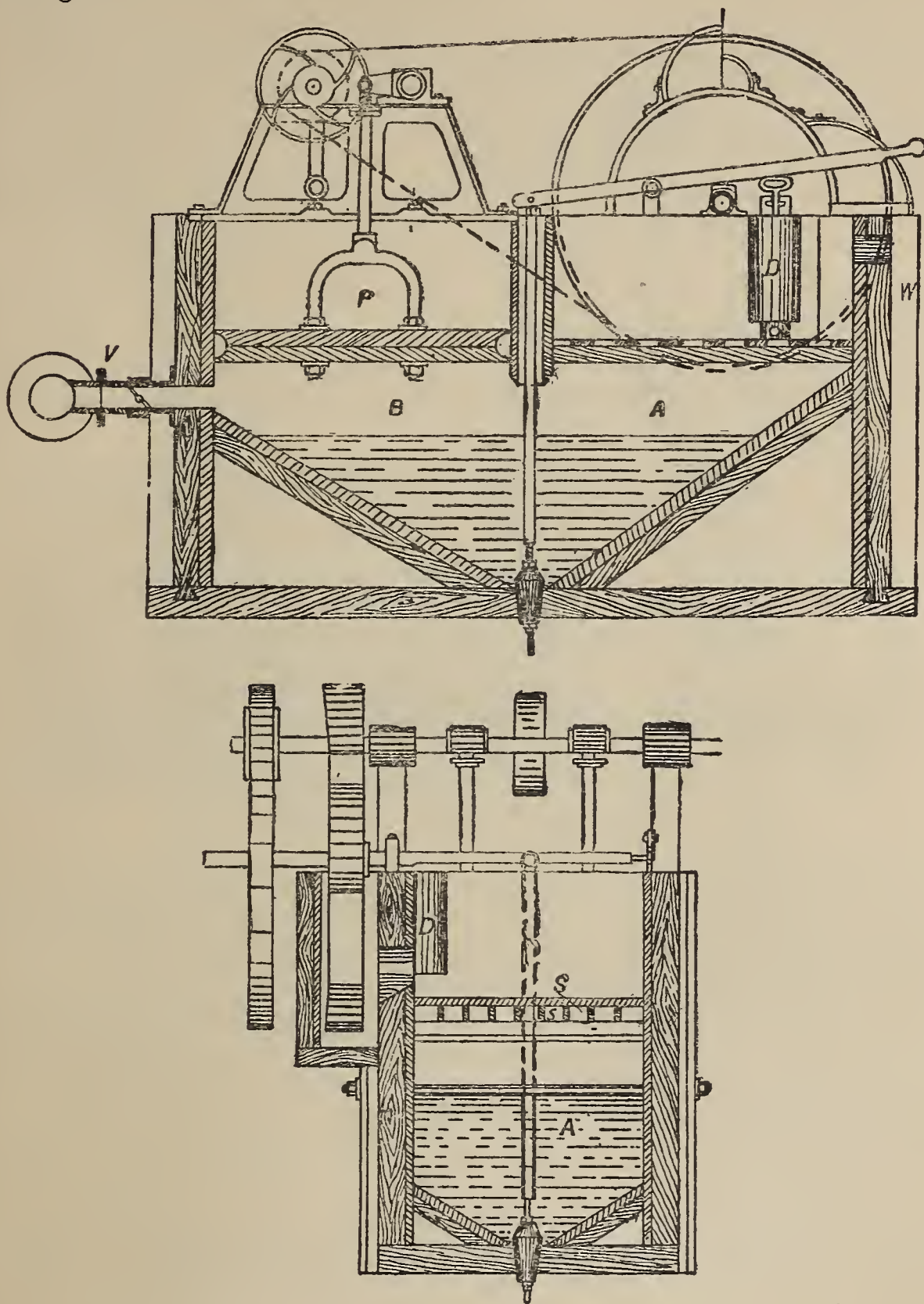


FIG. 713.—General arrangement of a Robinson Washing Plant.

side of the casing, but which starts above the level of the strainer, leaving a free space between the strainer and the lowest end of the compartment of about 3 in. It is open at both ends, and communicates with the outside of the machine through the opening R. It is provided with a sliding door which regulates the discharge of the shale.

"When the unwashed coal is introduced into the machine, and the piston descends, it drives water into the compartment B, and lifts the bed of the material resting on the strainer. On the return stroke, the heavier dirt falls faster than the lighter coal, while in the upstroke

the lighter coal is lifted farther than the heavier dirt; the result is, that the two substances separate into layers, the coal being, of course, the higher.

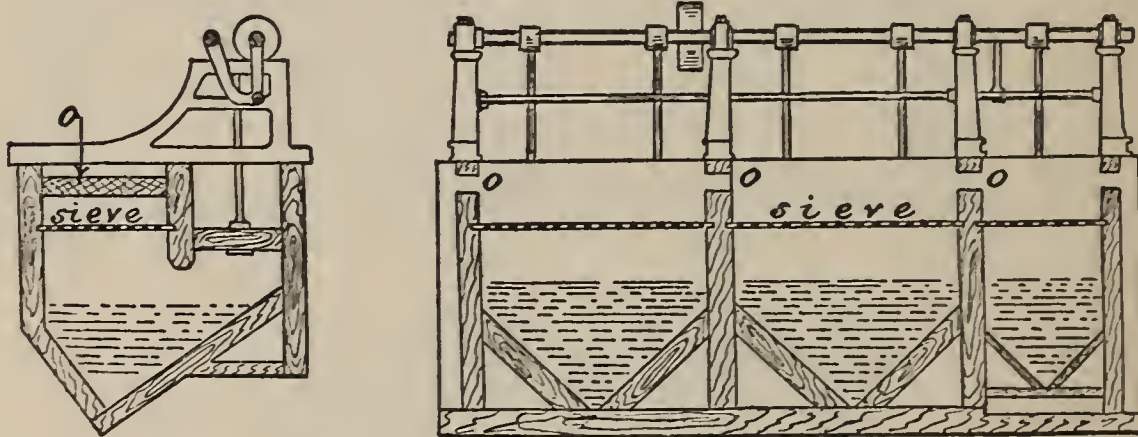


FIGS. 714, 715.—Lührig and Coppée Machines.

"The Felspar Washer is of similar construction, but differs materially in its method of working. It consists of a box, divided into two compartments by a longitudinal partition, in one of which the piston works as before (figs. 716, 717). It is also generally divided into

two or sometimes three compartments in the direction of its length, each communicating with the other by openings *o* along the side, and through these the washed coal passes away. In the nuts washer, the holes through the sieve are smaller than the size of the material being treated, and consequently no discharge takes place through them. In the felspar machine they are larger than the material, and the dirt passes through the sieve into the lower part of the apparatus. Three sieves are generally employed. The dirty coal is introduced at one end and gradually passes down over the remaining gratings, the clean material being finally discharged at the opposite end.

"The chief peculiarity is the introduction of a layer of felspar, from 2 or 3 in. thick, on each sieve, whose specific gravity is greater than that of the material to be concentrated, and yet less than that of the gangue. The sizes of the particles of this bed are larger than the holes in the sieve. The whole framework of the machine is



FIGS. 716, 717.—Felspar Machines.

filled with water up to the level of each sieve, and as the pistons work up and down, a volume of water is forced through the holes in the bottom of each sieve, lifting the bed and the layer of material on it, and then allowing the whole to fall again on the return stroke. The lighter coal rises to the surface, and the heavier dirt gradually finds its way through the bed of felspar, when it falls into the bottom of the compartment to be removed from time to time. It is essential for thorough cleaning that the size of the felspar should be as small as allowable, and that the particles of mineral forming the bed should be of convenient density, have well-defined rectilinear angles, and be of great durability to resist wear and tear. A point of considerable importance is the proper regulation of the delivery of water, which is controlled by a tap; upon this depends the progress of the material and the time it is operated upon.

"For very dirty coal, perhaps no machine does its work so efficiently as this; indeed, everyone gives it the character of removing dirt. It is, however, expensive in the first cost, but requires little attention. Much depends upon the percentage of dirt originally present in the coal. If it is small, and, say, one-half of it is removed, the coke from

the resulting product is a fair one; on the other hand, where the dirt amounts to from 15 to 30 per cent. and only 3 to 10 per cent. is taken away, the coke is very bad. With a dirty coal, probably it is best to use machines of this type."

Baum Washer.—This machine applies a new principle to coal

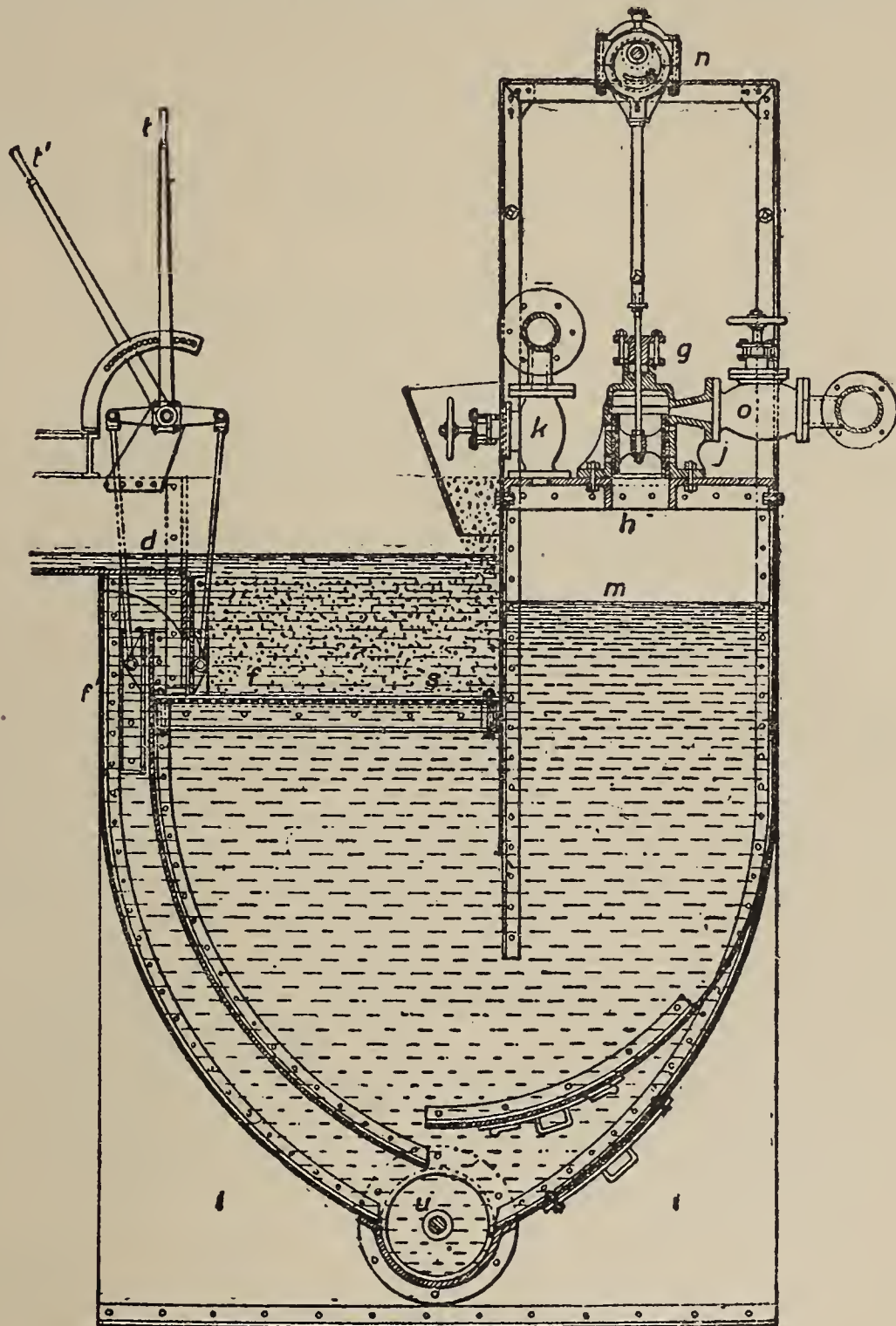


FIG. 718.—Baum Washer.

washing, inasmuch as compressed air is used, instead of a vertically reciprocating piston, for producing the oscillations of the water through the bed of the jigger. By this method it is claimed that all noise, vibration, and shock are done away with, and that less power is required than in an ordinary piston machine. Fig. 718 shows a

cross section of one of the jigging machines.* The jiggers are constructed of steel or wrought-iron plates connected by angle irons. The water-box *m* is closed at the top by a horizontal plate, to which is fitted a piston valve casing *g*, in which works a piston or cylindrical valve *h*, actuated by the eccentric *n*, and having ports or openings in the sides to control the admission and exhaustion of compressed air to and from the upper closed-in part of the water-box *m*. Openings *j* are provided in the sides of the casing *g* for the escape of the compressed air from the water-box; and *o* is a stop valve for admitting or shutting off the compressed air. The latter, at a pressure of $1\frac{1}{2}$ to 2 lbs. per sq. in., is drawn from the receiver, to which it is supplied by the air compressor.

In the coarse jigging machines the valve *h* makes from 50 to 70 strokes per minute, and in the fine jiggers from 75 to 110 strokes per minute, the alteration in speed being effected by means of coned pulleys on the driving and countershafts. The jigger sieve *s* is of $\frac{3}{8}$ in. mesh for the coarse and $\frac{1}{4}$ in. mesh for the fine jiggers. At the top of the machine there is an overflow bar *d*, over which the coal from the coarse jiggers is carried along channels to special nut pockets; *ff'* are two slides, adjustable by means of the levers *tt'*, for regulating the outflow of the dirt, which falls to the bottom of the jigging-box, and is conveyed away by means of a spiral conveyer *u* extending the whole length of the jig battery.

It will be noticed that the mesh used in the fine jigging machines is larger than much of the material treated, and it might be supposed that some of the latter, including the coal, would fall through the sieve. The machines, however, can be so regulated that the larger pieces of stone or shale are retained a considerable time on the sieve, and owing to the weight of material above them they set themselves parallel to the sieve and form a bed similar to the felspar bed employed in the Coppée fine jigging machines.

Cost of Coal Washing.—This will vary very much with the class of machine used, the quality of the coal, and the percentage of dirt accompanying it; and also on the amount of dressing or preparation the coal requires for the market.

The following table † shows the cost per ton of washing by various machines in the year 1886 :—

Name of Colliery.	Dry Cleaning.	Wet Cleaning.	Type of Washer.	Condition of Coal previous to Washing.
Barrow, . . .	1½d.	2½d.	Robinson.	Crushed.
Aldwark Main, . . .	...	2¾d.	Trough.	„
Nunnery, . . .	3d.	1·40d.	„	Not crushed.
Annesly, . . .	2½d. to 3d.	...	...	...
Clifton, . . .	3d. to 4d.	6d. to 7d.	Coppée.	Not crushed.
North Motherwell, . . .	...	0·7d.	Lührig.	„

* *Trans. I.M.E.*, vol. vii. p. 158.

† *Trans. M. I. Scot.*, Coal Cleaning Committee's Report, p. 179.

CHAPTER XVI

SURVEYING, LEVELLING, AND PLANS.

"SURVEYING is the art of ascertaining by measurement the shape and size of any portion of the earth's surface, and representing the same on a reduced scale, in a conventional manner, so as to bring the whole under the eye at once." *Surveying has also been defined by Mr Bennett H. Brough as "the art of making such measurements as are necessary to determine the relative positions of any points on the earth's surface. From such measurements a plan of any portion of the earth's surface may be drawn, and its area calculated."

In ordinary mine surveying three objects are in view, viz., to obtain a correct plan and section of the underground workings; to ascertain the correct position of all surface buildings, shafts, and other features, and the correct boundary line of the field or royalty to be worked, and to connect as accurately as possible the workings on the underground plan with the surface survey.

† "All surveys are conducted on nearly the same principles, the difference consisting in the style of the instruments used in the work, and the different methods of calculating the various data connected with the survey."

The instrument which is in most common use for ordinary underground surveys of mines is the miner's compass or dial, sometimes called the circumferentor. This is an instrument the chief use of which is to measure horizontal angles. As used with the loose needle the angles are measured in relation to the magnetic meridian, and as used with the vernier or fast needle the angles are measured either in relation to the magnetic meridian or in relation to any given base line. The instrument consists essentially of two parts, a needle swinging freely on a pivot, and the dial on which the angles are read. When surveying with the magnetic compass it is most important to remember that the needle does not point to the *true north*, and that the direction in which it points is subject to several variations from

* *A Treatise on Mine Surveying*, by Bennett H. Brough, p. 1.

† *Ibid.*

time to time. It is more than 300 years ago since observations were first made on the variations of the needle, and we are therefore not without some data to guide us. In Queen Elizabeth's time, *i.e.* about 1580, the variation was about $11^{\circ} 11'$ east; in Charles II.'s time, 1657, this variation had disappeared altogether and the needle coincided with true north. The needle swings backwards and forwards on each side of the true north like the pendulum of a clock, but with this difference, that it swings in centuries and not in seconds. Its maximum variation is $24^{\circ} 48'$ on either side, and this is termed the *secular variation*. Taking the average between 1580 and 1880, the variation was $8\frac{1}{4}$ minutes per year, and for the ten years between 1877 and 1887 the total variation was $1^{\circ} 12'$, giving an average of $7\cdot2'$ per year. Since 1660, in which year there was no variation, the declination has been towards the west, attaining its maximum in 1820, and then gradually decreasing, so that at the present time the magnetic meridian is again approaching the true meridian. The average annual variation of the needle may be taken at 8 minutes.

The magnetic declination at Greenwich in the year 1912 was $15^{\circ} 24'\cdot3$ to the west, as shown in fig. 719. At Edinburgh the magnetic declination is 3° greater than at Greenwich, while in Glasgow and Dublin it is $3^{\circ} 50'$ greater. In the North of England the variation is about $1\frac{3}{4}$ degrees greater than at Greenwich.

A very elaborate magnetic survey was carried out between the years 1884 and 1891, by Professors Rücker and Thorpe, to determine the amount of variation and the lines of attraction in the United Kingdom. Observations were made at 882 different places, and from their calculations and a map it will be possible to gain some idea how far local disturbances are likely to affect the needle at any given place.

As regards such disturbances it is found that the north-seeking pole of the compass is attracted to certain regions and to lines which can be traced for scores or even hundreds of miles. These lines they (Rücker and Thorpe) call *magnetic ridge lines*.*

The principal ridge lines determined were as follows:

- (1) In the Scotch coalfield a ridge line runs from the neighbourhood of North Berwick to the Clyde between Glasgow and Hamilton, turns south to Newmilns in Ayrshire, and finally runs in a northerly direction towards Ardrossan and Arran.
- (2) The Yorkshire coalfield is dominated by a ridge line which runs south-west from Harrogate towards Keighley, and follows the outcrop of the Millstone Grit to Matlock.



FIG. 719.

* *Trans. Inst. Min. Engs.*, vol. ix. pp. 418-419.

- (3) In the southern part of the Derbyshire coalfield the north-seeking pole of the magnet is attracted from all sides to a centre which is a little to the north-east of Nottingham.
- (4) In the Lancashire coalfield a ridge line runs from Rochdale to Wigan and Southport.
- (5) In the South Wales coalfield a ridge line runs from Risca to the south end of Ebbw Vale, and another passes from Brecon to Neath, and thence nearly due west through the centre of the coal measures.

If these ridge lines are therefore drawn upon a map, and the variation determined for places in their vicinity, the magnetic variation for any given locality may be pretty accurately ascertained.

The amount of variation may also be approximately determined for any locality in Great Britain by drawing lines on a map in the following directions : *

- (a) From Winchelsea (Sussex) by point of Sheppey Island and Ipswich, the magnetic declination is 30 minutes less than at Greenwich.
- (b) From Cowes (Isle of Wight) by Basingstoke and Great Grimsby 30 minutes more than at Greenwich.
- (c) From Start Point by Teignmouth, Newport (Mon.), the Peak to Seaham $1^{\circ}30'$ more.
- (d) From the Land's End by Great Ormes Head, Skiddaw, etc., $2^{\circ}30'$ more.
- (e) From Peel (Isle of Man) by Wigton to Falkirk, $3^{\circ}20'$ more. If these differences are applied to the declination at Greenwich, the amount of declination or variation on each line may be obtained. For any place situated on one of these lines the value of the variation will thus be at once ascertained, and for a place situated between any two lines a proportional variation may be made from the values found for the adjacent lines.

The variation of the magnetic meridian shows the necessity of recording on all colliery plans the date and amount of declination for the guidance of future working. Many accidents can be traced to the neglect of doing this at the time of drawing up the plan. The date of an old plan may be approximately ascertained by looking at the meridian line on it ; thus, if a plan is found having a meridian with a declination of $21^{\circ}00'$ west of the true north, we would say that it was probably made about the year 1864, the variation for that year having been $21^{\circ}03'$ at Greenwich. The needle is also subject to diurnal fluctuations.

Diurnal Variation.—In the afternoon the needle is drawn a few minutes to the west and in the early morning a few minutes to the

* *Colliery Manager's Pocket Book*, 1898, p. 151.

east. The needle stands at its mean position a little after 10 a.m. and a little before 7 p.m. The variation is greater in summer-time than in winter, but seldom exceeds about 10 minutes. This variation must be taken into account when making surveys in which accuracy is required, but under ordinary circumstances, such as the quarterly extension of colliery plans, the variation is too small to cause any appreciable error.

The declination of the needle may be due to various causes, such as :—

- (1) The presence of masses of magnetic rock.
- (2) Induction currents.
- (3) Magnetic storms.

The chief mineral that attracts the needle is magnetite, which is found in masses, veins, and sands. Tops of mountains in the Northern Hemisphere attract the north-seeking end of the needle. Beaches are often covered with magnetic sand; for instance, in Rothesay Bay there is a large bed of this description. Other minerals that attract the needle are magnetic pyrites and native iron. All crystalline rocks contain minerals which deflect the magnet more or less, and rocks which are dark-coloured do so more than others, as such rocks usually contain a fair proportion of iron compounds. Sometimes the needle is affected in positions where no such rocks are visible, a notable instance of this being at Melton-Mowbray, where there are no igneous rocks in the immediate neighbourhood, although the needle is deflected through as much as 67° , which may be due to the presence of large masses of basaltic rock lying beneath the surface.

Prof. A. W. Rücker,* speaking of this source of attraction, says:—"Magnetic rocks are often permanently magnetized, and may deflect a compass held close to them through 40° or 50° . The permanent magnetization is, however, irregular, and at a comparatively short distance the disturbing effect appears to be due almost exclusively to the uniform magnetization produced by the earth's magnetic field."

The general conclusions which he comes to, both as regards theory and experiment, are:—(a) "That dykes and thin uniform basaltic sheets produce no measurable effects except at distances from their edges, which are small multiples of their thickness; (b) That isolated masses of trap-rock, a few square miles in area, produce no important magnetic effects at distances comparable with their linear dimensions."

Induction Currents.—Induction currents may be produced by the neighbourhood of currents or of magnetic bodies inducing them, *i.e.* rocks containing iron or its compounds.

* *Trans. Inst. Min. Eng.*, vol. ix. p. 420.

Magnetic Storms.—The irregular fluctuations of the needle which occur from time to time proceed from magnetic storms. The word storm in this connection does not necessarily imply violent action, although occasionally the needle oscillates backwards and forwards. As a general rule, however, the needle becomes temporarily deflected some degrees beyond the normal variation over a large area, sometimes for 24 or 48 hours. Magnetic storms are invariably connected with displays of aurora, and, in some unexplained manner, with the occurrence of spots on the sun, and, as shown by Prof. Balfour Stewart, they commonly occur at intervals of ten or eleven years, when the sun spots are at their maxima. If the compass is used during one of these storms the needle sometimes becomes practically worthless for the time being.

Influence of Rails on the Needle.—The presence of iron or steel may cause the needle to deflect. Cast-iron in small quantities does not seem to have much influence, and those used underground at many Scotch collieries (weighing 28 lbs. per yd.) have so little effect that the surveys of those mines are almost invariably made with a free needle. Malleable iron rails have a more appreciable effect. With rails weighing 12 to 15 lbs. per yd., the deflection is seldom greater than 2° when the compass is placed *in the centre of* the two rails, and about 3 ft. above the ground.

Steel rails of the same weight exert more influence, and the needle is often deflected over as much as 10° to 15° in the presence of such rails, so that special precautions become necessary in making observations. Steel ropes and tools will also affect the needle, and in order to get a true reading the compass should be planted at least 15 ft. from any metal, while the presence of a large mass of metal necessitates its being placed still further away. In highly inclined workings, with the dip towards the north, it frequently happens that if a road is driven in the direction of the magnetic meridian, the rails become strongly magnetized on account of the earth currents passing longitudinally through them. Rails in such a position influence the needle to a much greater extent than would otherwise be the case.

Surveying in the presence of iron may be carried out with approximate accuracy if readings are taken at every station instead of at every second station, and by taking back and foresights. In these circumstances, both back and foresight readings will be subject to the same degree of error, and, consequently, if a correct bearing of any particular line can be obtained, a fairly accurate survey can be made.

Dip of the Needle.—A magnetic needle does not, in our latitudes, assume a perfectly horizontal position. The inclination of a freely suspended needle is about 67° at Glasgow towards the north. The dip varies in the same way as the declination, and ranges from 75° , the maximum, to 66° , the recorded minimum at Greenwich. In mining dials this tendency is counterbalanced, so that the needle moves only

in a horizontal plane. All needles have not exactly the same variation, and for this reason it is very important that magnetic bearings made at the surface should be taken with the same compass that has been used for the underground surveys.

The length of needle in common use for underground work is $5\frac{1}{2}$ in. It consists of a strip of steel with an agate or ruby centre, and a cross-cut line marking the north-seeking end. In some cases a small vernier is fixed to one end, so that the bearing may be read to the nearest minute.

The dial plate is usually constructed of brass, although in some instances aluminium has been used with success. The figures may be marked on this plate in various ways, according to the practice of different makers. The method of reading will be explained later.

Determination of True Meridian.—To determine the true meridian, and from this the exact amount of magnetic declination, various methods may be employed, the best two being to make observations of the Pole star or of the sun.

Observations of Pole Star and Sun.—This method is accomplished by using a theodolite and bringing the telescope of the instrument to bear upon the Pole star at stated intervals before and after its culmination. A reading is taken several hours before its culmination, both on the horizontal and vertical circles of the theodolite. After a lapse of the same period subsequent to its culmination the star is sighted a second time, and a reading taken as before. The line bisecting the angle obtained by the two readings will indicate the true meridian. If the observations are taken during winter, which is the best time, the readings are taken at an interval of 11 hours 58 minutes for the Pole star. Similar methods are resorted to of finding the meridian by means of observations of the sun.

Setting out the Meridian Line.—In every mining district a permanent meridian line should be set out, so as to enable surveyors to determine the true meridian at any time.

The best method of permanently marking out the meridian line is to insert in the ground, 4 or 5 ft. deep, a large hard stone, granite, if possible, 6 or 8 ft. long $\times$ 2 ft. broad. The stone should be well faced and firmly set in cement to keep it from shifting. In the upper surface of the stone is fastened a brass plate, a foot square, let in so as to be perfectly horizontal, and on this the meridian line is shown by a fine engraved line. For practical purposes a point may be fixed at some part of the royalty, and a line laid out connecting it with some permanent point, such as the centre of the shaft, and the bearing of the magnetic meridian with this line recorded every year or every second year; the difference in the readings will be the amount of the annual variation.*

* *A Treatise on Mine Surveying*, p. 50.

A very convenient method is to fix a pin in the ground on some part of the colliery which is unlikely to be affected by the proximity to underground workings, and to sight from this point a number of permanent objects within range, such as steeples, factory chimneys, etc. The mean of the angles made by the axis of the magnetic, and the imaginary lines connecting the point with the objects selected, determines the variation in the meridian in a more satisfactory manner than by relying on observations in one line only.

To ascertain if there are any local attractions affecting the needle, a number of observations should be made on a straight line, the correct bearing of which is known.

Colliery Plans.—A number of plans are published by the Ordnance Survey Department which are of great use in mining operations. The smallest are on a scale of 1 in. to the mile, showing the roads, railways, and chief land marks, such as farmhouses, plantations, streams, etc. The next size is 6 in. to the mile, showing every detail, such as fences and other boundaries. On these plans the levels are also marked, showing the height of various points above the mean sea-level at Liverpool. The positions at which the levels have been taken are indicated by dots, and the height marked beside them in figures, thus $\odot$ 683·3. Where these observations have been made, *bench marks*, as they are called, are cut on the walls of buildings, pavements, etc. Their positions are indicated thus: 

Contour lines are also indicated on these plans by means of dotted lines. These contour lines indicate that each point on the line is at the same height (given in small figures within the areas thus outlined) above sea-level.

This information is very useful to the mining engineer in aiding him to determine the levels of the various pits and bore-holes which may be put down on an estate, and in enabling him to arrive at a conclusion as to the best position in which to sink new shafts or to lay down sidings and other works. The 1 in. and the 6 in. maps can be obtained coloured, showing the geological formation of the various coalfields and the outcrops of the various seams, the depths of the pits sunk, the position and extent of faults, where proved, and much other useful information. The largest scale plans published by the Ordnance Survey is 25·344 in. to the mile, or $\frac{1}{2500}$ of the actual size of the district surveyed. These plans are very accurate, and show every surface-feature that was in existence at the time of the survey, together with the bench marks, etc. Unfortunately, the paper on which the maps are printed is found to contract and expand considerably, and this may sometimes lead to an error, but that may be guarded against to a certain extent by using the scale printed on the plan itself in preference to a detached scale. On these sheets every enclosure is numbered; and a book of reference to these numbers is published for each parish represented, giving the

area of these enclosures and the nature of the ground, such as pasture land, wood land, gardens, etc., etc. In the latest editions of these plans the area of each enclosure is marked.

The levels marked on an Ordnance Survey plan are not always correct for colliery districts, as the continual subsidence of the underground workings affects the surface level. As an instance of this it was found recently, when a new survey of one of the Scotch coal-fields was made, that nearly every bench mark had subsided from 5 to 7 ft.

The Ordnance Survey also publish vertical sections of the various coalfields, corresponding with lines shown in the plans, showing the seams, and horizontal sections showing the nature of the intervening strata.

The smallest scale permitted for colliery plans, by the provisions of the Coal Mines Regulation Act, is $\frac{1}{2}$ inch corresponds to 1 chain. Sometimes scales of 1 in. to 66 ft. are used, but a plan of an extensive royalty drawn to that scale becomes inconvenient and unwieldy to work from. In preparing a colliery plan the first step is to make an accurate survey of the surface boundaries and determine the position of the various shafts and surface buildings. This is usually done with a theodolite. The details of fences, etc., may be filled in by enlargement from the 25 in. Ordnance Survey maps; but as many of these plans are very much out of date, it is often necessary to make a complete new survey of all the surface lines within the boundary. For this purpose the theodolite may be used, but where the details are somewhat intricate it is more expeditious to use the mining compass. A plan which has merely been enlarged from the 25 in. scale without check surveys cannot be relied upon when workings are being carried on near the boundaries, but it may be useful as a general plan to guide operations.

The information which should be shown on colliery plans is as follows:—All underground workings showing the position of coal faces and the position of all pillars of coal; faults and dykes showing the direction and amount of throw; all the air and water-courses and the haulage roads. The thickness of the coal should be marked at frequent intervals, and special care should be taken to indicate clearly what has been found in exploration workings. The dip and rise of the strata should be indicated with an arrow and the rate of inclination marked thereon. Levels should also be marked at various points, showing the height above a given datum line, one, two, or three thousand feet below sea-level being convenient datum lines.

The date of the survey of all working faces should be neatly marked and the workings should be surveyed and extended in ink every three months. The position of all shafts should be shown and the depth marked, and a section must be given showing the strata

sunk through, or, if that is not practicable, one showing the average thickness of the coal and the nature of the roof and pavement. The surface lines are usually drawn in with Indian ink, and the boundaries indicated by an edging of colour.

To indicate the underground workings, colour is generally used instead of Indian ink, a separate colour being used for each seam. Faults and troubles should be shown distinctively coloured. When two or more superimposed seams are worked, and require to be shown on the same plan, each seam is coloured differently, as stated above, but it is far better to make a separate plan for each seam worked. In pillar and stall workings only the road need be coloured, while the solid is being worked, but after the pillars have been removed it is usual to colour the waste or exhausted area and show it by hatched lines.

In longwall workings the waste only need be coloured and the roads left uncoloured. Roadways going through solid coal are generally shown by unbroken lines, while those going through waste, as in longwall, are usually shown with dotted lines. The two commonest colours used in making plans are red and blue. On surface plans water is shown blue, roads 'Burnt Sienna,' and buildings indicated in a wash of Indian ink. It is generally found sufficient to colour the roads, buildings, and water.

All colliery plans should be drawn on strong mounted paper, and great care should be taken that the paper is thoroughly seasoned before being used. (See *Note*, p. 765.)

Underground Surveying.—For the great majority of underground surveys the instrument used is some form of mining compass. In cases where there is no disturbing magnetic influence, such as in sur-

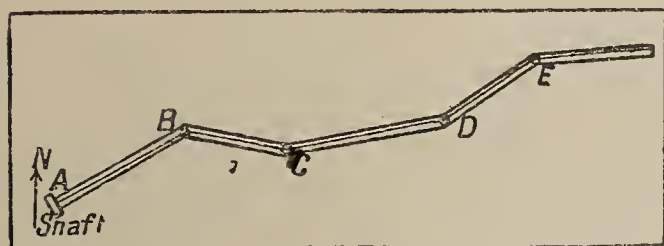


FIG. 720.

veying the roadway shown in fig. 720, the compass is first planted at B and then carefully levelled, the needle being allowed to settle. A lamp is now held at A in the centre of the shaft, or on one of the guides, and the sights of the compass rotated until the hair cuts the light. The bearing is then read off and recorded in the survey book, the measurement being taken at the same time by the assistants, and likewise recorded in the book. By this time a man has gone forward to the point C, and places there a lamp on the floor as near to the centre of the road as possible.

The bearing from B to C is now taken and recorded in the same way. The surveyor then lifts the compass and plants it at D, and takes the bearings D C and D E in the same way as before. There are various ways of booking a survey like this, two of the commonest methods being here shown.

First Method:

No. of Bearing.	Bearing.	Distance Links.	Remarks.
(1)	N. 70 E.	100	From centre of No. 1 Pit.
(2)	S. 85 E.	126	
(3)	N 75 E.	167	
(4)	N. 5 E.	122	

Second Method:—With the second method it is desirable to make a small sketch at the same time as the bearings are being written down, so as to give the surveyor a general idea of the ground he is surveying (fig. 721).

This is a very satisfactory and expeditious method of surveying, provided the surveyor has had sufficient practice to sketch neatly. There are one or two points of which note must be taken.

Take bearing No. 2, which reads S. 85 E. A common mistake among learners in marking a bearing like this would be to record it as N. 85 E., and a mistake like that is almost impossible to check when the survey comes to be plotted. Take now the bearing B C, as shown on fig. 722 ; whichever end of the needle may be looked at there is only the one figure which can be read off, viz., 85°, and this is accordingly marked in the book. Similarly the last bearing might have been marked N. 5 W., whereas in the other case it is simply marked 175°, and it is hardly possible to make a mistake. The only mistake which is liable to occur with this latter method is

that such a bearing as D E might have been sketched as turning to the right instead of to the left. A mistake like that, however, is easily detected when plotting, and with a little care underground in watching the general direction of the road, whether towards the north or towards the south, this source of error should be easily avoided.

It occasionally happens that a few bearings have to be taken at

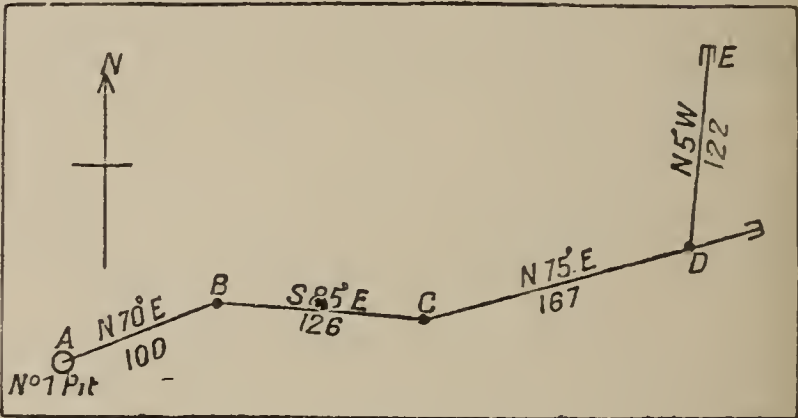


FIG. 721.

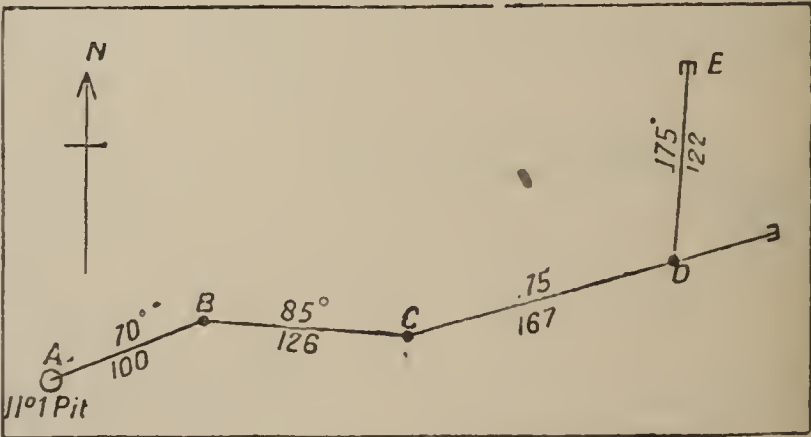


FIG. 722.

some point where magnetic attractions are exerted, and that a free vernier compass cannot be employed for the purpose. To do this with the loose needle the method usually adopted is shown in fig. 723.

Planting the compass at B, the bearing reads 60° from B to A and 90° from B to C. Planting again at C, a back sight is taken to B and reads 83° , and a foresight to D reads 135° . Similarly at D the back sight reads 133° and the foresight 70° , and at E the back sight is 73° and the foresight is 92° . At E there is no iron to attract the needle, but as a check back sights are taken at F, and as this also

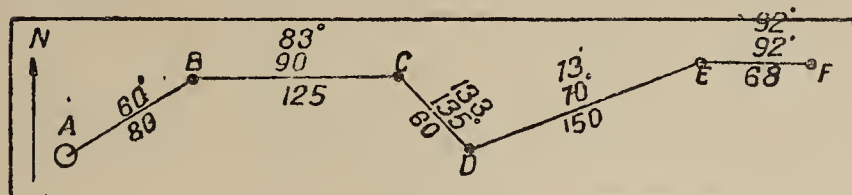


FIG. 723.

records 92° it indicates that the bearing taken from E is correct. Now the bearing from E to D is 73° , and this bearing having been taken from a point where there is no attraction is bound to be correct. When the compass was at D the bearing from D to E read 70° , therefore the amount of attraction at D is $73 - 70 = 3^\circ$, to be added to the observed bearing, so that the bearing 133 taken from D to C is 3° wrong and should be 136° . In the same way at C the correct bearing C to D is 136° and the compass reads 135° ,—that is, 1° to be added to the compass bearing, so that the correct bearing C to B is $83 + 1 = 84^\circ$. Again, at B the compass reads 90 from B to C instead of 84° , so that 6° must be *deducted* this time from all bearings taken at the point B, and the correct bearing from B to A becomes 54° .

The survey would now be plotted :

A to B 54° , B to C 84° , C to D 136° , D to E 73° , and E to F 92° .

The student should never think of seeking the *average* bearing of the back and foresights in a survey of this kind, as such a method would be quite incorrect, and a careful surveyor would endeavour to secure a sight free from magnetic disturbance at each end of the survey, so that the one may be checked by the other.

The Fixed Needle or Vernier Compass.—This instrument may be used in two ways underground. The first method is by taking the angles between each two bearing lines by placing *the vernier at zero* at every station, as is sometimes done with the theodolite when taking a surface survey. This method of surveying is, in the writer's opinion, quite unsuitable for general underground work, and will not be further described.

The second method is to take the bearings and book them precisely the same as if they were loose needle bearings.

Let fig. 724 represent a working which is to be surveyed with the vernier along the road from A to F, which is laid with steel rails. Suppose that at B¹ there is an old roadway without rails, and where the compass can be planted at a sufficient distance back from the

main road to avoid disturbing influences. The first operation is to set up the compass at B^1 . Fix the arrow on the vernier plate opposite zero, then unclamp the needle, and after allowing it to settle turn the dial round until the needle also points to zero. The clamp screws are then tightened to secure the compass to the tripod and the clamping screw of the vernier slackened. If the dial is now turned round, the arrow on the vernier

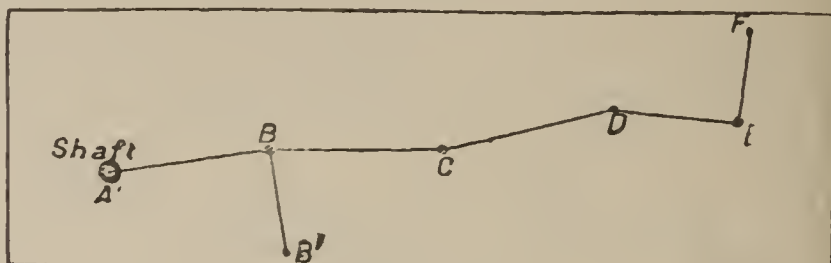


FIG. 724.

plate still remains pointing to the north, as it will be moving along with the north end of the needle. The sights are now turned to cut the light held at B and the vernier screw is then tightened up. Before lifting the compass a lamp is placed vertically below the centre, and for this purpose a brass plumb line with a thin flexible cord is used. The instrument is now lifted and planted exactly above the lamp at B, and carefully levelled. The lower screws of the tripod are loosened and the compass turned to sight back to the lamp at B^1 . When this has been done the lower screws are again tightened up. It is evident now that the compass, as planted at B, is lying in exactly the same line as it was when placed at B^1 and the arrow on the vernier plate still points to north. The vernier clamp screw is now slackened and the bearing B to A taken and recorded in the survey book. The bearing B to C is also taken, and while looking through the compass to C the vernier clamp screw is tightened. The compass is again lifted forward to C, and before the vernier is slackened the back sight towards B is taken, so that the vernier still remains pointing to the north. The bearing C D is now taken. In the same way the bearings D E and E F are also taken. Suppose F is at the coal face, and that it is possible to obtain a clear bearing at that point. The vernier may now be checked to see if it still corresponds with the needle. If it does the survey may be taken as correct, but the student will very often find that his work has not been done with sufficient accuracy, and that the needle does not coincide with the vernier. If the work is important a new survey must be made, but in many cases it will be sufficient to make an allowance for the error when plotting the survey. For instance, if the last bearing E F in fig. 724 is 1° wrong, then this error may be divided among the total number of bearings (which in this case is five), so that 12 minutes would require to be added or deducted, as the case may be, in each case. It is not always possible to get a clear bearing at the commencement of the survey. In such a case the survey may be taken from the face outwards, or if that is not convenient then a start may be made at the pit

bottom the vernier being put at the incorrect north indicated by the magnetic needle, and the survey made inwards in the usual way until a clear bearing can be obtained. If at this point the needle is, say, 5° different from the vernier, then this difference must be allowed on

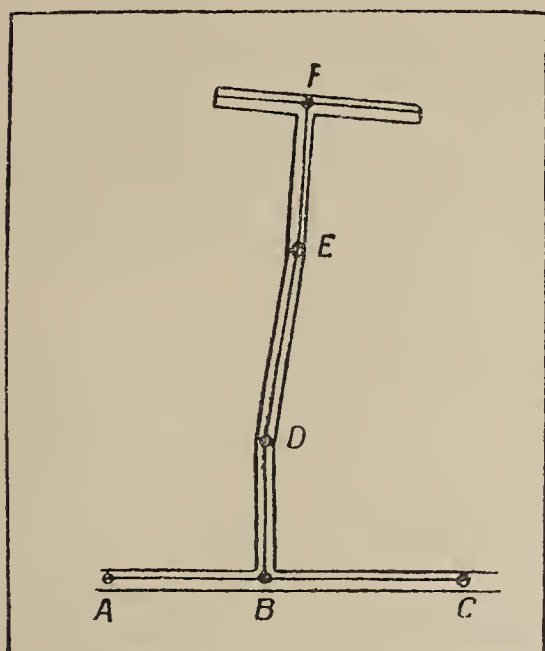


FIG. 725.

each bearing up to the point where the clear bearing was obtained. The vernier is then adjusted to coincide with the magnetic needle, and the survey proceeds as before. When a side road, off the main road, has to be surveyed with the vernier it is necessary to leave in two marks on the main line of survey. Suppose at B, on fig. 725, the road B D E F has to be surveyed. The bearing A B is taken and a mark left at A, the bearing B D is also taken and a lamp planted below the compass at B. The instrument is now shifted forward to D, a back sight taken, and the position of the lamp at B is now carefully marked and

the bearing D E taken, and so on, till F is reached. The compass is now brought back and carefully plumbed over the mark left at B and the vernier fixed at the bearing A B, as before obtained. The back sight is now taken to A, the vernier unclamped and the foresight taken to C, the survey proceeding in the usual way. To insure accuracy when working with the vernier a good plan is to use two tripods. The spare tripod is planted at the point where the foresight is to be taken, and a lamp is held exactly at the centre. After taking the sight the compass is lifted off its tripod and carried forward and placed on the spare tripod, and the back sight taken to a light held exactly at the centre of the last tripod. The surveyor has then to wait until this tripod is brought forward and planted in the position for the front sight. The learner working with the vernier often makes mistakes by slackening the wrong screws, with the result that the whole work has to be done over again. Assistants, too, often lift the lamp from the back sight point before it has been looked at, with the result that the base line for the forward sight is destroyed, and part of the work has to be repeated. These mistakes should be carefully avoided.

Measurements.—In surveying, lengths are usually measured by the imperial chain, which consists of 100 links, each link being 7.92 in. long, and therefore its total length is 66 ft. By adopting this unit of length, the chain is made to bear certain ratios to other standard measurements; for instance, $\frac{1}{4}$ chain is equal to 1 pole, and 1 mile equals 80 chains, and 10 chains = 1 furlong. Again, 1 acre =

4840 sq. yds., and $22^2 = 484$ yds. = 1 sq. chain, therefore we have 10 sq. chains in 1 acre, and as 10 sq. chains are equal to 10000 sq. links, 100,000 sq. links are equal to 1 acre.

Measurements are also made by chains 100 ft. in length; these chains are largely used in civil engineering work, and also in metal mining districts. By using such a length of chain, measurements can be made much more rapidly, and are easier dealt with for surface surveys. The unit of measurement in ore mines is usually the imperial chain, but very often the unit is 1 fathom and a chain used 10 fathoms long. On the Continent the unit is 1 metre, which is equal to 39.37 in.

The best chains are made of the finest mild steel, as with this material they can be made light, and are less easily bent.

There are many sources of inaccuracy which arise from using the chain, and great care has to be taken to avoid them. The chain should be frequently checked by some standard mark, as it is very liable to become stretched, or some of the links may be bent when it becomes too short.

In most large towns standard measurements are marked with brass plates on some public building. The chain should be checked at one of those standard marks, and similar permanent marks should be laid out at the colliery, so that the chain may be checked at any time and always kept correct.

Care must also be taken when measuring that none of the links are interlocked, as this would also cause considerable error.

Another method of measuring is by means of steel tapes. These tapes are largely used in some American mines. In Pennsylvania, tapes 300 to 600 ft. long are employed. Measurements taken in this way are likely to be more accurate than with a chain, but steel tapes must not be allowed to kink, as they are then very apt to break. The difficulty connected with tapes is the reading of them in muddy and wet mines, where they get rusted. To obviate this, small bits of brass wire, with a certain number of nicks or cuts in them, are soldered on at regular distances apart.

For very correct measurements, surveyor's rods are used; these are made of lancewood, in pieces about 5 ft. long, joined together by a 'scarf' joint. They are often used to check the length of the other measuring instruments, and are also useful for marking out a standard chain. Glass rods have also been used on important government surveys.

Measuring on the Slope.—In chain work on the slope, corrections must be made to get the proper horizontal measurement. There are several methods of doing this, such as: (1) by ascertaining the angle of inclination; (2) measuring along the slope and making deduction from every measurement; (3) by 'stepping' the chain. When incline measurements are required to be accurately made, the angle of inclination is ascertained by a clinometer or similar instrument,

and the actual horizontal measurements can be ascertained by calculation from tables compiled for the purpose. Many mining compasses are fitted with an external arc for taking the angles of inclination simultaneously with the horizontal angles.

The method that is most largely adopted by surveyors is to measure along the slope and make deductions for each length, according to the inclination, the amount of deduction necessary being derived from the tables. In the absence of these, the following approximate method may be employed.

Example.—A seam dips 1 in 8; find the horizontal lengths corresponding to the following measurements, 360, 470, 832.

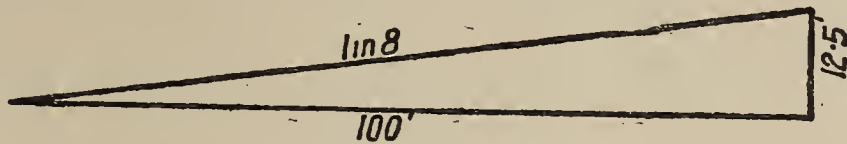


FIG. 726.

If L is the horizontal length required, D the measured distance on the slope, and d the dip, $L = \sqrt{D^2 - \left(\frac{D}{d}\right)^2}$

$$\begin{aligned} \therefore (1) \quad L &= \sqrt{129600 - 2025} = 357; \\ (2) \quad L &= \sqrt{220900 - 3364} = 465; \text{ and} \\ (3) \quad L &= \sqrt{692224 - 10816} = 825. \end{aligned}$$

In the above examples decimal fractions have been disregarded, although in practice they must be considered, or the result may lead to error when considerable distances are involved.

Stepping the Chain.—This is a method sometimes adopted to ascertain the horizontal measurement on inclined workings, and is very expeditious, but only approximate results can be obtained, and the method should not be adopted on gradients of more than 5° ; when the inclination is greater it is too inaccurate.

The method of 'stepping' the chain will be understood from fig. 727. The chain should be taken in not more than half lengths or

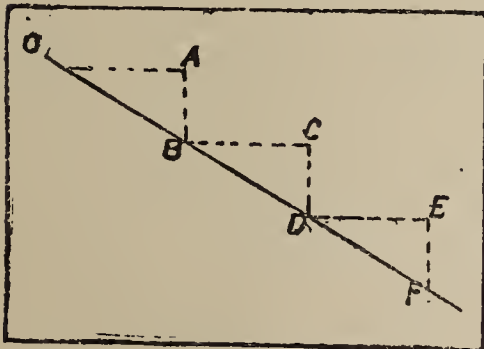


FIG. 727.

quarter lengths, and stretched out firmly to A, say, and a cord and plumb-bob dropped vertically to B, the distance OA being then carefully measured. The same process is repeated at C and E, the distance BC and DE being found in the same way. The vertical distances AB, CD, EF need not be measured, unless they are wanted as a check on the work.

Plotting the Survey.—The lines and angles of a survey are usually represented on paper on a very small scale compared to the original lengths and angles. The process of

putting these lines and angles on a plan is termed plotting. The plotting of surveys may be divided into three different methods, viz., by means of protractors, by rectangular co-ordinates, and by chords. The first method is the one most largely used for colliery work, and can be done very accurately with a good protractor.

Protractors.—There are various kinds of protractors used for this purpose, the commonest sort being a half circle with or without a movable arm.* They may be made of brass, ivory, celluloid, or white metal. The semicircle is divided from 0° to 180° in opposite directions. To plot the survey the straight side of the protractor is laid along an assumed meridian line (fig. 728), and the bearings pricked off by making a mark with a fine pencil or needle at the desired angle and numbering the bearings as they are taken off. All bearings say to south-east and north-east will be first taken off and

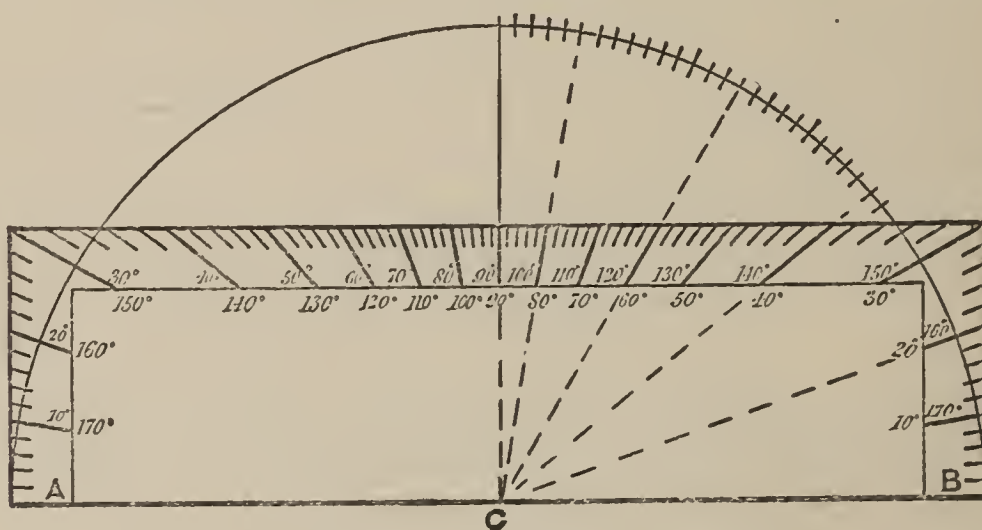


FIG. 728.

numbered; the protractor will then be reversed and all the bearings to south-west and north-west pricked off and likewise numbered. In laying the protractor on the meridian line its centre should be placed on some fixed point, such as the intersection of a line drawn at right angles to the meridian line, which would, of course, represent an east and west line. The first line may then be taken from this centre point to the first bearing and the distance marked off, or it is sometimes more convenient to parallel the line to a different part of the paper, clear of the bearings altogether. For this purpose two set-squares or a parallel ruler may be used, but the latter should not be used unless it is of fair size and provided with rollers. The disadvantage of this method of plotting is that the plan gets spoiled by the large number of pencil or needle marks that would be required. For office work the semicircular protractor is often provided with a movable arm and a small vernier, which enables it to be used with

*The protractor shown in the figure is made of boxwood. A more reliable instrument is a semicircular protractor made of horn.

great accuracy. A long steel 'straight-edge' is laid along the meridian line, and the protractor is fitted on to this with its straight side against the straight-edge. The bearings and lines can then be taken off rapidly and accurately by means of the movable arm, and without requiring to prick them off or mark them on the plan.

The circular protractor divided to the full circle is more accurate and convenient for plotting a survey. It may be used in much the same way as the semicircle. When provided with a swinging arm the bearings can be taken off very rapidly and accurately. Another method of plotting is to use a large cardboard protractor and parallel ruler. This cardboard protractor is usually 12 to 18 in. square and divided from 0° to 360° , numbered in opposite directions. The centre portion of the cardboard is cut out, and the north and south line made to coincide with the plan meridian. The parallel ruler is placed on the required bearing and transferred direct to the plan. This saves much time and keeps the plan free of pencil marks, which are necessary with the ordinary circular protractor. The bearings can be taken very accurately, owing to the large diameter of the protractor. In the making of the Government Ordnance Survey Plans such protractors are always used.

In Scotland a method of plotting, which can be very rapidly and accurately done by what is known as a 'table' protractor, is largely used in mining engineers' offices. This protractor consists of a large circle 2 ft. in diameter. This circle is fitted into a square wooden frame or table *a*, in which it revolves on a central axis *b*. On the inside revolving circle is fixed a circular sheet of paper or scroll plan, with the working meridian on it; this line is made to coincide with the north and south line on the protractor. A small brass plate *c*, with an arrow engraved on it, is fixed to the square frame, and the bearing required is made to coincide with this arrow, the line being transferred on to the paper direct by the aid of a T-square laid across the protractor. The construction and method of using this protractor will be understood from the illustration in fig. 729.

The bearings, after being plotted on these scroll plans, are transferred to the large colliery plans by tracing paper. A large amount of work can be done very rapidly by this method, and the large size of the protractor insures its accuracy.

Plotting by Co-ordinates.—Where surveys have to be plotted with great accuracy, the method of using rectangular co-ordinates is to be preferred to any other. * It consists in assuming two fixed *axes* crossing at right angles to each other at a fixed point or origin, and in calculating the perpendicular distances, or *co-ordinates*, of each station from these axes. If the true meridian has been ascertained, it may be made to represent one of the axes. In fig. 730, which illustrates this method of plotting, the north and south line, and east and west drawn at right angles have been taken to represent the required axes.

* *A Treatise on Mine Surveying*, by Bennett H. Brough, pp. 140–143.

Now every line that is taken in a survey will depart from the north and south line by a definite amount, according to its angle and distance, and similarly it will also depart a certain amount from the east and west line. The distance that the line departs from the north and south line is termed its *latitude*, and the distance that it departs from the east and west line is termed its *departure*. The latitude of a point may be defined as its distances due north or south of some fixed point. The distance that one end of a line is due north or south of the other end is called the difference of latitude of the

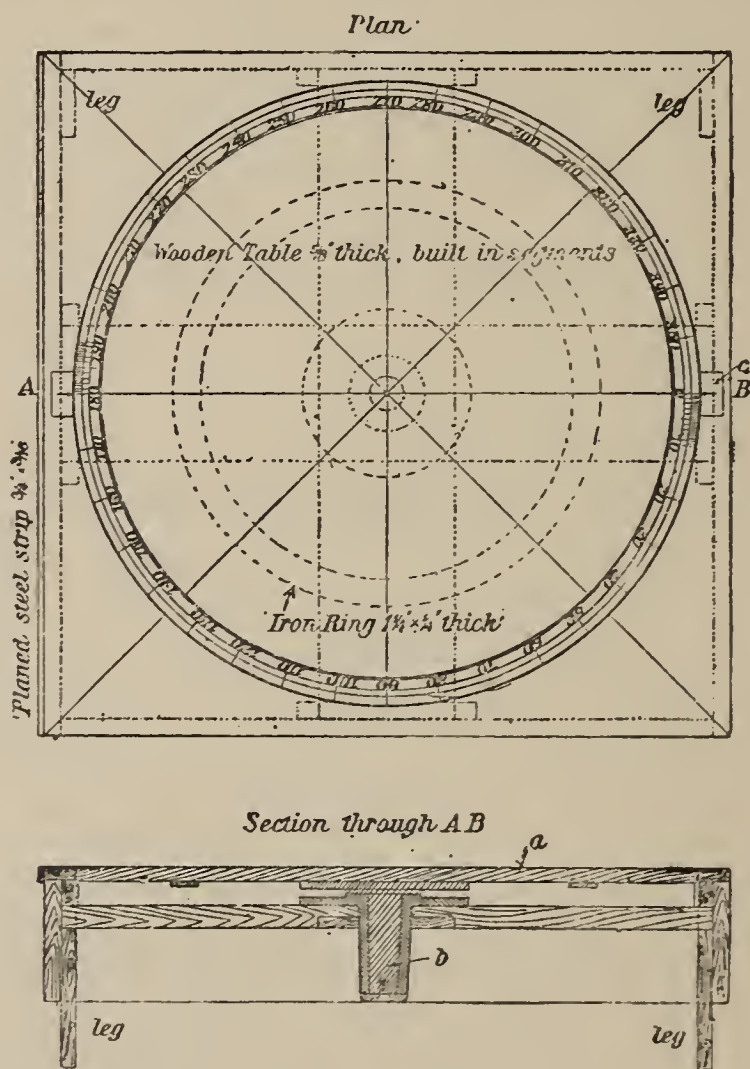


FIG. 729.

ends of the line, and similarly the distance which one end of a line is east or west of the other end is called the difference of longitude of the ends of the line or the departure.

The latitude = distance $\times$ cosine of bearing.
 „ departure = distance $\times$ sine of bearing.

To obviate the tedious process of calculating each bearing, the latitudes and departures may be taken from a book of traverse tables. The best book of tables for this purpose is that by R. Lloyd Gurden,

in which the tables are computed for every minute of angle. The data for the survey plotted in fig. 730 were as follows :

Bearing.	Distance. Links.	Latitude.		Departure.	
		N +	S -	E +	W -
N. 10° E.	100	98·5	...	17·4	...
N. 77° E.	130	29·2	...	126·7	...
S. 67° E.	134	...	52·4	123·3	...
S. 53° E.	136	...	81·8	108·6	...
S. 20° W.	42	...	39·5	...	14·4
S. 29° E.	132	...	115·4	64·0	...
S. 42° W.	86	...	63·9	...	57·5
S. 89° W.	78	...	1·4	...	78·0
N. 70° W.	110	37·6	...	...	103·4
N. 65° W.	106	44·8	...	...	96·1
N. 85° W.	47	4·1	...	...	46·8
N. 66° W.	120	48·8	...	...	109·6
N. 53° E.	113	91·4	...	65·8	...

Before these co-ordinates can be used for plotting, they must be reduced to total latitudes and departures. To effect this the northings and eastings are regarded as positive quantities, while the southings and westings are regarded as negative quantities, and the reductions are then computed by taking the algebraical sum at each station, which from the above data would be as follows :

No.	Total Latitude.	Total Departure.
(1)	+ 98·5	+ 17·4
(2)	+ 127·7	+ 144·1
(3)	+ 75·3	+ 267·4
(4)	- 6·5	+ 376·0
(5)	- 46·0	+ 361·6
(6)	- 161·4	+ 425·6
(7)	- 225·3	+ 368·1
(8)	- 226·7	+ 290·1
(9)	- 189·1	+ 186·7
(10)	- 144·3	+ 90·6
(11)	- 140·2	+ 43·8
(12)	- 91·4	- 65·8
(13)	00·0	00·0

To plot from this table, draw a north and south line, and an east and west line, making the junction of these two lines the starting point for the first station. From this point (A, fig. 730) take line No. 1 and measure off 98·5 northwards and 17·4 eastwards, making a mark at the end of these distances. From the end of distance 98·5 draw a line at right angles to north and south, and from the end of distance 17·4 draw another line at right angles to east and west, continuing this line till it meets the line drawn at right angles to north and south. Join the meeting point of these two lines to the point A, and this will give the first line of the survey. The same procedure is followed in regard to the other lines, measuring off all the positive latitudes to the north and negative latitudes to the south,

while the positive departures go to the east and negative departures to the west. The student will understand this method more fully if he plots down the above survey once or twice, and works out the numbers for himself. If care be taken when getting out the ordi-

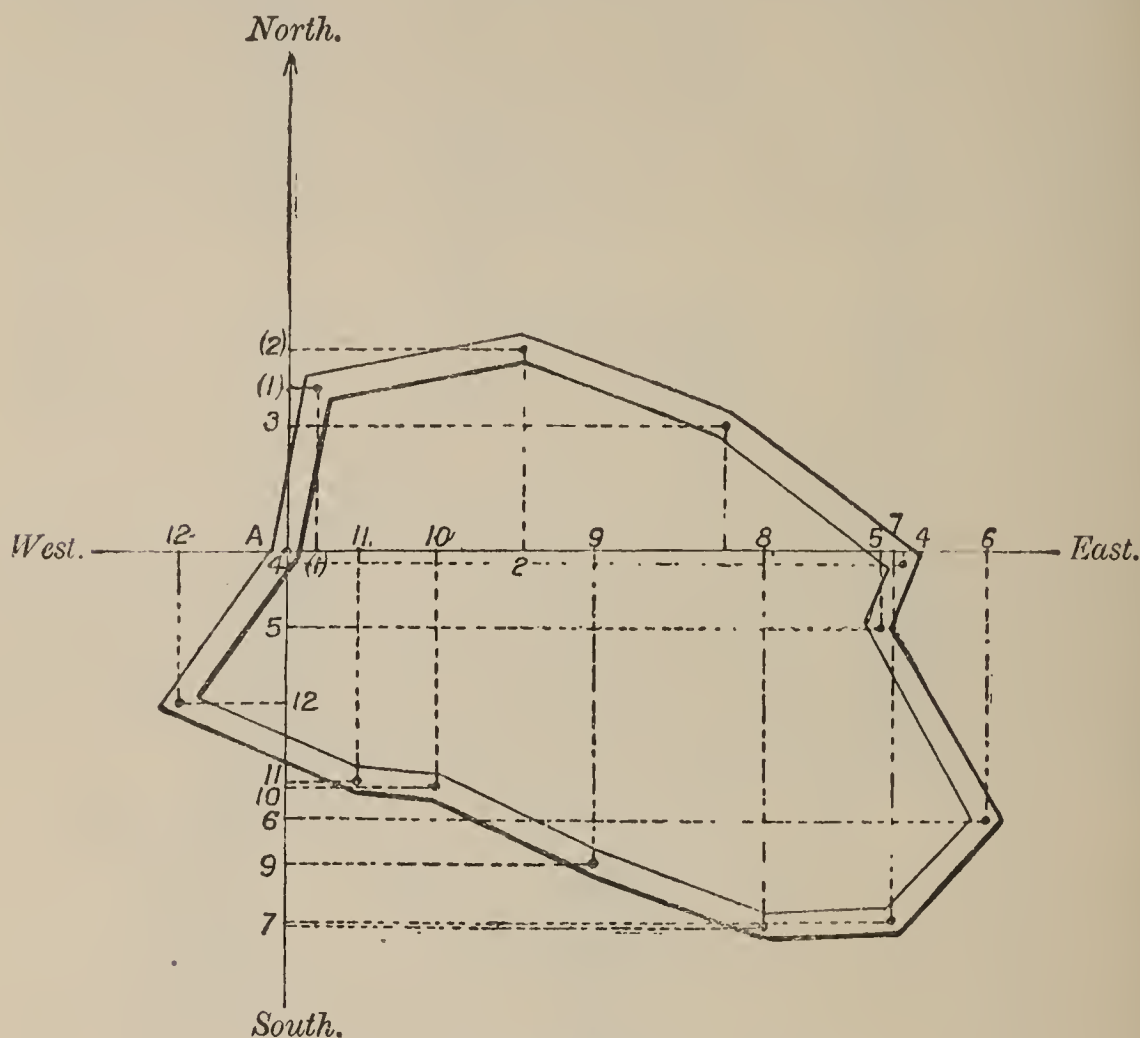


FIG. 730.—Plotting by Co-ordinates.

nates, this method of plotting insures great accuracy, and is specially useful for important surface or other surveys.

Plotting by Chords.—A tied survey may be plotted from one meridian, without the aid of a protractor, by using a table of chords.

No.	Bearing.	Chords, Radius = 1000.	Distance.
(1)	N. 10° E.	174	100
(2)	N. 77° E.	1245	130
(3)	S. 67° E.	1103	134
(4)	S. 53° E.	892	136
(5)	S. 20° W.	347	42
(6)	S. 29° E.	500	132
(7)	S. 42° W.	716	86
(8)	S. 89° W.	1401	78
(9)	N. 70° W.	1147	110
(10)	N. 65° W.	1074	106
(11)	N. 85° W.	1351	47
(12)	N. 66° W.	1089	120
(13)	N. 54° E.	892	113

In the survey already shown plotted by co-ordinates, using the same data, and from the table of chords given, we would have to draw a circle on the paper, with a radius preferably of 10 in., and through the centre of the circle draw a north and south line and an east and west line. With a pair of dividers and a scale, lay off the chords along the circumference of the circle according to the direction of the bearing. The chords being taken from a radius of 1000 would have

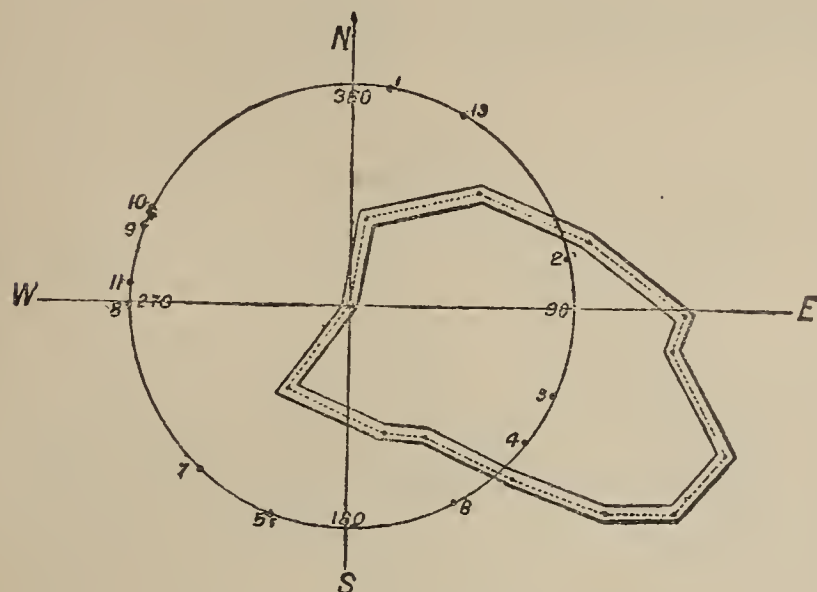


FIG. 731.—Plotting by Chords.

to be reduced by two decimal places to plot with a circle whose radius is 10, thus the first chord would be 1.74, the second 12.45, and so on. When the chords are all measured off, the rest of the work is performed in the same way as when a circular protractor has been used. This method of plotting is not to be recommended for colliery surveying, as

drawing circles repeatedly on the plan and pricking off the chords would soon break the surface of the paper and spoil it.

Calculation of Areas.—The three principal methods of calculating plan areas are: (1) by dividing the total area into triangles or squares; (2) calculating the area by means of ordinates; (3) by mechanical methods, such as measuring instruments. The areas of coalfields are nearly all calculated in acres, roods, and poles. The statute acre is equal to 10 sq. chains or 100,000 sq. links.

Calculation by Triangles.—The plan area may be divided into a number of triangles, either right angled or otherwise. If the triangles are right angled ones the calculations are much simplified. When the triangle possesses a right angle (fig. 732) the area

$$A = \frac{\text{base} \times pp}{2}.$$

If none of the angles are right angles, as in fig. 733, then an angle of 90° may be constructed by dropping a perpendicular pp from any apex on to the opposite side, and the area of the two triangles thus constructed may then be found separately. Or if we have two sides and know the included angle, as in fig. 734,

$$\text{then the area } A = \frac{ab \times \sin \alpha}{2}.$$

When none of the angles are known the following formula is used for calculating the area,

$$A = \sqrt{S(s-a)(s-b)(s-c)}$$

where S is the semi-perimeter or half the sum of the sides. If logarithms are used, the formula becomes

$$\text{Log } A = \frac{1}{2} \{ \log s + \log(s-a) + \log(s-b) + \log(s-c) \}$$

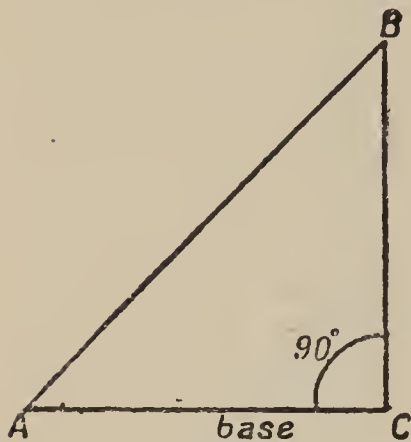


FIG. 732.

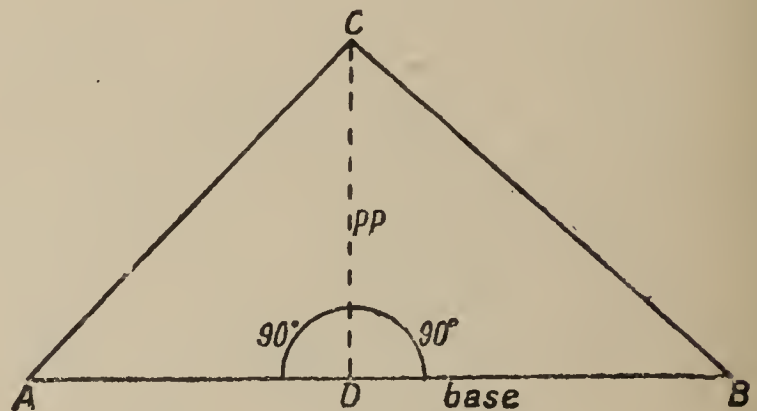


FIG. 733.

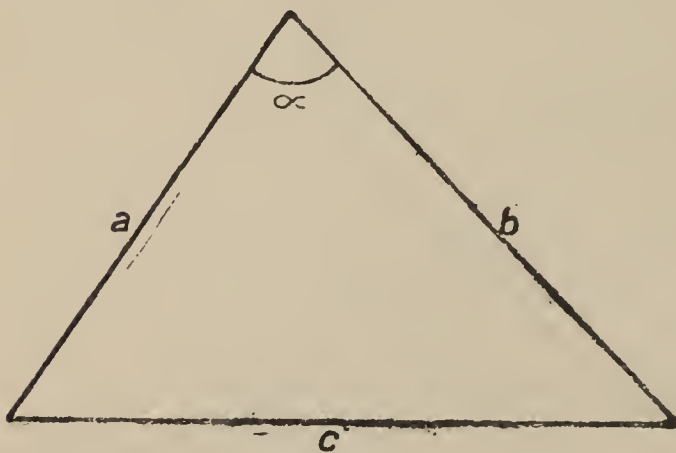


FIG. 734.

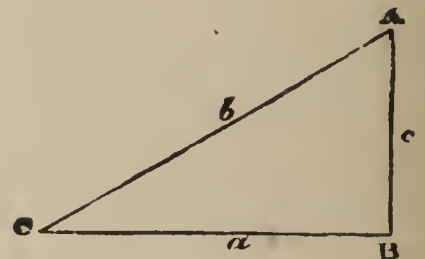


FIG. 735.

In the above triangle, if the respective sides a , b , and c are 1200, 1100, and 1000 links, find the area in acres, roods, and poles.

$$\text{Here } S = \frac{1200 + 1100 + 1000}{2} = 1650$$

$$\therefore A = \sqrt{1650 \{ (1650 - 1200)(1650 - 1100)(1650 - 1000) \}}$$

$$= \sqrt{1650 \times 450 \times 550 \times 650}$$

$$= \sqrt{265443750000}$$

$$= 515309.37 \text{ sq. links.}$$

To reduce this to acres, roods, and poles, count off five decimal places thus :—

5·15309

4

0·61236

40

24·49440

∴ Area = 5 acres, 0 roods, 24·49 poles.

Or by logarithms

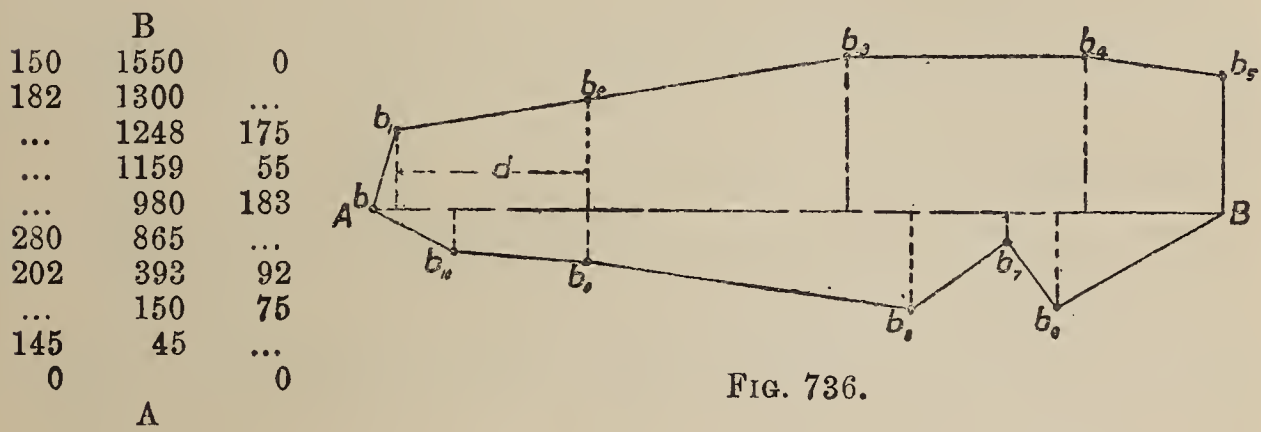
Log A = $\frac{1}{2}\{\log 1650 + \log(1650 - 1200) + \log(1650 - 1100) + \log(1650 - 1000)\}$

= $\frac{1}{2}\{3·2174 + 2·6532 + 2·7403 + 2·8129\}$

and Log A = 5·7119, and A = 515310·00 sq. links.

Calculation by Ordinates.—When the area to be calculated has an uneven outline, and is not of too great extent, the method of ordinates may be used. It consists in running a chain line or axis through the greatest length of the area to be calculated, and taking offsets at right angles to the chain line. The offsets should be taken sufficiently close so as to make each figure approximately a trapezoid, therefore the more offsets are taken the more accurate will the calculation become.

An example will show more clearly this method of calculating areas.



From the above data an irregular field was surveyed by running a chain line through it from A to B, and taking offsets right and left. By referring to fig. 736 it will be seen that the first offset on the left hand *b* is *o*, and the next *b*₁, 45 links along the chain line is 145, so that if we call the distance between the two offsets *d*, the area of this trapezoid = $d \left(\frac{b + b_1}{2} \right) = 45 \left(\frac{0 + 145}{2} \right) = 3262·5$ sq. links, and so on with the others, taking *b*₁ and *b*₂ for the next trapezoid. We may avoid the division by 2 for each trapezoid, by dividing the total sum of all the trapezoids at the end. Taking the offsets on the left hand

side first from A to B and the right hand from B to A, we get the following :—

$$\begin{aligned}
 (1) \quad 45 (0 + 145) &= 6525 \\
 (2) \quad 348 (145 + 202) &= 120756 \\
 (3) \quad 472 (202 + 280) &= 227504 \\
 (4) \quad 435 (280 + 182) &= 200970 \\
 (5) \quad 250 (182 + 150) &= 83000 \\
 (6) \quad 302 (0 + 175) &= 52850 \\
 (7) \quad 89 (175 + 55) &= 20470 \\
 (8) \quad 179 (55 + 183) &= 42602 \\
 (9) \quad 587 (183 + 92) &= 161425 \\
 (10) \quad 243 (92 + 75) &= 40581 \\
 (11) \quad 150 (75 + 0) &= 11250
 \end{aligned}$$

$$967933 \div 2 = 483966.5 \text{ sq. links.}$$

The area would therefore be 4 acres, 3 roods, 14.34 poles.

If the area to be calculated is large this process would be tedious, and areas with curved outlines can often be calculated with sufficient accuracy by the process of *equalising* or *giving and taking*; i.e. to draw a straight line through the curved boundary, leaving as much space outside the straight line as there is inside it. The whole area is then divided into triangles and calculated as already shown.

Calculation by Instruments.—This method is now much resorted to in mining engineers' offices for calculation of plan areas. The instrument for measuring such areas is called a *planimeter*, and for a full description of these the reader is referred to Brough's *Mine Surveying*, pp. 159, 160.

Levelling.—In connection with colliery operations a good deal of levelling is often required, both on the surface and underground. Levelling is defined as "the art of finding the difference between two points which are vertically at different distances from a plane parallel with the horizon."

To find the difference of level between any number of points three methods may be adduced :—(1) Trigonometrical ; (2) physical ; and (3) geometrical.

In *trigonometrical levelling* the lengths and angles have to be measured, this being usually accomplished by a theodolite, but it is a method not often resorted to, except under exceptional circumstances, such as when ascertaining inaccessible points on steep mountain sides.

Physical levelling is based on the change of atmospheric pressure at different altitudes from the centre of the earth, and is found by means of the aneroid barometer, which, as is well known, records the varying atmospheric pressure at different heights. With an ordinary barometer the mercury column falls on ascending hills on an average about 1 in. for every 900 ft. of ascent. This method is never practised for ordinary levelling, but it is exceedingly useful and much used by surveyors for ascertaining great altitudes where ordinary levelling would be impossible, or in making rough preliminary surveys in new and strange lands.

Geometrical levelling is the method most commonly employed by surveyors, and can be carried out with the different levelling instruments in use, or by the ordinary 'spirit level,' which forms a part of all such instruments.

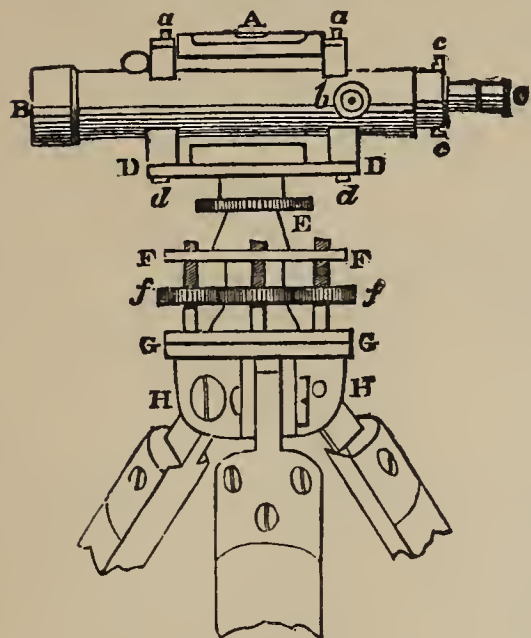


FIG. 737.

The instrument most largely used in Great Britain is that known as the 'Dumpy Level.' Fig. 737 shows the construction of this instrument. * A is the spirit level attached by screws at *aa* to the telescope B C. The small circle near the object end B of the telescope represents a small transverse spirit level used to show whether the cross wire of the telescope is truly horizontal. D D is a flat bar or oblong plate fixed on the top of the vertical axis E. To this bar the telescope is attached by adjusting screws *d d*.

The hollow vertical axis turns upon a spindle fixed to the upper parallel plate F, the spindle being continued downwards, and being attached to the lower parallel plate G by a ball and socket joint. There are four levelling screws *f*, by which the vertical axis is set truly vertical. The lower plate is screwed on the tripod head H. The tripod consists of three wooden legs similar to those on a theodolite. In some instruments a compass is carried on the top of the plate *dd* for taking the bearings of lines of trial sections. The telescope is similar to that of the theodolite, except that the diaphragm contains one horizontal wire and two parallel vertical wires, as shown in fig. 738. It is usually 9 to 14 in. in length, the lines within the telescope being filaments of spider's web.

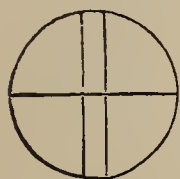


FIG. 738.

Adjusting the Level.—Before proceeding to level a section of ground the temporary adjustments of the instrument will have to be attended to each time the level is set up in a new position. The legs of the instrument should be firmly planted in the ground, and the parallel plates made as horizontal as possible. The levelling of the instrument is accomplished by placing the telescope with spirit level over one pair of the levelling screws, and adjusting the screws by turning both in the one direction either inwards or outwards with the thumb and forefinger, till the bubble in the spirit level is brought exactly to the centre; the telescope is now turned horizontally through an angle of 180° and put over the other pair of screws, and the spirit level adjusted as before. This may have to be repeated three or four times until the bubble in the spirit level rests exactly in the centre with the telescope turned into any posi-

* *Treatise on Mine Surveying*, p. 165.

tion. Fig. 739 shows the positions of the telescope during the adjusting process. Great care is required in levelling the instrument properly, and the operation should not be carried out too rapidly. The next operation is to adjust the telescope to prevent 'parallax,' that is, to move the eye-piece and to focus until the cross wires are seen with perfect distinctness.* Further adjustment may have to be made in sighting the levelling staff and repeated each time the staff is shifted. The nearer the object or staff the further the inner tube must be drawn out.

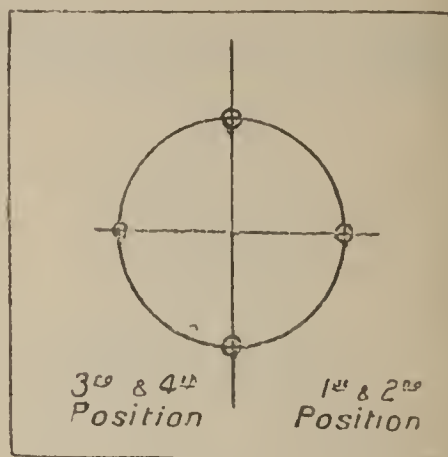


FIG. 739.

Levelling a Section.—Levelling is of two kinds, simple and compound. Simple levelling has only one line of collimation, whilst compound levelling entails constant changes of collimation. Simple levelling is resorted to when the difference in height between two points which are not far separated is required, and where one planting of the instrument will suffice. This is illustrated in fig. 740, where the instrument is placed as near as possible equidistant between A and B, the two points the difference of whose height requires to be determined. The telescope is first directed towards A, and a back

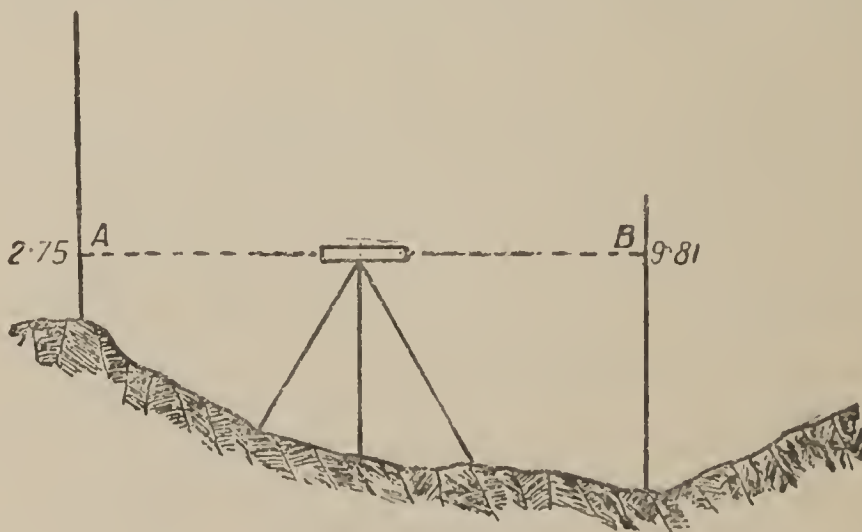


FIG. 740.

sight taken in which the line of collimation cuts the staff at 2.75; the telescope is then turned round and directed to B, and a foresight taken in which the line of collimation cuts the staff at 9.81; consequently there is a fall in the ground from A to B of 7.06 ft. When the foresights and intermediate are greater than the back sights the ground falls; if less it rises and will be deduced accordingly.

Compound Levelling.—When a long section of ground requires to

* Parallax means an *apparent change* in the position of an object caused by a change in the position of the observer.

be levelled, the undulations of the ground must be followed by varying lines of collimation, according to the rise or fall in the section.

This method of levelling is illustrated in fig. 741. The instrument is placed between A and B, and the reading of the back sight to

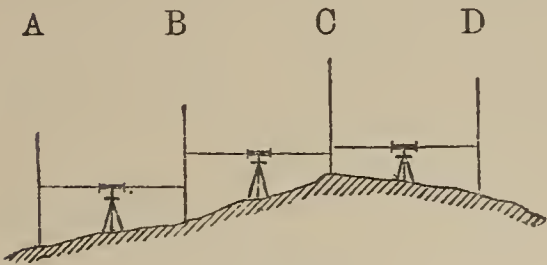


FIG. 741.

the staff is taken; the telescope is then turned and a foresight taken to B. The instrument is now moved forward to a fresh position between B and C and a back sight taken on to the staff B at the same position as that at which the last foresight was taken; the telescope is again turned and a sight taken

to C. The instrument is again shifted forward between C and D and a new line of collimation obtained, the same process being repeated till the whole section is levelled.

Datum Line.—In levelling, a datum line is generally selected for reducing the levels taken by the instrument to one fixed standard or relative height above sea-level. The Ordnance datum is “the approximate mean water-level at Liverpool,” and all levels marked on Ordnance Survey plans are the altitudes in ft. above this datum. It is not, however, necessary or usual to adopt the Ordnance datum. Any height above the Ordnance datum may be selected, but it is best to select the datum sufficiently low to be below the lowest point in the section levelled, so that in plotting the levels will all be positive.

Booking the Levels.—In booking the levels various methods are employed, but the commonest and best method is as follows:—

Back Sight.	Inter- mediate Sight.	Fore- sight.	Rise.	Fall.	Reduced Levels.	Distance.	Remarks.
					Datum.	Links.	
9·6	...	...	...	...	100·00	00·00	From centre of bridge.
...	8·30	...	1·30	...	101·30	100·00	
...	7·50	...	0·80	...	102·10	200·00	
...	6·32	...	1·18	0·16	103·28	300·00	
...	6·48	...	...	...	103·12	400·00	
...	5·63	...	0·85	...	103·97	500·00	
...	4·20	...	1·43	...	105·40	600·00	
...	3·20	...	1·00	...	106·40	700·00	
...	...	2·21	0·98	...	107·39	800·00	
8·70	7·10	...	1·60	...	108·99	900·00	
...	6·23	...	0·88	...	109·86	1000·00	Opposite engine house.
...	4·35	...	1·88	...	111·74	1100·00	
...	3·45	...	0·90	...	112·64	1170·00	Opposite shaft.
...	3·50	...	...	0·05	112·59	1200·00	2 ft. from outside rail.
...	3·30	...	0·20	...	112·79	1300·00	
...	3·20	...	0·10	...	112·89	1400·00	
...	...	2·25	0·95	...	113·84	1460·00	7 ft. from outside rail.
18·30	...	4·46	14·05	0·21	113·84	...	
4·46	...	...	0·21	...	100·00	...	
13·84	...	...	13·84	...	13·84	...	

This method of booking levels gives more accurate results than any system. It may, however, be more compressed, as the work can be tested when the levels are reduced ready for plotting. If the sum of the back and foresights are added up, and the sum of the rises and falls also added up, then the difference between the sum of the back sight and foresight ought to be the same as the difference between the total of the rises and falls. The difference between the last reduced level and the original datum should also coincide with these two results, as shown in foregoing example.

Another method of booking the levels may here be shown, in which only four or five columns are used. This system of booking is employed when rapid results are required on the ground, and can be done very quickly when practised regularly, but is not a method to be recommended for students in general, on account of liability to error.

Instrument Height.	Sights.	Reduced Levels. Datum.	Remarks and Distances
62.75	12.75 B	50.00	On floor of machine house.
	8.10	54.65	On level of discharging platform.
	11.25	51.50	At peg. 0
	8.70	54.05	73
73.61	{ 1.35 F	61.40	100
	{ 12.21 B	61.40	100
	6.80	66.81	120
	4.50	69.11	200
	4.40	69.21	272
	3.45	70.16	303½
79.79	{ 1.87 F	71.74	Fence 308½
	{ 8.05 B	71.74	308½
	6.10	73.69	400
	5.70	74.09	500
	6.70	73.09	600
	8.50	71.29	700
	11.65	68.14	800
	12.00	67.79	820
70.49	{ 13.10 F	66.69	Loading bank.
	{ 3.80 B	66.69	„ „

In this method of levelling a datum is selected, as in the above at 50, the instrument is then directed to the staff and a back sight taken of 12.75, this is added to the datum, and will of course give the height of the instrument above the datum line.

Other intermediate sights are taken and recorded in the same column. These are deducted from or added to the height of the instrument, as the case may be, until a foresight marked F is reached; the instrument being now moved, another back-sight reading taken, and a new height obtained, and the reduced levels worked out as before.

Plotting the Levels.—When the levels are all reduced and ready for plotting, a horizontal base line A B, fig. 742, is drawn on the paper and the distances ab , bc , cd , de , ef , etc., laid off to scale. At distance O, the first vertical line aa is drawn at right angles to A B, the height of this line being the selected datum line say 50 as an example. At b , or 73 from a (from example), the first reduced level (bb) is set up, and the distance 54.05 marked off. In the same way all the other reduced levels are set up until the section is completed. When plotting sections like this, it is usual to use two different scales for the horizontal and vertical distances, the vertical lines being drawn to scale from 3 to 6 times greater than the horizontal distances. By doing so, the depths of cutting and amount of embankment are shown with greater clearness than if both were drawn to the one scale, and requirements can be ascertained with much greater accuracy.

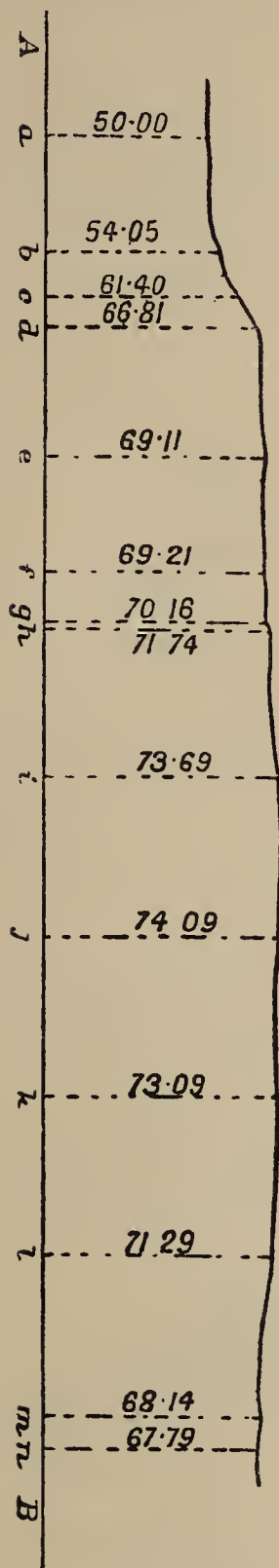


FIG. 742.

Surveying Problems.—The following worked examples are given, which may prove useful to the reader, more especially if he be studying for one of the mining examinations.

Question.—Find the area of a parallelogram whose sides are 7 and 15 respectively, and the included angle is 30° (fig. 743).

$$\text{Area} = BC \times DC \times \text{Sine } 30^\circ.$$

$$\text{Sine of } 30^\circ = 0.5000.$$

$$\therefore \text{Area} = 15 \times 7 \times 5 = 52.5 \text{ Ans.}$$

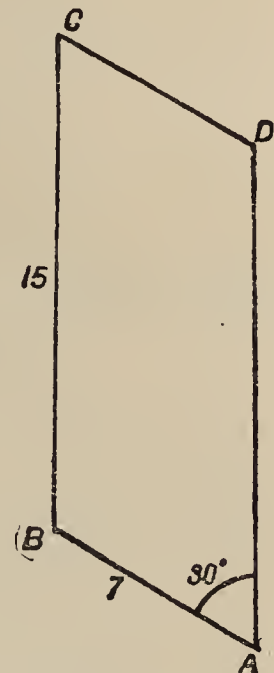


FIG. 743.

Question.—Find the ratio of the sides of a square to its diagonal. Find the length of side of square if the diagonal is 10 (fig. 744).

(1) The ratio of the side of a square to its diagonal is as $1 : \sqrt{2}$ or $\text{Ratio} = \frac{\text{side of square}}{\text{diagonal}} = \frac{1}{\sqrt{2}}.$

$$\therefore \text{Diagonal} = \sqrt{2} \text{ side.}$$

(2) If the diagonal of a square is 10, the side of square $= \frac{10}{\sqrt{2}} = 7.07 \text{ Ans.}$

Question.—A, B, and C denote the lengths of the sides of a triangle; how may its area be determined in terms of its sides?

$$\text{Let } S = \frac{A+B+C}{2} \text{ then area} = \sqrt{S(S-A)(S-B)(S-C)}.$$

Question—The sides (A B, B C, C D, and D A) of a field measure 28, 45, 60, and 57 yards in length respectively. The angle ABC is a right angle. Find the area of the field in square yards (fig. 745).

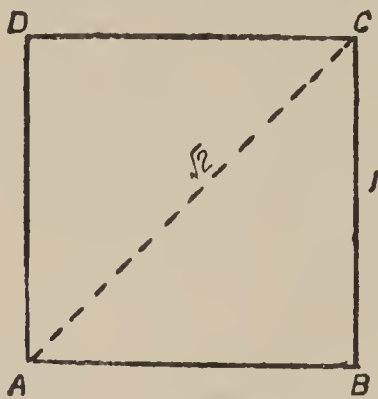


FIG. 744.

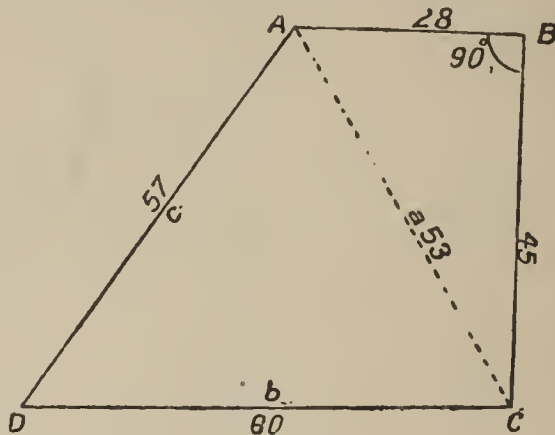


FIG. 745.

$$\text{The area of the triangle } ABC = \frac{28 \times 45}{2} = 630.$$

$$\text{The length of } AC = \sqrt{AB^2 + BC^2} = 53.$$

$$S = \frac{53 + 60 + 57}{2} = 85.$$

and the area of the triangle ADC =

$$\sqrt{S(S-a)(S-b)(S-c)} = \sqrt{85(32 \times 25 \times 28)} = 1379.8$$

The total area of the field, therefore, equals $1379.8 + 630 = 2009.8$ sq. yds.

Question.—The shape of a coalfield is rectangular, the sides 1 mile and $\frac{1}{2}$ mile respectively. It contains three workable seams of coal of the following thickness and specific gravity.

1st seam, 3 ft. 0 in. thick. Specific gravity, 1.28.

2nd seam, 2 ft. 9 in. thick. Specific gravity, 1.29.

3rd seam, 2 ft. 6 in. thick. Specific gravity, 1.30.

Required the average available amount of coal in the field, after making a fair deduction for loss in working, there being no special pillars to be left in.

$$1 \text{ mile} = 80 \text{ chains. } \therefore \text{area of field} = 80 \times 40 = 3200 \text{ sq. chains}$$

$$1 \text{ acre} = 10 \text{ sq. chains. } \therefore \text{area of field in acres} = \frac{3200}{10} = 320.$$

The total tons of mineral in a field = area $\times$ thickness of seam in in. $\times$ specific gravity $\times 100$.

$$\therefore \text{total quantity in 1st seam} = 320 \times 36 \times 1.28 \times 100 = 1474560 \text{ tons}$$

$$\text{,, ,, 2nd ,,} = 320 \times 33 \times 1.29 \times 100 = 1362240 \text{ ,,}$$

$$\text{,, ,, 3rd ,,} = 320 \times 30 \times 1.30 \times 100 = 1248000 \text{ ,,}$$

$$\text{total quantity in three seams} = 4084800 \text{ tons.}$$

Allowing 15 per cent. for faults, bad coal, and loss in working, the total available amount will be $\frac{4084800 \times 85}{100}$ or 3472080 tons.

Question.—A seam dips 1 in 3; find the length of a level mine to cut a fault of 10 fathoms which intersects the seam (fig. 746).

An approximate solution, assuming the fault to be at right angles to the seam.

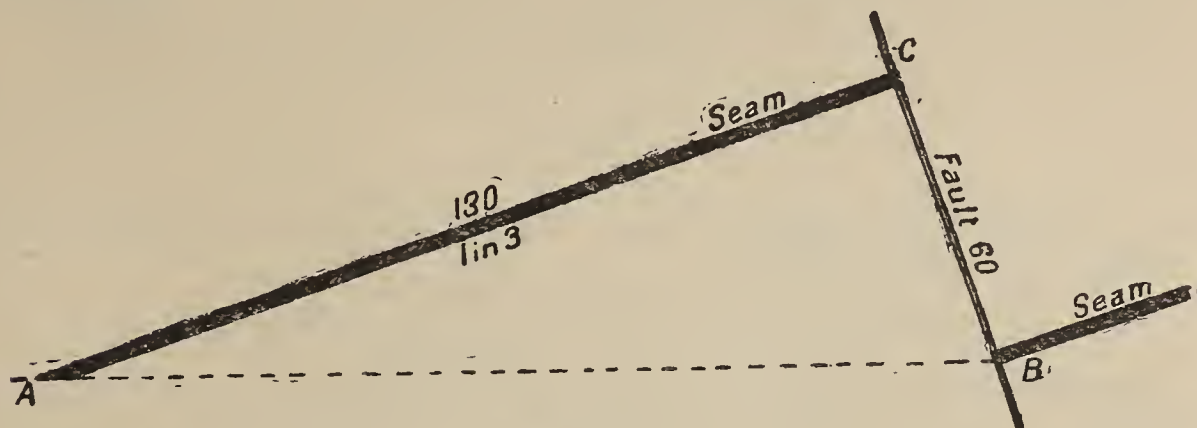


FIG. 746.

If the seam rises 1 in 3, what length of slope will give a rise of 60 ft.?

$$1 : 60 :: 3 = \frac{60 \times 3}{1} = 180 \text{ ft. } \therefore AC = 180 \text{ ft.}$$

$$\therefore AB^2 = AC^2 \times BC^2 = \sqrt{36000} = 189.7 \text{ ft. Ans.}$$

If in the above problem the mine rises 1 in 70, find what the length would be (fig. 747).

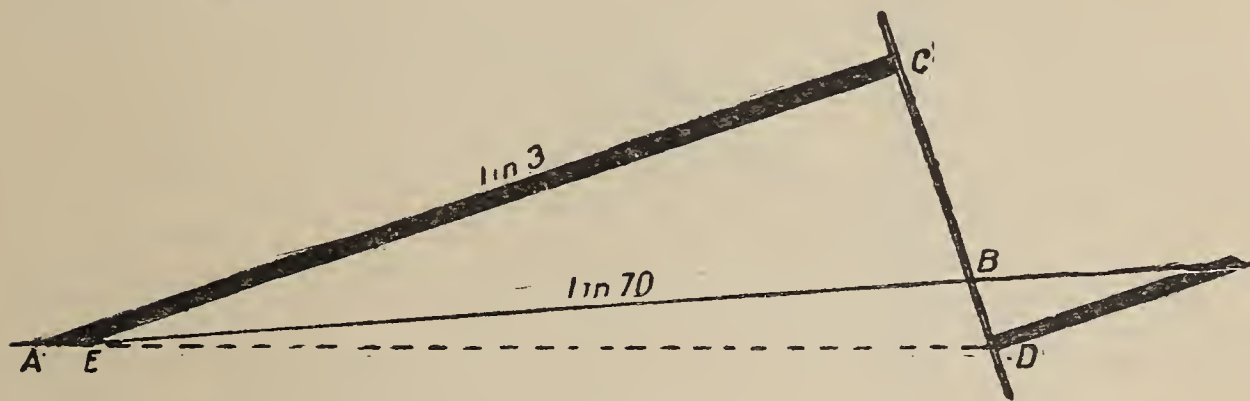


FIG. 747.

Let AD be an imaginary level line, and suppose we go along AD (A to E) 3 ft., then the line AC has risen 1 ft. above AD at E. The line AB has also risen $\frac{3}{70}$ ft. above AD. $\therefore$ in 3 ft. AB has parted from AC $1 - \frac{3}{70} = \frac{67}{70}$ ft.

If we gain $\frac{67}{70}$ ft. in 3 ft. how far must we go to gain 60 ft.?

$$\frac{60 \times 3}{\frac{67}{70}} = 188.05 \text{ ft. } AC = 188.05$$

$$\therefore AB^2 = (188.05)^2 \times (60)^2 = \sqrt{38737.80} = 196.8 \text{ ft. Ans.}$$

Question.—A seam dips 1 in 7 towards the south. Level course is N. 70° E.; find the true bearing of a road rising 1 in 60.

Assume A C and A B to be of equal length (fig. 748). A C is level, and A B rises 1 ft. in 60 ft., therefore the point B is 1 ft. higher than

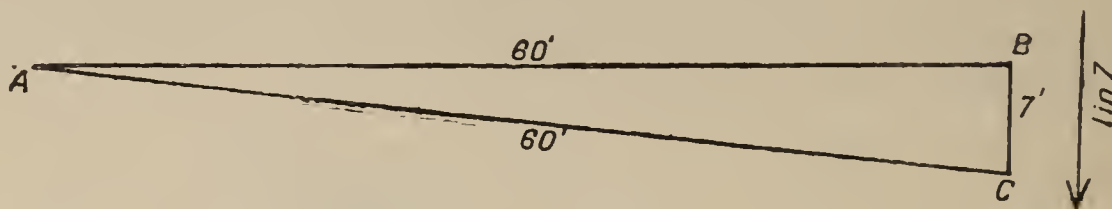


FIG. 748.

C. Now the seam is rising 1 in 7, and if it rises 1 ft. from C to B, distance C B must be 7 ft.

$$\text{By trigonometry } \sin \angle C A B = \frac{BC}{AB} = \frac{7}{60} = 0.1166.$$

From tables $\sin 0.1166 = 6^{\circ} 42'$ and the bearing of road $= 70^{\circ} - 6^{\circ} 42' = \text{N. } 63^{\circ} 18'$ E. Ans.

Or let the diameter of the circle be 120 ft., then circumference $= 120 \times 3.1416 = 377$ ft. nearly.

$$\text{and } 377 : 7 :: 360^{\circ} = 6^{\circ} 41'.$$

To find bearing

$$\begin{aligned} A B &= \text{N. } 70^{\circ} \text{ E.} \\ \therefore A B &= \text{N. } 70^{\circ} - 6^{\circ} 41' \text{ E.} \\ &= \text{N. } 63^{\circ} 19' \text{ E.} \end{aligned}$$

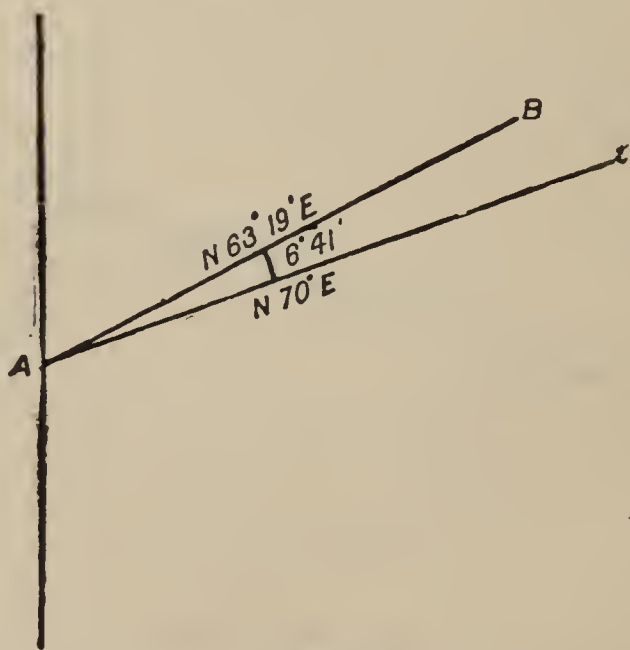


FIG. 749.

Question.—A seam is dipping 1 in 3 due south, a road is driven N. 45° W. Find the inclination of road. A road driven N. 90° W. will be level.

$$\sin 45^{\circ} = 1.4142136$$

If A C = 100, B C also = 100 and

$$AB = \sqrt{100^2 \times 100^2} = \sqrt{20000} = 141.42136$$

The seam is rising 1 in 3. $\therefore$ B is $\frac{100}{3} = 33.3$ ft. above C and also 33.3 ft. above

A; so that AB rises 33.3 ft. in 141.4 ft. and inclination = $\frac{141.42}{33.3} = 1$ in 4.24.

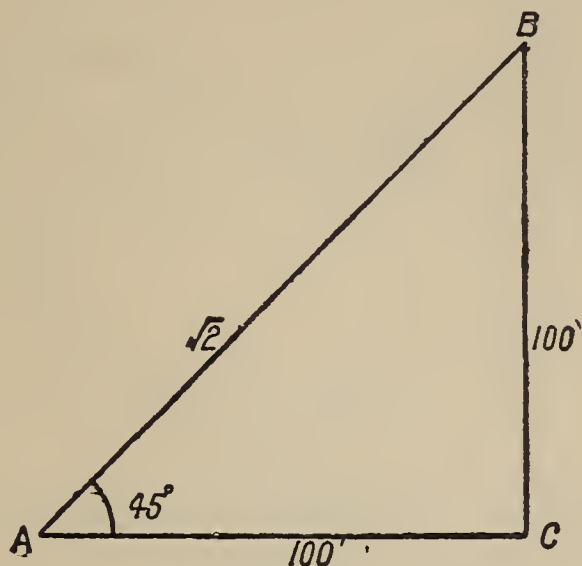


FIG. 750.

Question.—A seam is dipping 1 in 5 due south. A road is driven N. 60° W. Find its inclination.

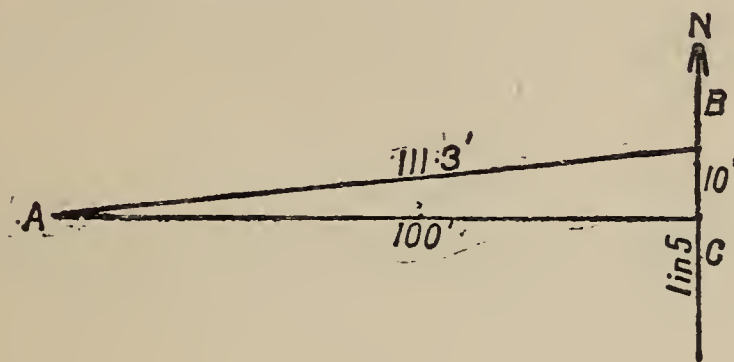


FIG. 751.

If $AC = 100$, $BC = 50$. $\therefore AB = \sqrt{100^2 \times 50^2} = 111.3$, if $BC = 50$ and is dipping 1 in 5 then B is $\frac{50}{5} = 10$ ft. higher than C or than A, so that AB rises 10 ft. in 111.3 and $\therefore$ inclination = $\frac{111.3}{10} = 1$ in 11.3 Ans.

Question.—A road has been driven 1° off the bearing. Find how much it is out in 500 ft.

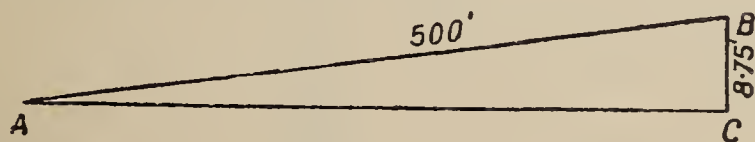


FIG. 752.

By trigonometry $BC = AB \times \sin BAC = 500 \times .0175 = 8.75$ ft. Ans.

An approximate method of working a question of this kind, and

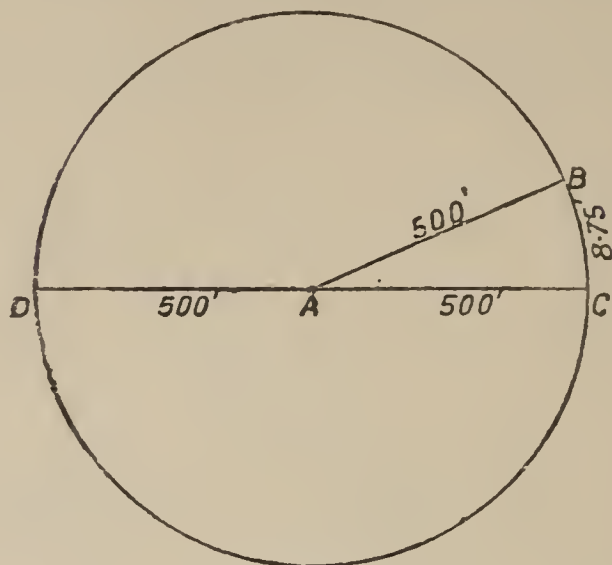


FIG. 753.

which may be useful to students who have no acquaintance with trigonometry, may be explained as follows:—

Let BCD be a circle with a radius of 500 ft. (fig. 521).

∴ The circumference of this circle = $2 \times 500 \times 3.1416$
= 3141.6 ft.

Suppose the arm AB is swung round a complete circle it will have described an angle of 360° , and the point B will have travelled 3141.6 ft. Now, by proportion, if B travels 3141.6 ft. for 360° , how many ft. will it travel for 1° ?

∴ $360^\circ : 1^\circ :: 3141.6 \text{ ft.}$

or $\frac{1 \times 3141.6}{360} = 8.72 \text{ ft. Ans.}$

This distance, of course, is measured along the arc of the circle, but in a case like this the difference between the length of the arc and that of its chord is infinitesimal, and would be negligible for all practical purposes.

Question.—If a road has been driven for 200 yards and A is then found to be 30 yds. off the true course, which must have been the error in bearing in setting out?

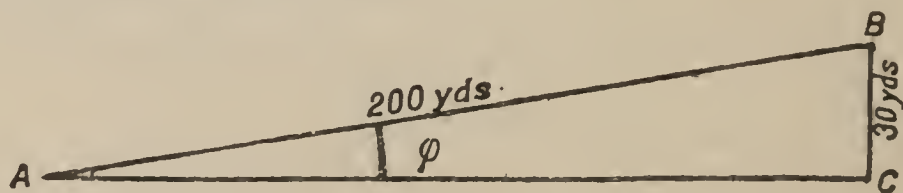


FIG. 754.

By trigonometry $\sin \theta = \frac{BC}{AB}$

$$= \frac{30}{200}$$

∴ $\sin \theta = .1500$

∴ $\theta = 8^\circ 38' \text{ Ans.}$

or by the approximate method mentioned above we have a circle whose radius is 200 yds., $\therefore$ circumference $= 2 \times 200 \times 3.1416$
 $= 1256.64$ yds.

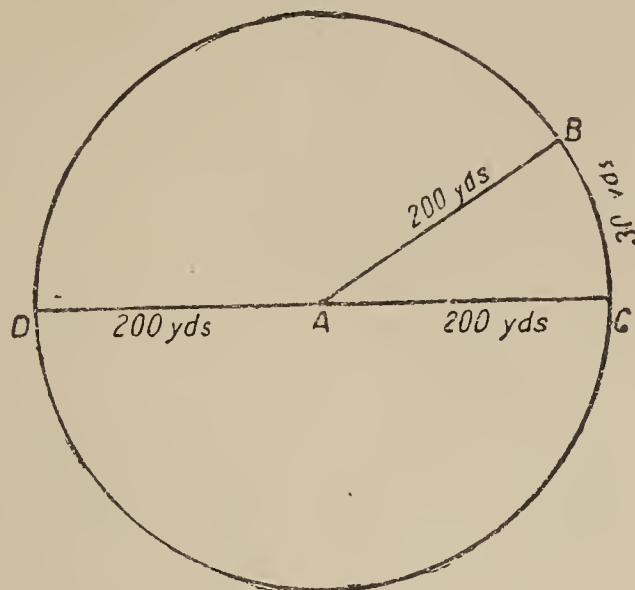


FIG. 755.

But the circumference is also $= 360^\circ$.

$$\therefore 1256.64 \text{ yds.} : 30 \text{ yds.} :: 360^\circ$$

$$\therefore \frac{30 \times 360}{1256.64} = 8^\circ 35' \text{ Ans.}$$

Note.—In addition to the ordinary colliery plans, the Coal Mines Act 1911 enacts that a separate plan showing the system of ventilation, the intake coloured blue and the return red, must be made and kept at the mine. A plan must also be kept showing the position of the electrical apparatus underground.

INDEX.

- ABSOLUTE temperature, 516.
- Accidents, report on, 254.
to boilers, 703.
- Accommodation, siding, 699.
- Action of bucket pump, 464.
of controller, 361.
of plunger pump, 465.
- Adelaide drill, 127.
- Adjusting screws, 346.
the level, 755, 756.
- Advantages of coal-cutters, 171, 172.
of compressed air, 188.
of electricity, 202.
of fans, 586.
of longwall, 217.
- Air, 519.
capacity of output of, 584, 585.
composition of, 519, 520.
compressed, 178-188.
crossings, 593.
moisture in, 520, 521.
motors, 186, 187.
pipes, 598.
speed of upcast, 562.
- Air-current, guiding the, 592-599.
laws affecting the, 599-603.
reversing the, 576, 577, 578.
splitting the, 605-608.
- Air-vessels, 505, 506.
- Allardice conveyor, 437-439.
- Alternating current, 197.
- Analysis, gas, 547-552.
method of, 549.
of coal, 9, 10, 11, 12, 13.
- Anderson-Boyes machine, 143, 144, 146.
- Anemometer, 612.
- Angus Smith's test for CO₂, 533.
- Anthracite, 9, 10, 13.
- Anticline, 3, 4.
- Apparatus, Ærophen, 683, 684, 685.
coal cleaning, 712, 713.
- Apparatus, Draeger, 676-679, 691.
Lunge's, 534, 535.
Meco, 672-676, 688.
oxygen reviving, 687-693.
Proto, 670, 671, 672.
Pulmotor, 691.
rescue, 666-698.
Weg, 679-683.
- Apparent dip, 3.
- Approved safety lamps, 664, 665.
- Aprons for rods, 461.
- Aqueous rocks, 1.
- Arching, brick, 287.
- Area of electrical conductors, 205, 206.
of suction pipes, 448, 449.
- Areas, calculation of, 751-754.
- Armoured cables, 193.
- Arrangement of haulage engines, 398, 402.
of main and tail rope, 394.
of pit-bottom, 375.
of pumps, 507, 508.
- Arrangements of surface, 699-730.
- Arrault's cutter, 22, 23.
- Ash, 289.
- Atmosphere, height of, 521.
- Atomic weight, 516.
- Attachments for ropes, 312.
spring, 321.
- Automatic trap doors, 594, 595, 596, 597.
- Auxiliary fans, 586, 587, 588.
- BALANCE brace, 383.
West and Darlington's, 464.
- Balancing pump rods, 463.
- Bang pieces, 461.
- Banking out, 706, 707, 708.
- Bar machines, 149.
- Barclay engine, 470.
fan, 571, 572.
- Barometer, 610.

- Barring, shaft, 59, 60.
 sizes of, 60.
 Baum washer, 729, 730.
 Béche, 18, 20.
 Beech, 289.
 Belts, travelling, 710, 711.
 Bench marks, 737.
 Bevel pulleys, 414, 415.
 Birch, 289.
 Bituminous coal, 9, 10.
 weight of, 13.
 Black-damp, 527-536.
 composition of, 527, 528.
 production of, 528, 529.
 properties of, 529.
 specific gravity of, 528.
 tests for, 529-536.
 Blakett conveyor, 429, 430.
 Blasting, cost of, 118, 119.
 gelatine, 104.
 Blind pits, 391, 392.
 Blocks, catch, 412, 413.
 Blood solution test for CO, 539.
 Blown-out shots, 117.
 Bobbinite, 103.
 Boghead coal, 12.
 Bogie clip, 412.
 Bohres electric lamp, 646, 647.
 Boiler chimneys, 706.
 Boilers, 701-706.
 accidents to, 703.
 combustion in, 705.
 Cornish, 701.
 corrosion of, 703.
 efficiency of, 704, 705.
 evaporative power of, 703, 704.
 Lancashire, 702.
 required, 705.
 strength of, 705.
 Booking the levels, 757.
 Bord and pillar, 218.
 Bore-holes, surveying, 37.
 uses of, 16, 17.
 Boring, 16.
 accidents in, 26.
 bits, 17, 18.
 cost of, 25.
 Japanese method of, 25.
 machine, hand-power, 32, 33, 36.
 Mather & Platt system of, 26, 27.
 methods of, 17.
 other objects of, 35.
 out shafts, 74.
 plant, 25.
 preliminary operations of, 19.
 rods, 17, 18.
 length of, 20.
 speed of, 24.
 Boring tools, 20, 21.
 Bortz diamonds, 30.
 Bothwell conveyor, 426-428.
 Bottle, sampling, 551.
 Bottom pillar, 209.
 Boxes, junction, 195, 196.
 Boyle's law, 517.
 Brake-staff, 19.
 Brakes for drums, 339-342.
 Branches, main and tail rope, 395.
 Brattice, 595, 596, 597.
 Breaking load of ropes, 314, 315.
 Breathing apparatus, 666-698.
 Ærophor, 683, 684, 685.
 Draeger, 672-679.
 Meco, 672-676.
 Proto, 670, 671.
 Weg, 679-683.
 Briart's guides, 324.
 wheel, 399.
 Brick arching, 287.
 drums, 68, 69.
 walls and girders, 282.
 Briggs' loop, 640.
 Brown coal, 13.
 Bucket pump, 464.
 action of, 464, 465.
 Buckets, pump, 453.
 Buildings, size of, 212.
 Bull engine, 469.
 Bulldog powder, 103.
 Buntons, shaft, 59, 60, 61.
 Burner for safety lamps, 656.
 Burnett wedge, 120.
 CABLES, armoured, 193.
 electric, 191-195.
 firing, 116.
 fixing, 194.
 shaft, 193, 194.
 three-core, 192.
 Cage conductors, 321.
 shoes, 329-332.
 Cages, 328, 329.
 changing tubs on, 334.
 gates for, 333, 334.
 keys for, 347.
 safety, 345.
 speed of, 332.
 Calcareous coal, 6.
 Calculation by instruments, 754.
 by ordinates, 753, 754.
 of areas, 751, 754.
 of triangles, 751, 752.
 Calculations for winding engines, 367-369.
 Calorie, 10.
 Calorific power, 10, 13, 15.

- Calorimeter, 10, 14.
- Cannel coal, 11.
- Capacity of output of air, 584, 585.
- Cappell fan, 566.
- Cappings for ropes, 312.
- Carbon, 519.
 - dioxide, 103, 527-536.
 - monoxide, 103, 536-540.
 - detection of, 537, 538, 539.
- Care of ropes, 310.
- Carriage incline, 390.
- Catch blocks, 412, 413.
- Ceag electric lamp, 641.
- Centrifugal pumps, 491.
- Chain attachments, 401.
 - endless, 398.
 - machines, 145.
 - shearing machine, 149.
 - stepping the, 745.
 - wheel, 399.
- Charge, firing the, 113.
- Charles' law, 516, 517.
- Chests, water, 440.
- Chimneys, boiler, 706.
- Chisels, boring, 17, 18.
- Chlorate mixtures, 103, 109, 110.
- Chocks, wooden, 258.
- Choice of coal-cutter, 162.
 - of washer, 717.
- Choke-damp, 527-536.
 - composition of, 527, 528.
- Circular ropes, 309.
 - shafts, 41-51.
 - cost of, 58, 59.
 - temporary lining of, 42, 43.
 - timbering of, 42, 43.
- Clacks, pump, 451.
- Clanny lamp, 627, 628.
- Clarke & Steavenson machine, 140.
- Classification of explosives, 101.
- Clearance, loss by, 186.
- Cleavage, 2.
- Clinostat, 37, 38.
- Clip, bogie, 412.
 - Fisher's, 409.
 - Rutherford, 410.
 - Scotch, 408.
 - Ward and Lloyd's, 411.
- Clips, haulage, 407-411.
- Coal, anthracite, 9.
 - bituminous, 10.
 - cannel, 11.
 - chemical analysis, 10, 11, 12, 13.
 - classification of, 8, 9, 10.
 - cleaning, 699-730.
 - conveyors, 424-439.
 - types of, 426.
 - definition, 8.
- Coal, gas, 9, 11.
 - mineralogical character of, 11.
 - origin of, 8.
 - screening the, 713.
 - search for, 16.
 - selection of, 13.
 - specific gravity of, 10.
 - steam, 10.
 - washing, 714-730.
 - winding, 295.
- Coal-cutters, advantages of, 171, 172.
 - conditions for, 163.
 - cost of installing, 166.
 - on skids, 167.
 - on wheels, 167.
 - power to drive, 167.
- Coal-cutting, 134.
- Coal-dust, 612-623.
 - danger of, 613.
 - dealing with, 612-623.
 - formation of, 617.
 - watering, 617-621.
- Coal-measures, 7.
 - division of, 7.
- Coefficient of expansion, 516.
- Coffering, 55-57.
 - thickness of, 58.
- Coke, 11, 12.
- Collection of gas samples, 550, 551.
- Colliery plans, 737.
- Colouring colliery plans, 739.
- Column, motive, 556, 567, 568.
- Combustion, rate of, 705.
 - spontaneous, 232-234.
- Commencing sinking, 42.
- Compass, vernier, 741, 742.
- Composition of air, 520.
- Compound levelling, 756, 757.
 - pumping engine, 470, 471.
 - wound dynamos, 198.
- Compressed air, 178-188.
 - advantages of, 188.
- Compressing fan, 575.
- Compressors, dry, 180, 181.
 - losses in, 185.
 - power required for, 184.
 - spray, 182.
 - stage, 182.
 - valve for, 183, 185.
 - wet, 182, 183.
- Concrete lining, 47.
- Conducting air-currents, 592-599.
- Conductors, capacity of, 208.
 - electrical, 205.
 - loss in, 207.
 - resistance of, 207.
- Conical drum, 317.
 - drums, 338.

- Construction of safety lamps, 653.
 Contiguous seams, 227-232.
 Continental timbering, 267.
 Contour lines, 737.
 Controller, action of, 361.
 Caledonia, 359, 360.
 Cornish boilers, 701.
 engine, 469.
 Corrosion of boilers, 703.
 Cost of blasting, 118, 119.
 of boring, 25, 30, 31.
 plant, 25, 26, 33.
 of coal washing, 730.
 of haulage, 418.
 of horse haulage, 382.
 of lining, 65.
 of machine cutting, 170, 171.
 of power transmission, 202.
 of preserving timber, 293.
 of pumping, 509, 510.
 of timber, 275.
 of upkeep of safety lamps, 656, 657.
 Counterbalancing cages, 316, 317.
 Coupled engines, 303.
 Coupling, spring, 321.
 Courrières system, 260.
 Cradle, walling, 46.
 Creep, 221.
 Crossings, air, 593.
 Crow's-foot, 21, 23.
 Crushing strain of timber, 289.
 Cunynhame-Cadman device, 639.
 Curb, wedging, 56.
 Curbs, wooden, 42, 43.
 Current, continuous, 197.
 Currents, induction, 734.
 Curves, working of, 400, 401, 414, 415.
 Cut, depth of, 167.
 Cut-chain incline, 386.
 Cutter, Arrault's, 22, 23.
 Hardy Puncher, 160.
 Oeynhausen's, 22, 23.
 Cutting by percussive machines, 168.
 commencing, 169.
 method of, 169, 170.
 Cylinder, sinking, 69, 70, 71.

 DALMELLINGTON, working at, 231.
 Datum line, 757.
 Davey's differential engine, 478, 479.
 Davis calyx drill, 33.
 operation of, 34.
 Davy lamp, 627.
 Decking cages, 334, 335.
 Declination of needle, 734.
 Deepening shafts, 96, 97.
 Definition of explosive, 101.
 Deflector lamp, 634.

 Departure, 748, 749.
 Depth of cut, 167.
 Detaching hooks, 342-345.
 Detection of CH_4 , 545, 546.
 of CO , 537, 538, 539.
 Determination of true meridian, 736.
 Detonators, 111.
 charge of, 112.
 fixing of, 113.
 Deviation of bore-holes, 37, 38.
 Diamond boring, 28, 29.
 cost of, 31, 32.
 speed of, 30.
 machine, 142.
 system, advantages of, 33.
 Diamonds, bortz, 30.
 Dip, apparent, 3.
 direction of, 2.
 of needle, 735.
 true, 3.
 Direct-acting pumping engines, 468,
 472, 473.
 Direct-rope haulage, 392.
 Direction of pillars, 222.
 Disadvantages of electricity, 202.
 of longwall, 218.
 Disc machines, 137-145.
 Dislocations, 4.
 Diurnal variation, 733, 734.
 Division of rocks, 6.
 Doors, trap, 593, 594, 595.
 Double stall working, 247.
 Downthrow, 4, 16, 17.
 Draeger apparatus, 676-679.
 Drawing roads, main, 265.
 timbering, 263.
 Drift, ventilating close, 596.
 Drill, Adelaide, 127.
 calyx, 33.
 Conqueror, 123, 124.
 Darlington, 126.
 Flottmann, 130, 131, 132.
 hammer, 128.
 Hardy-Simplex, 133.
 Ingersoll, 128.
 Siskol, 129, 130.
 Drills, hand, 123, 124.
 machine-power, 124-133.
 rock, 123.
 Driving pulleys, 403.
 through loose ground, 269.
 Drum, conical and spiral, 317, 318.
 Drums, brakes for, 339-342.
 brick, 68, 69.
 conical, 338-342.
 winding, 337, 338.
 Dry compressors, 180.
 Dumb drift, 561, 562.

- Durability of ropes, 310.
- Dustless zones, 621.
- Duty of pumping engines, 506, 507.
- Dykes, 4, 5, 16.
- Dynamite, 105.
 - efficiency of, 105.
- Dynamos, 197.
 - horse-power of, 208.
- Dysart thick seam, 233.

- EADIE's joint, 447.
- Earth, the, 1.
 - thickness of, 1.
- Efficiency, manometrical, 581, 582.
 - of boilers, 704, 705.
 - of electric haulage, 199, 200.
 - pumping, 201.
 - of fan, 582, 583.
- Electric cables, 191-195.
 - fuses, 114, 115.
 - haulage, 419.
 - lamps, 640-652.
 - power machines, 139-145.
 - sinking pump, 505.
 - winders, 362-366.
 - advantages of, 366.
 - types of, 363.
- Electrical firing of shots, 114.
 - plant, efficiency of, 199.
 - failures, 198, 199.
 - terms, 202, 203.
- Electricity, 190-208.
 - advantages of, 202.
- Electrostatic stress, 527.
- Elevation of shafts, 52, 60, 61.
- Elevators, 712.
- Elliot wedge, 120.
- Elm, 289.
- Endless chain haulage, 398.
 - wheels for, 399.
 - rope, 401-417.
 - advantages of, 419.
 - curves, 414, 415.
 - on inclined roads, 415-418.
- Engine, Davey's differential, 478, 479.
- Engines, duty of pumping, 506, 507.
 - position of, 306.
 - pumping, 468-479.
 - seats for, 307.
 - sizes of winding, 366-369.
 - speed of, 304.
 - winding, 301-305.
- Enlarging shafts, 93-96.
- Equal splits, 606.
- Equipment of rescue corps, 693-698.
- Equivalent orifice, 578, 579, 580.
- Evan Evan's lamp, 632.
 - Thomas' lamp, 633.
- Evan's sinking pump, 504.
- Evaporative power, 10.
 - of boilers, 703, 704.
- Exhaust steam, 177-178.
- Expansion, coefficient of, 516.
 - joint, 447.
- Exploders, 116.
- Explosions, precautions against, 113.
- Explosives, 101-111.
 - composition of, 106-109.
 - Order, 105.
 - permitted, 105.
- Extracting pillars, 222, 225.

- FACES, supporting working, 255.
- Fan, advantages of, 586.
 - auxiliary, 586, 587.
 - Barclay, 571, 572.
 - Cappell, 566.
 - compressing, 575.
 - cost of, 586.
 - efficiency of, 582, 583, 584.
 - Guibal, 563, 564.
 - method of driving, 588.
 - Rateau, 573.
 - Schiele, 565.
 - selection of a, 574.
 - Ser, 572.
 - Sirocco, 569, 570, 571.
 - speed of, 585, 586.
 - tests of, 588-592.
 - Waddle, 564.
 - Walker, 567, 568.
- Fans, 563-592.
- Fault, downthrow, 4.
 - dumb, 5.
 - normal, 4.
 - overlap, 5.
 - reversed, 5.
 - step, 5.
 - throw of, 4.
 - trough, 5.
 - upthrow, 4.
- Faults, 4, 5, 6.
- Feeding of horses, 379, 380.
- Fences for cages, 333.
- Filling safety lamps, 663.
- Fir, 289.
- Fire-damp, 540-547.
 - accumulations of, 552.
 - blowers of, 540, 541.
 - explosions of, 544, 545.
 - ignition of, 542, 543, 544.
 - indicators, 636-640.
 - means of detecting, 545, 546, 547.
- Firing cables, 116.
 - shots, authority for, 115.
 - electrically, 115.

- Firing, simultaneous, 116, 117.
 the charge, 113.
 Fisher's clip, 409.
 Fixed inclined screens, 713.
 needle compass, 741, 742.
 Fixing cables, 194.
 Flat ropes, 308.
 Flottmann drill, 130, 131.
 Folding doors, Galloway, 63, 64.
 Form of lamp burner, 656.
 Formation of coal-dust, 616.
 Formulæ for ropes, 314, 315.
 for tubbing, 54, 55.
 for turbine pumps, 500, 501.
 Fors electric lamp, 642, 643.
 Foundations for engines, 307.
 Frames, height of, 300.
 pithead, 295-302.
 steel, 300.
 Freezing system, cost of, 89.
 of sinking, 84.
 Friction, in pipes, 187.
 loss by, 187.
 of air, reducing, 603.
 Furnace ventilation, 556-563.
 cost of, 560.
 Fuses, electric, 114, 115.
 safety, 113, 114.
- GALLOWAY folding doors, 63, 64.
 scaffold, 48, 49.
 Garland, water, 50.
 Gas analysis, 547-552.
 apparatus, 548, 549.
 collection of samples of, 550, 551.
 marsh, 540-547.
 Gases, atomic weight of, 516.
 compound, 518, 527-547.
 diffusion of, 518.
 elementary, 518, 519.
 given off by gunpowder, 102.
 in mines, 514-547.
 molecular weight of, 516.
 properties of, 516.
 specific gravity of, 515.
 symbols of, 516.
 Gates for cages, 333.
 Gauge, water, 611.
 Gear, haulage, 150.
 Whitmore, 356-360.
 Geared pumping engines, 468, 473.
 Gelatine, blasting, 104.
 Generators, electrical, 197.
 Geometrical levelling, 755.
 Gibb conveyor, 430, 431, 432.
 Gillot & Copley machine, 138.
 Girders, sizes of, 286.
 supporting roads by, 277-284.
- Gradient of horse roads, 378, 379.
Gradients Renversés, 244.
 Graham's law, 518.
 Grapnel tube, 23.
 Greaves washer, 720.
 Ground, driving through loose, 269.
 Guibal fan, 563, 564.
 Guides, Briart, 324.
 cage, 321-326.
 rope, 326, 327.
 steel, 322.
 Guiding pump rods, 460.
 the air-current, 592-599.
 Gunpowder, 102, 103.
 efficiency of, 105.
- HAILWOOD electric lamp, 651, 652.
 Haldane gas analysis apparatus, 548, 549.
 tube, 530-533.
 Hammer drills, 128.
 Hand drills, 123, 124.
 Hand-power boring machine, 32, 33.
 Hardy Puncher machine, 159, 160.
 Hardy-Simplex drill, 133.
 Harrison machine, 157.
 Haulage, advantages of endless rope, 419.
 clips, 407-411.
 cost of, 418.
 direct rope, 392.
 electric, 419, 420.
 endless rope, 401-417.
 gear for coal-cutters, 150.
 main and tail rope, 393-401.
 manual, 376.
 methods of, 370.
 problems, 421-424.
 roads for, 370, 371.
 timbering of, 271.
 stationary engine, 392.
 tests of electric, 199, 200.
 underground, 370-439.
 working of, 395.
 Heading machines, 135-137.
 Heenan and Froude's tippler, 710, 711.
 Height of atmosphere, 521.
 of pithead frames, 300.
 Helmets, smoke, 686, 687.
 Hepplewhite-Gray lamp, 630.
 Highly inclined roads, supporting, 273.
 seams, 234-245.
 Hooks, detaching, 342-345.
 Hopkinson machines, 147.
 Hoppit, sinking, 89, 90, 91.
 Horizontal pumping engines, 472, 473.
 Horse haulage, cost of, 382.
 roads, 377-379.

Horse traction, 377.
 Horses, feeding of, 379, 380.
 stables for, 381.
 Hurdle screen, 598.
 Hydraulic cartridge, 121.
 keps, 349.
 pump, Moore's, 486, 487.
 Hydrogen, 102, 519.
 carburetted, 540-547.
 sulphuretted, 539, 540.
 Hygrodeik, 526.
 Hygrometer, 521, 522.

ILGNER winder, 363.
 Inclination of horse roads, 378, 379.
 of sidings, 699, 700.
 Inclined roads, supporting, 273.
 screens, fixed, 713.
 Inclines, blocks for, 387.
 carriage, 390, 391.
 cut-chain, 386.
 endless rope, 389.
 self-acting, 383-390.
 Indicators, fire-damp, 636-640.
 Induction currents, 734.
 Influence of rails on needle, 735.
 Ingersoll drill, 128.
 Ingersoll-Sergeant machine, 155, 156.
 Injection compressors, 182.
 Instructions for rescue corps, 693-698.
 Instruments, calculation by, 754.
 used at collieries, 609.
 Iron riders, 91, 92, 93.
 ropes, 308.

JEFFREY chain machine, 145, 146.
 machine, 144.

Jig brow, 383.
 Jiggers, haulage, 407-411.
 Jigging screens, 714.
 Junction boxes, 195, 196.

KASELOWSKY pump, 487, 488.
 Keps, cage, 347.
 hydraulic, 349.
 Stauss, 347.
 Kibble, 62, 63, 89, 90, 91.
 wooden, 91.
 Kilsyth, working seam at, 240.
 Kiltongue seam, 229.
 Kind-Chaudron method, 75, 76, 77,
 79.
 advantage of, 81.
 Kind's plug, 24.
 King's hook, 344.
 Knock-off's for haulage, 395.
 Koch system of sinking, 86, 87, 88.
 Koepé's system of winding, 319.

LABOUR, disposition of, 65.
 for machine cutting, 164.
 Lagging deals, 42, 43.
 Lamp, Bohres, 646.
 Ceag, 641, 642.
 Chesneau, 639.
 Clanny, 627, 628.
 Clowes, 637.
 Davy, 627.
 deflector, 634.
 Evan Evan's, 632.
 Evan Thomas', 633.
 Fors, 642, 643, 644.
 Gray-Sussman, 640, 641.
 Hailwood's, 651, 652.
 Hepplewhite-Gray, 630, 631.
 Marsaut, 628, 629.
 Mueseler, 629, 630.
 Paterson, 635, 636.
 Pieler, 638.
 Stach, 644, 645, 646.
 Stephenson, 628.
 Stokes', 637.
 Wolf, 631.

 alkaline, 647, 648, 649, 650.
 Wolf-Dahlmann, 632.
 Lamps, cleaning safety, 662.
 construction of safety, 653.
 cost of upkeep of, 656, 657.
 filling safety, 663.
 form of burner for, 656.
 lighting power of, 653, 654, 655.
 power of safety, 653-655.
 list of approved safety, 664, 665.
 locking of, 657.
 safety, 626-663.
 testing safety, 659-662.

Lancashire boilers, 702.

Larch, 289.

Latitude, 748, 749.

Laws affecting air-currents, 599-603.

Laying out face for machines, 168.
 roads, 371.

Lazy-tongs, 77.

Leakage, loss by, 187.

Length of lifts, 466.

 of stroke, 467.

 of walls, 211.

Lens Colliery, timbering at, 260.

Level, adjusting the, 755, 756.

Levelling, 754-759.

 a section, 756.

 compound, 756, 757.

 geometrical, 755.

 physical, 754.

 trigonometrical, 754.

Levels, booking the, 757, 758.

 plotting the, 759.

- Life of ropes, 310.
 Lignite, 9, 13.
 weight of, 13.
 Line, datum, 757.
 Lines, contour, 737.
 Lining, concrete, 47.
 shafts, cost of, 65.
 temporary, 43, 44.
 tubes, 23, 24.
 removing, 24.
 Lippman method of sinking, 80.
 Longwall, 209-218.
 advantages of, 217.
 disadvantages of, 218.
 laying out, 210.
 Loose ground, driving through, 269.
 Loss by clearance, 186.
 by friction, 187.
 in power transmission, 176, 177, 185.
 Lührig washer, 725-729.
 Lunge's apparatus, 534, 535.
- MACGEORGE clinostat, 37, 38.
 Machine cutting, cost of, 170.
 labour for, 164.
 face, timbering at, 164.
 motive power for, 165.
 Machinemen, daily report for, 179.
 Machine-power drills, 124-133.
 Machines, Anderson Boyes, 143, 146.
 Bar, 149-154.
 chain, 145.
 choice of, 162.
 Clarke & Steavenson, 140.
 classification of, 135.
 coal-cutting, 134-174.
 conditions suitable for, 163.
 diamond, 142.
 disc, 137-145.
 electric power, 139.
 Gillot & Copley, 138.
 Hardy, 159, 160.
 Harrison, 157.
 heading, 135, 136, 137.
 Hopkinson, 147.
 Ingersoll-Sergeant, 155, 156.
 Jeffrey, 144.
 laying out face for, 168, 169.
 Meco, 161, 162.
 method of working, 155.
 Morgan-Gardner, 157.
 number of, 135, 136.
 percussive, 152-162.
 Pick-Quick, 150.
 Rigg & Meiklejohn, 138, 139.
 Siskol, 157, 158.
 work done by, 168.
- Magnetic ridge lines, 732, 733.
 storms, 735.
 Main and tail rope haulage, 393-401.
 roads, timbering of, 265.
 Maintenance of rescue corps, 693-698.
 Management of ropes, 310.
 Manometrical efficiency, 581, 582.
 Marks, bench, 737.
 Marsaut lamp, 628, 629.
 Marsh gas, 102.
 Mason's hygrometer, 521.
 Mather & Platt's system of boring, 26, 27.
 Measurements, underground, 743, 744.
 Measuring on the slope, 744, 745.
 Mechanical efficiency of fan, 583, 584.
 ventilation, 563-592.
 wedges, 120.
 Meco apparatus, 672-676, 688.
 Men employed in sinking, 66.
 Meridian, determination of true, 736.
 line, setting out, 736, 737.
 magnetic, 732, 733.
 true, 732, 733.
 Methane, 540-547.
 Method of cutting, 169.
 of driving fans, 588.
 Methods, choice of, 209.
 of gas analysis, 548, 549.
 of sinking, special, 66.
 Triger's, 73.
 Mickley conveyor, 426.
 Mines, gases present in, 514-547.
 Modes of working, 209-252.
 Moisture in air, 520, 521.
 Molecular weight, 516.
 Moore's hydraulic pump, 486, 487.
 Moss-box, 78, 80.
 Motive column, 556, 557, 558.
 Motors, air, 186, 187.
 electrical, 198.
 horse-power of, 208.
 Mueseler lamp, 629, 630.
 Murton coal washer, 720.
- NATURAL ventilation, 554, 555.
 Necessity for timbering, 253.
 Needle, declination of the, 734.
 dip of, 735.
 influence of rails on, 735.
 Newbattle Colliery, 48.
 Niddrie, workings at, 235-239.
 Nitrate mixtures, 103, 105.
 Nitro-compounds, 103, 104, 105.
 Nitrogen, 102, 519.
 Number of boilers required, 705.
 of men for sinking, 66.

- OAK, 289.
 Objects of boring, 16, 35.
 Observations of pole star, 736.
 Oddie-Barclay pump, 483-486.
 Oeynhausens's joint, 22.
 Offsets for pump-rods, 461, 462.
 Order of working seams, 251.
 Ordinates, calculation by, 753, 754.
 Orifice, equivalent, 578, 579, 580.
 Output of air, 584, 585.
 Overwinding, prevention of, 350.
 Oxide, carbonic, 536.
 Oxygen, 519.
 reviving apparatus, 687-693.

 PACK, water, 226.
 Packs, size of, 212.
 Panel system, 225, 226.
 Paterson lamp, 635, 636.
 Pattberg system, 81.
 Peat, weight of, 13.
 Percussive boring, 17.
 machines, 152-162.
 cutting by, 168.
 method of working, 155.
 Physical levelling, 754.
 Pick-cutter, 154.
 Pick-Quick machines, 150.
 Pile-driving, 66, 67, 68.
 Pillar and stall, 218-225.
 size of bottom, 218, 219, 220.
 Pillars, direction of, 222.
 extracting, 222, 223, 224.
 size of, 220, 221.
 Pine, pitch and red, 289.
 Pipe supports, 447.
 Pipes, air, 598.
 size of, 448, 449.
 suction, 448.
 thickness of, 449, 450.
 weight of, 451.
 Piston, leakage at air, 187.
 pumps, 475.
 Pit-bottom, arrangement of, 375.
 Pithead frames, 295-302.
 pulleys, 335, 336.
 Pits, staple, 391.
 Plan of shaft, 52, 60, 61.
 Plans, colliery, 737.
 colouring of colliery, 739.
 scales for colliery, 738.
 Plant, boring, 25, 26.
 failures, electrical, 198, 199.
 Plotting by chords, 750, 751.
 by co-ordinates, 747-750.
 the levels, 759.
 the survey, 745-751.
 Plunger pump, 453, 465.

 Plunger pump, action of, 465.
 Pole star, observations of, 736.
 Position of shot-holes, 117, 118.
 of winding engines, 306.
 Potential, difference of, 204.
 Power, evaporative, 703, 704.
 transmission of, 175.
 cost of, 202.
 losses in, 208.
 Preparing wood for shafts, 65.
 Preservation of timber, 292, 293.
 Prevention of overwinding, 350.
 Primus washer, 722, 723.
 Problems, haulage, 421-424.
 in surveying, 759-765.
 Production of black-damp, 528
 Prop-drawer, 262.
 Properties of black-damp, 529.
 of gases, 516, 517, 518.
 Propping of faces, 212.
 Props, steel or iron, 276.
 tapered, 257.
 underset of, 259.
 Protector lock, 658, 659.
 Proto breathing apparatus, 670, 671,
 672.
 Protractors, 746, 747.
 Pulleys, bevel, 414, 415.
 haulage, 399, 403, 494, 495.
 roller, 414.
 Pulmotor, 691, 692.
 Pump bucket, 464.
 buckets, 453.
 centrifugal, 491.
 electric sinking, 505.
 fittings, 446.
 Kaselowsky, 487, 488.
 Moore's hydraulic, 486, 487.
 Oddie-Barclay, 483-486.
 pipes, 446, 447.
 Pulsometer, 489.
 Riedler, 479, 480.
 rods, 454-460.
 balancing the, 463.
 guiding the, 460.
 sizes of, 458, 459.
 sinking, 501-505.
 turbine, 491-501.
 valves, 451, 452.
 Pumping, 440-513.
 calculations on, 510-513.
 cost of, 509, 510.
 efficiency of electrical, 201,
 electrical, 200.
 engines, Cornish, 469.
 direct-acting, 468.
 duty of, 506, 507.
 geared, 468.

- Pumping engines, horizontal, 472, 473.
 special types of, 469.
 underground, 473-476.
 plant, choice of, 444.
- Pumps, arrangement of, 507, 508.
 plant required for, 468.
 shaft, 444.
 speed of, 467.
 underground, 445.
 water delivered by, 467, 468, 510, 513.
 working of, 445.
- Punch-props, 42, 43, 61.
- QUESTIONS, haulage, 421-424.
 pumping, 510-513.
- RACKINGS, shaft, 59.
- Rails, 370.
 friction of, 371.
- Rams, pump, 453.
- Rate of combustion, 705.
 of sinking, 66.
- Rateau fan, 573.
- Rectangular shaft, cost of lining, 65.
 lining of, 59, 60, 61.
 tools required, 65, 66.
 shafts, 41, 59, 60, 61.
 sinking, 59.
- Reducing air friction, 603.
- Relative density of gases, 515.
- Rescue apparatus, 666-698.
 corps, 693-698.
 station, 669.
- Reversing air-current, 576, 577, 578.
- Reviving apparatus, 687-693.
 Draeger, 691.
 Meco, 688, 690.
 Pulmotor, 691, 692.
- Revolving screens, 714.
- Rider, iron, 91, 92.
 wooden, 92, 93.
- Ridge lines, magnetic, 732, 733.
- Riedler pump, 479, 480, 483.
 valve, 185.
- Rigg & Meiklejohn machine, 138, 139.
- Ringer and chain, 262.
- Rings, skeleton, 44.
 water, 50, 51.
- Ritchie conveyor, 433-435.
- Roads, gradient of horse, 378, 379.
 haulage, 370, 371.
 horse, 377.
 timbering drawing, 263.
 width of, 211, 212.
- Robinson washer, 724.
- Rock drills, 123.
- Rocks, aqueous, 1, 2.
 cleavage of, 2.
 definition of, 1.
 division of, 1.
 igneous, 1, 2.
 metamorphic, 1, 2.
 stratified, 1.
 unstratified, 1.
- Rods, boring, 17, 18.
 pump, 454.
 sizes of pump, 458, 459.
- Roller pulleys, 414.
- Roofs, good and bad, 253.
 method of supporting, 256.
- Rooms, width of, 222.
- Rope, endless, 401-417.
 guides, 326, 327, 328.
 speed of endless, 405.
 taking up slack, 404.
 transmission, cost of, 189.
 horse-power of, 189, 190.
- Ropes, cappings for, 312-314.
 care of, 310.
 circular, 309.
 flat, 308.
 life of, 310.
 reducing strain on, 319, 320.
 strength of, 314, 315, 316.
 tension on, 311.
 weight of, 316.
 winding, 308-316.
- Rotary boring, 17, 28.
 tipplers, 708, 709, 710.
- Royalty, 39.
- Runaway blocks, 413.
- Running sand, 67, 68.
- Rutherford and Thomson clip, 410.
- SAFETY cages, 345.
 fuses, 113, 114.
 hooks, 342-345.
 lamps, 626-663.
 approved list of, 664, 665.
 construction of, 653.
 filling of, 663.
 lighting, 663.
 lighting power of, 653-655.
 locking of, 657, 658.
 testing of, 659-662.
 trimming of, 662.
 riders, 91, 92, 93.
- Sampling bottle, 551.
- Sand pack, 226, 227.
 running, 67, 68.
- Scales for colliery plans, 738.
- Schiele fan, 565.
- Screen, hurdle, 598.
- Screens, fixed inclined, 713.

- Screens, revolving, 714.
 - shaking, 714.
- Screws, adjusting, 346.
- Seam, Drumgray, 216.
 - Kiltongue, 229.
 - Main, 214.
 - Top Hard, 213.
 - Virgin, 229.
 - Warren, 215.
- Seams, contiguous, 227-232.
 - highly inclined, 234-245.
 - loss in working, 252.
 - order of working, 251.
 - working thick, 248.
- Seats for engines, 307.
- Secular variation, 732.
- Segment of tubbing, 53.
- Selection of a fan, 574.
- Self-acting inclines, 383-390.
- Ser fan, 572.
- Series wound dynamos, 197.
- Setting out meridian line, 736, 737.
- Shaft cables, 193, 194.
 - lining, 42, 45, 47, 59, 60, 61.
 - pumps, 444, 445.
- Shafts, cost of sinking, 58, 59.
 - covering of sinking, 62, 63, 64.
 - deepening, 96, 97.
 - enlarging, 93, 94, 95, 96.
 - form of, 41, 42.
 - number of, 39, 40.
 - rectangular, 59, 60, 61, 65.
 - site of, 40, 41.
 - size of, 39, 40.
 - ventilation of, 92, 93.
- Shamrock Colliery, 241.
- Shoes for cages, 329, 330.
- Shot-holes, position of, 117, 118.
- Shots, blown-out, 117.
 - firing of, 113, 115.
- Shunt wound dynamos, 198.
- Siding accommodation, 699.
- Sidings, inclination of, 699, 700.
- Simultaneous firing, 116, 117.
- Single-core cables, 192.
- Single stall working, 245.
- Sinking, 39.
 - and walling simultaneously, 48.
 - by brick drum, 68, 69.
 - by Chaudron method, 75, 76.
 - by freezing method, 84.
 - by Gobert's method, 85, 86, 87.
 - by Koch method, 86.
 - by Lippman method, 80.
 - by Pattberg method, 81, 82, 83.
 - by piles, 66, 67, 68.
 - by Poetsch method, 84, 85.
 - by Triger's method, 73.
- Sinking cylinder, 69, 70, 71.
 - frames, 62, 63.
 - kibble, 89, 90, 91.
 - pumps, 501-505.
 - rate of, 66.
 - scaffold, 46, 48.
 - special methods of, 66.
 - specification for, 97-100.
 - tank, 69, 70.
- Siphons, 441-444.
 - water delivered by, 442-444.
- Sirocco fan, 569, 570.
- Siskol drill, 129, 130.
 - machine, 157, 158.
- Size of packs, 212.
 - of pillars, 218-221.
- Sizes of pipes, 448, 449.
 - of pump rods, 458, 459.
 - of winding engines, 366.
- Skeleton rings, 44.
- Skid machine, 167.
- Slack rope, taking up, 404.
- Sliding collar, 27.
 - joint, 22.
- Slope, measuring on the, 744, 745.
- Sludger, 18.
- Smoke helmets, 686, 687.
- Spears, pump, 454.
- Special types of pumping engines, 469.
- Specific gravity of black-damp, 528.
 - of coal, 8, 15.
 - of gases, 515.
- Speed of air in upcast, 562, 563.
 - of cages, 332.
 - of engines, 304.
 - of fans, 585.
 - of pumps, 467.
- Spiral drum, 317.
- Splits, equal, 606.
 - unequal, 607.
- Splitting the air, 605-608.
- Spontaneous combustion, 232.
- Spray compressors, 182.
- Spring attachment, 321.
 - coupling, 321.
- Square work, 248.
- Stables, horse, 381.
- Stach electric lamp, 644, 645.
- Stall, double, 247.
- Stalls, width of, 222.
- Staple pits, 391.
- Stauss keps, 347.
- Steam, 175-178.
 - coal, 10.
 - exhaust, 177.
 - pumps, underground, 473-476.
- Steel frames, 300.
 - guides, 322.

- Steel props, 276.
 - pump pipes, 446.
 - ropes, 308.
 - sets, 276-282.
 - Stephenson lamp, 628.
 - Stepping the chain, 745.
 - Stoop and room, 218.
 - Storms, magnetic, 735.
 - Strain on ropes, 319.
 - Strata, dip of, 2.
 - inclination of, 2.
 - strike of, 2.
 - Stratification, 2.
 - Strength of boilers, 705.
 - of ropes, 314.
 - of timber, 287, 288.
 - Stress, electrostatic, 527.
 - Strike, 2.
 - Stringing deals, 42, 43.
 - Stroke, length of pump, 467.
 - Suction pipes, 448.
 - Sulphur, 519.
 - Sulphuretted hydrogen, 102, 539, 540.
 - Summit seam, working of, 244.
 - Sun, observations of, 736.
 - Supporting by walls and girders, 282.
 - haulage roads, 271.
 - highly inclined roads, 273.
 - wide spaces, 268.
 - working faces, 255.
 - Supports, pipe, 447.
 - Surface arrangements, 699-730.
 - Survey, plotting the, 745-751.
 - Surveying, 731-765.
 - boreholes, 37, 38.
 - definition of, 731.
 - problems, 759-765.
 - underground, 739-743.
 - Sylvester prop-drawer, 262.
 - Symbols of gases, 516.
 - Syncline, 3, 4.
 - System, panel, 225.
 - Systematic timbering, 259.
 - Systems of ventilation, 553, 554.
-
- TAIL-ROPE balance, 317.
 - Tanks, water, 440, 441.
 - Tapered props, 257.
 - Tapping wastes, 35.
 - Temperature, absolute, 516.
 - Temporary lining, 42, 43.
 - Tension on ropes, 311.
 - Ten-yard seam, 250.
 - Tesla Mines, California, 243.
 - Test for CO by blood solution, 539.
 - of fans, 588-592.
 - of Waddle fan, 590, 592.
 - Testing safety lamps, 659-662.
 - Tests for black-damp, 529-536.
 - Thermal units, 15.
 - Thermometer, 609.
 - Thick seams, working of, 248.
 - Thickness of pipes, 449, 450.
 - Thompson's calorimeter, 14.
 - Thomson conveyor, 435-437.
 - Three-core cables, 192.
 - Tiller, 19.
 - Timber, 255.
 - cost of, 275.
 - preserving, 293.
 - crushing strain of, 289.
 - preservation of, 292, 293.
 - strength of, 287, 288, 289.
 - withdrawing, 261.
 - Timbering, 253-294.
 - at machine face, 164.
 - Continental, 267.
 - Courrières system of, 260.
 - drawing roads, 263.
 - haulage roads, 271.
 - Lens system of, 260, 261.
 - necessity for, 253.
 - systematic, 259.
 - Welsh, 265.
 - Tipplers, 708-711.
 - rotary, 708, 710.
 - Tonge's wedge, 121, 122.
 - Tools, boring, 20, 21.
 - sinking, 65, 66.
 - Top Hard seam, 213.
 - Torbanite, 12.
 - Traction, horse, 377.
 - Transmission, hydraulic, 188, 189.
 - loss in, 176, 177.
 - of power, 175.
 - cost of, 202.
 - losses in, 208.
 - rope and rod, 189, 190.
 - Trap doors, 593, 594, 595, 596, 597.
 - Travelling belts, 710, 712.
 - Trepans, sinking, 75, 78.
 - Triangles, calculation by, 751, 752.
 - Triger's method of sinking, 73.
 - Trigonometrical levelling, 754.
 - Trimming safety lamps, 662.
 - Trough washer, 717, 718, 719.
 - True meridian, 732.
 - determination of, 736.
 - Tubbing, English, 54.
 - German, 54.
 - iron, 53-56.
 - segment of, 53.
 - thickness of, 54, 55.
 - Tube, grapnel, 23.
 - Haldane, 530, 531, 532, 533.

- Tubs, 371-375.
 attaching chain to, 401.
 Turbine pumps, 491-501.
 formulae for, 500, 501.
 high-lift, 497-500.
 principle of, 493, 494.
 UNDERGROUND cables, 194.
 fans, 586, 587, 588.
 measurements, 743, 744.
 steam pumps, 473-476.
 surveying, 739-743.
 Unequal splits, 607, 608.
 Useful effect of fan, 582.
 VALVE, leakage at air, 187.
 Riedler, 185.
 Valves for compressors, 185.
 pump, 451.
 Vapour pressure, 522-525.
 Variation of magnetic needle, 732, 733,
 734.
 Ventilating sinking shafts, 92, 93.
 Ventilation, 514-625.
 calculations, 623, 624, 625.
 furnace, 556-563.
 mechanical, 563-592.
 natural, 554, 555.
 systems of, 553, 554.
 waterfall, 556.
 Vernier compass, 741, 742.
 Virgin seam, 229.
 Visor, 351-356.
 Voltages, electrical, 196.
 WADDLE fan, 564, 565.
 Walker fan, 567.
 Walker's hook, 342.
 Walling, brick required for, 51.
 cost of, 51, 53.
 scaffold, 46.
 shaft, 45, 52.
 Walls, length of, 211.
 Ward and Lloyd clip, 411.
 Ward-Leonard winder, 363.
 Washer, Baum, 729, 730.
 choice of, 717.
 Greaves, 720, 721.
 Lührrig, 725-729.
 Murton, 720.
 Primus, 722, 723.
 Robinson, 724, 725.
 trough, 717-719.
 Washing coal, 714-730.
 Wastes, tapping, 35, 36.
 Water delivered by pumps, 467, 510,
 513.
 by siphon, 442.
 high pressure, 188.
 Water pack, 226, 227.
 rings, 50, 51.
 Waterfall ventilation, 556.
 Water-gauge, 611.
 Watering coal-dust, 617, 618, 619, 620.
 Wedge, Burnett, 120.
 Elliot, 120.
 hydraulic, 121.
 Tonge's, 121.
 Wedges, mechanical, 120.
 Wedging curb, 56.
 Weg breathing apparatus, 679-683.
 Weight of pipes, 451.
 Welsh timbering, 265.
 West and Darlington's balance, 464.
 West's hook, 343.
 Wet compressors, 182.
 Wheels, haulage, 399, 404, 405.
 White-damp, 536.
 Whitmore brake, 340, 341.
 overwind gear, 356, 357, 358.
 Wide spaces, supporting, 268.
 Width of roads, 211, 212.
 of stalls, 222.
 Winders, electric, 362-366.
 Winding coal, 295-369.
 drums, 337, 338.
 engines, 301-305.
 calculations for, 366-369.
 position of, 306.
 sizes of, 366.
 Koepe's system of, 319.
 ropes, reducing strain on, 319.
 Withdrawing timber, 261.
 Wolf electric lamp, 647, 648, 649.
 safety lamp, 631, 632.
 Wood & Burnett tippler, 708, 709.
 Wood chocks, 257.
 preparing for shafts, 65.
 weight of, 13.
 Work done by machines, 168.
 Working at Shamrock Colliery, 241.
 at Tesla Mines, 243.
 by single stall, 245.
 contiguous seams, 227-232.
 endless rope, 405.
 faces, supporting the, 255.
 haulage curves, 400.
 highly inclined seams, 234-245.
 of pumps, 445, 446.
 order of, 251.
 thick seams, 248.
 waste in, 252.
 Workings, laying out, 210.
 old, 35.
 Worthington pump, 476, 477.
 ZONES, dustless, 621.

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